Date: March 7, 2025

Ref: UKW-RPT-GUI-534

NI 43-101 Technical Report

Muntanga Uranium Project in the Southern Province of Zambia

For: GoviEx Uranium Inc.

Prepared by: Ukwazi Transaction Advisory (Pty) Ltd
SRK Consulting (UK) Ltd

GoviEx URANIUM

UKWAZI transaction advisory

www.ukwazi.com

Important Notice

This notice is an integral component of the NI 43-101 technical report ("TR") on the Muntanga Uranium Project in the Southern Province of Zambia (TR) and should be read in its entirety and must accompany every copy made of the TR. The TR has been prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects.

The TR has been prepared for GoviEx Uranium Inc. ("GoviEx") by Ukwazi Transaction Advisory (Pty) Ltd ("Ukwazi"), SRK Consulting (UK) Limited ("SRK"), SGS Bateman (Pty) Ltd ("SGS") and Cresco Global Ltd ("Cresco"). The TR is based on information and data supplied to Ukwazi, SRK, SGS, Cresco and other parties and where necessary the authors have assumed that the supplied data and information are accurate and complete.

The conclusions and estimates stated in the TR are to the accuracy stated in the TR only and rely on assumptions stated in the TR. The results of further work may indicate that the conclusions, estimates and assumptions in the TR need to be revised or reviewed.

Ukwazi and SRK have used their experience and industry expertise to produce the estimates and approximations in the TR. Where Ukwazi and SRK have made those estimates and approximations, it does not warrant the accuracy of those amounts, and it should also be noted that all estimates and approximations contained in the TR will be prone to fluctuations with time-and changing industry circumstances.

The TR should be construed in light of the methodology, procedures, and techniques used to prepare the TR. Sections or parts of the TR should not be read or removed from their original context.

The TR is intended to be used by GoviEx, subject to the terms and conditions of its contracts with Ukwazi, SRK, SGS and Cresco. Recognising that GoviEx has legal and regulatory obligations, Ukwazi, SRK, SGS and Cresco have consented to the filing of the TR with Canadian Securities Administrators and its System for Electronic Document Analysis and Retrieval ("SEDAR"). Except for the purposes legislated under provincial securities laws, any other use of this TR by any third party is at that party's sole risk.

Title page

Project name: Muntanga Uranium Project

Title: NI 43-101 Technical Report: Feasibility Study of the Muntanga Uranium Project, Zambia

Location: Southern Province, Republic of Zambia

Effective date of Technical Report: January 23, 2025

Effective date of Mineral Resource: January 31, 2024

Effective date of drilling database: December 31, 2024

Effective date of Mineral Reserve: January 1, 2025

Qualified Persons:

1. **Jacobus Johannes Lotheringen**, B Eng (Mining Engineering), South African Institute of Mining and Metallurgy (SAIMM) – Member (Reg no 701237) and Professional Engineer registered at the Engineering Council of South Africa (ECSA) (Reg no 20030022), employed by Ukwazi Transaction Advisory (Pty) Ltd as a principal mining engineer.

2. **André Marcel Deiss**, B.Sc.(Hons) Geology, registered at the South African Institute of Mining and Metallurgy (SAIMM) - Member Reg. no. 705005, the South African Council for Natural Scientific Professions (SACNASP) as Professional Natural Scientist (Geological Science) - Reg. no. 400007/97 and Engineers and Geoscientists of British Columbia as a Professional Geoscientist - P.Geo. Reg. no. 62356, employed by SRK Consulting (Canada) Inc. as an Associate Consultant (Resource Geology).

3. **Robert J Bowell**, BSc (Geochemistry), Hons, PhD (Geochemistry), Royal Society of Chemistry – Chartered Chemist (Member no 332782), Professional Geologist for the province of Newfoundland and Labrador. (Reg no 10809), Geological Society of London - Chartered Geologist (Reg no 1007245), Institute of Mining, Metallurgy and Materials – Fellow, employed by SRK UK as a geochemist.

4. **Alan Mitchell Clegg**, B.Sc. (Mining Engineering), Fellow of the SAIMM (Reg no 701825) and Professional Engineer registered at the ECSA (Reg no 20050117), employed as an independent valuation expert.

Certificate of Qualified Person

Jaco Lotheringen

Principal mining engineer and managing director
Ukwazi Transaction Advisory (Pty) Ltd ("Ukwazi").

Email: jaco@ukwazi.com

To accompany the report titled: **NI43-101 Technical Report: Muntanga Uranium Project in the Southern Province of Zambia**

I, Jaco J Lotheringen, a Principal Mining Engineer and registered Professional Engineer, hereby certify that:

1. I am responsible for the preparation of the technical report titled "NI 43-101 Technical Report: Muntanga Uranium Project in the Southern Province of Zambia" with effective date of 7 March 2025 (the "Technical Report") relating to GoviEx Uranium Inc's Muntanga Uranium Project (the "Project"). I was responsible for the chapters on Mineral Reserve Estimates (Section 15) and Mining Methods (Section 16). In addition, I was responsible for Sections 1 - 6, 18, 23, 24, 25 and 26.

2. I visited the Project site from 22 to 23 April 2024.

3. I am currently a technical advisor, principal mining engineer and managing director of the Ukwazi , with an office address of Building E (1st floor), Irene Link Office Park, 5 Impala Avenue, Doringkloof, Centurion, 0157, Gauteng, South Africa.

4. I have a Bachelor of Engineering degree (BEng. - Mining Engineering).

5. I am registered as a Professional Engineer with the Engineering Council of South Africa (1st Floor, Waterview Corner Building, Lake Office Park, 2 Ernest Oppenheimer Ave, Bruma, Johannesburg, 9301, South Africa), since 2003 with registration number 20030022.

6. I am also a member of the Southern African Institute of Mining and Metallurgy (7th Floor, Rosebank Towers, 19 Biermann Ave, Rosebank, Johannesburg, 2196, South Africa) with membership number 701 237.

7. I have been actively involved in the mining industry since 1997 with extensive experience in multiple mining methods and commodities. I have completed numerous Mineral/ Ore Reserve estimations and related public documentation based on South African, Australian and Canadian reporting requirements since 2010.

8. I have read the definition of "qualified person" set out in National Instrument 43-101 of the *Standards of Disclosure for Mineral Projects* ("NI43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI43-101.

9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

10. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, for which the omission to disclose would make the Technical Report misleading.

11. I am independent of GoviEx Uranium Inc. applying the test in Section 1.5 of the NI 43-101.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them or GoviEx Uranium Inc. for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public.

Signed in Centurion, South Africa on 7 March 2025.

("signed and sealed")

J.J. Lotheringen
Qualified Person: Mineral Reserve
Pr. Eng. (Mining)
MSAIMM 701237
ECSA 20030022

SRK Consulting (UK) Limited
5th Floor Churchill House
17 Churchill Way
Cardiff CF10 2HH
Wales, United Kingdom
E-mail: enquiries@srk.co.uk
URL: www.srk.com
Tel: + 44 (0) 2920 348 150

CERTIFICATE OF QUALIFIED PERSON

Robert John Bowell
Corporate Consultant (Geochemist)
SRK Consulting (UK) Ltd
Email: rbowell@srk.co.uk

To accompany the report titled: **NI 43-101 TECHNICAL REPORT: MUNTANGA URANIUM PROJECT IN THE SOUTHERN PROVINCE OF ZAMBIA**

I, Robert J Bowell, a Chartered Professional Chemist, Chartered Geologist and a Certified Professional European Geologist, do hereby certify that:

1. I am responsible for the preparation of the technical report titled "NI 43-101 Technical Report: Muntanga Uranium Project in the Southern Province of Zambia" and dated effective March 07, 2025 (the "Technical Report") relating to GoviEx Uranium Inc's Muntanga Uranium Project (the "Project"). In particular, Sections 1 to 6, 13 and 15 to 26.

2. I visited the Chirundu project from 3 to 7 May 2011 as part of a due diligence for a third party. During the visit, I observed drilling, core and drill chip library, sample preparation, and data collection. In addition, I visited the site for the Muntanga Project from 7 to 11 May 2022 to view all prospects in the FS and examine new core.

3. I am currently employed as a consulting geochemist to the mining and mineral exploration industry, as a Corporate Consultant Geochemist with SRK Consulting (UK) Ltd, with an office address of 5th Floor Churchill House, 17 Churchill Way, Cardiff, CF10 2HH, UK.

4. I graduated with a Bachelor of Science Degree, First Class Honours in Geochemistry from Owen's College, Manchester University, Manchester UK, June 1988.

5. I graduated with a Doctorate in Geochemistry from Southampton University, Southampton, UK in June 1991.

6. I am a Chartered Chemist of the Royal Society of Chemistry, London, UK and have been since 1997. Membership number 332782.

7. I am a chartered Professional Geologist for the province of Newfoundland and Labrador. Registration number 10809.

8. I am a Chartered Geologist and Certified Professional European Geologist through the Geological Society of London since 1997 and European Association of Professional Geologists since 2000. Registration number 1007245.

9. I am a Fellow of the Institute of Mining, Metallurgy and Materials and have been since 2010.

10. I have been employed as a geochemist in the mining and mineral exploration business and in applied academia, for the past 34 years, since my graduation from university.

11. I have read the definition of "qualified person" set out in National Instrument 43-101 of the *Standards of Disclosure for Mineral Projects* (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101. The Technical Report is based upon my personal review of the information provided by the Issuer. My relevant experience for the purpose of the Technical Report is:

    - Geochemist, SRK Consulting from 1995 to date;
    - Exploration Geochemist with BHP Minerals, Hammersmith, London, 1991-1994;
    - Exploration Geologist, Ashanti Goldfields, Ghana, 1988
    - Uranium exploration experience as a geochemist and geometallurgical consultant, from 1998-1999; 2005-2006; 2007-current
    - Experience in the above positions working with and reviewing uranium mineralogy and geology, uranium analysis, resource estimation methodologies, geometallurgical testwork for uranium, uranium metallurgy, geochemical data quality, assurance and quality control in concert with resource estimation geologists and engineers.
    - As a consultant, I have been involved in several previous competent person's reports for uranium projects including NI 43-101 technical reports, 2002; 2006-08, 2009, 2010, 2013-15, 2017-18, 2021.

12. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

13. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, for which the omission to disclose would make the Technical Report misleading.

14. I am independent of GoviEx Uranium Inc. applying the test in section 1.5 of NI 43-101.

15. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them or GoviEx Uranium Inc. for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public.

Dated in Cardiff, United Kingdom, March 07, 2025

This signature has been scanned. The author has given permission to its use for this particular document. The original signature is held on file.

SRK CONSULTING (UK) LIMITED Co. No: 01575403

("signed") ("sealed")

***Eur.Geol.* Robert Bowell PhD C.Chem. C.Geol**

Corporate Consultant (Geochemist)

#332782, Chemist; #1007245, Geologist; 10809 PEGNFL

CERTIFICATE OF QUALIFIED PERSON

Andre Marcel Deiss

Associate Consultant (Resource Geology)

SRK Consulting (Canada) Inc.

Email: adeiss@srk.co.uk

To accompany the report titled: **NI 43-101 TECHNICAL REPORT: MUNTANGA URANIUM PROJECT IN THE SOUTHERN PROVINCE OF ZAMBIA**

I, Andre M. Deiss, a registered Professional Natural Scientist, Geological Science (Pr.Sci.Nat.) and Professional Geoscientist (P.Geo.), do hereby certify that:

1. I am responsible for the preparation of the technical report titled "NI 43-101 Technical Report: Muntanga Uranium Project in the Southern Province of Zambia" and dated effective March 07, 2025 (the "Technical Report") relating to GoviEx Uranium Inc's Muntanga Uranium Project (the "Project"). In particular, Sections 1, 7 to 12, 14, and 25 to 26 related to geology and mineral resources.

2. I did not visit the Project site but worked on the Muntanga Project geology and mineral resources with the previous QP, Mr. Revering until early 2024 until Mr. Revering was no longer in SRK's employ when I took over become resource QP to ensure continuation and consistency in the technical work.

3. I am a resource geologist consulting to the mining and mineral exploration industry and employed as as an Associate Consultant (Resource Geology) with SRK Consulting (Canada) Inc. located at 2600–320 Granville St., Vancouver, BC V6C 1S9, Canada.

4. I graduated with a Bachelor of Science Degree, First Class Honours in Geology from the University of the Witwatersrand, Johannesburg, South Africa in May 1994.

5. I am a Professional Natural Scientist (Geological Science) registered with the South African Council for Natural Scientific Professions (SACNASP), South Africa and in good standing since 1997 (Registration no. 400007/97).

6. I am a Professional Geoscientist (P.Geo.) registered with the Engineers and Geoscientists of British Columbia, Canada (Registration number 62356).

7. I am a member of the South African Institute of Mining and Metallurgy since 2010 (Registration no. 705005)

8. I have been employed as a geologist in the mining and mineral exploration operationally and in consulting, for the past 30 years since my graduation from university.

9. I have practiced my profession for 31 years in the mining industry related to exploration, mine operations and project evaluations, with a specialization in geological modelling, mineral resource estimation, production reconciliation, grade control, exploration and production geology. Specific to uranium, I have worked on numerous uranium projects located in Africa, North America and South America.

10. I have read the definition of "qualified person" set out in National Instrument 43-101 of the *Standards of Disclosure for Mineral Projects* (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

12. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, for which the omission to disclose would make the Technical Report misleading.

13. I am independent of GoviEx Uranium Inc. applying the test in section 1.5 of NI 43-101.

14. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them or GoviEx Uranium Inc. for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public.

Dated in Vancouver, Canada, March 07, 2025

PROFESSIONAL
PROVINCE OF
A. M. DEISS
62356
BRITISH COLUMBIA
GEOSCIENTIST

Andre Marcel Deiss, B.Sc. (Hons)
Associate Consultant (Resource Geology)
Pri.Sci.Nat (400007/97**), P.Geo. (**62356**)**

CERTIFICATE OF QUALIFIED PERSON

Alan Mitchell Clegg

Independent Specialist Advisor, Consultant (Mining) & Competent Valuator (CV)

Ukwazi Transaction Advisory (Pty) Ltd

Email: alan@ukwazi.com

To accompany the report titled: **NI 43-101 TECHNICAL REPORT: MUNTANGA URANIUM PROJECT IN THE SOUTHERN PROVINCE OF ZAMBIA**

I, Alan Mitchell Clegg, a Registered Professional Engineer (Pr.Eng), Certified Project Management Professional (PMP), Registered Professional Construction Project Manager (Pr.CPM), and Competent Valuator (CV) do hereby certify that:

1. I am responsible for the preparation of the technical report titled "NI 43-101 Technical Report: Muntanga Uranium Project in the Southern Province of Zambia" and dated effective March 07, 2025 (the "Technical Report") relating to GoviEx Uranium Inc's Muntanga Uranium Project (the "Project"). In particular, Sections 1 to 6, 13 and 15 to 19 and 21 to 26.

2. I have not visited the project site but am familiar and have over 25 years' experience with the type of mineralisation, extraction methods and processing of the targeted commodity, i.e. Uranium.

3. I am currently employed as an Independent Specialist Advisor, Consultant (Mining) & Competent Valuator (CV), with Ukwazi Transaction Advisory (Pty) Ltd with office address of Building E (1st floor), Irene Link Office Park, 5 Impala Avenue, Doringkloof, Centurion, 0157, Private Bag X159, Centurion, 0046.

4. I graduated with a Bachelor of Science Degree in Mining Engineering from University of Newcastle Upon Tyne, Newcastle, UK, July 1982.

5. I am a Registered Professional Engineer (Mining) (Pr.Eng) with the Engineering Council of South Africa (ECSA). Membership number 20050117.

6. I am a Registered Professional Construction Project Manager (Pr.CPM) with the South African Council for Project & Construction Management Professions (SACPCMP). Registration number D/999/2006.

7. I am a Certified Project Management Professional (PMP) with the Project Management Institute (RSA & USA), since 2005. Registration Number 571190.

8. I am a Fellow of the Institute of Mining & Metallurgy (FSAIMM), No. 701825, since 1998.

9. I am a Fellow of the Institute of Quarrying (RSA & UK) (FIOQ), No. OC35, since 2000.

10. I am a Fellow of the Institute of Directors (RSA & UK) (FIOD), No. 17278095, since 2016.

11. I have been working in the Mining sector since June 1974 and employed as a Qualified Mining Engineer in the mining and energy business for the past 43 years, since my graduation from university.

12. I have read the definition of "qualified person" set out in National Instrument 43-101 of the *Standards of Disclosure for Mineral Projects* (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101. The Technical Report is based upon my personal review of the information provided by the Issuer. My relevant experience for the purpose of the Technical Report is:

    - Goldfields & Anglo-American Gold & Uranium Division from 1978 to 1988 operating Uranium Mining;
    - Director Mining Bateman Industrial Ltd – Uranium Mining & Processing Equipment specification 1989 to 1993
    - Managing Director & Chief Consulting Engineer Tamrock Africa – Uranium Mining Equipment & Mining Systems Specification 1994 to 1999
    - Director & Consultant Engineer Proudfoot Consulting, Uranium Mines productivity & systems engineering 1999-2000;
    - Director Mining Engineering TWP Consulting Group, Uranium Projects design 2001 to 2008
    - Executive Chairman, Director & Consulting Engineer Afrasia Mining & Energy Consulting, Systems design, operations & process consulting for Kazatomprom Kazakhstan & Kyrgyzstan ISR Uranium mines, Exploration & Mines development for Uranium One, and others, 2008 to 2013
    - Non-Executive Technical Director for Samruk Kaznya Invest (SKI) consulting on Uranium Investments and mining asset valuations, 2013 to 2018
    - Experience in the above positions working with and reviewing all aspects of the Uranium mining, processing and associated value chain in concert with other discipline professionals.
    - As an independent specialist advisor and consultant to Ukwazi, I have been involved in several competent person's reports for uranium projects including NI 43-101 technical reports, since 2014 to date.

13. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

14. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, for which the omission to disclose would make the Technical Report misleading.

15. I am independent of GoviEx Uranium Inc. applying the test in section 1.5 of NI 43-101.

16. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them or GoviEx Uranium Inc. for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public.

Dated in Centurion, Gauteng, South Africa, March 07, 2025

("signed") ("sealed")

ALAN MITCHELL CLEGG Pr.Eng Pr.CPM PMP FSAIMM FIOQ

Independent Specialist Advisor (Mining) & Competent Valuator (CV)

Engineer #20050117 Valuator #701825 Project Professional #571190 Construction Manager #D/999/2206

Consent of Qualified Person

TO: British Columbia Securities Commission
Alberta Securities Commission
Ontario Securities Commission
Toronto Stock Exchange Venture Exchange

AND TO: GoviEx Uranium Inc.

Dear Sirs/Mesdames,

RE: GoviEx Uranium Inc. (the "Company")

I, the undersigned, am an author of the Technical Report prepared in accordance with the National Instrument 43-101 – Standards of Disclosure for Mineral Projects titled "NI43-101 Technical Report: Muntanga Uranium Project in the Southern Province of Zambia" with effective date of 7 March 2025 (the "Report"), which supports the disclosure in the Company's news release dated 23 January 2025 (the "News Release").

I hereby consent to the public filing of the Report, and the use of extracts from, or a summary of, the Report in the News Release.

I hereby confirm that I have read the News Release and that the News Release fairly and accurately represents the information in the sections of the Report for which I am responsible.

Dated: 7 March 2025

Yours truly,

("signed and sealed")

J.J. Lotheringen
Qualified Person: Mineral Reserve
Pr. Eng. (Mining)
SAIMM 701237
ECSA 20030022

CONSENT OF QUALIFIED PERSON

TO: British Columbia Securities Commission

Alberta Securities Commission

Ontario Securities Commission

TSX Venture Exchange

AND TO: GoviEx Uranium Inc.

Dear Sirs/Mesdames,

RE: GOVIEX URANIUM INC. (THE "COMPANY")

I, the undersigned, am an author of the technical report prepared in accordance with National Instrument 43-101 – Standards of Disclosure for Mineral Projects titled "NI 43-101 Technical Report: Muntanga Uranium Project in the Southern Province of Zambia" and dated effective March 07, 2025 (the "Report"), which supports the disclosure in the Company's news release dated January 23, 2025 (the "News Release").

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I hereby confirm that I have read the News Release and that the News Release fairly and accurately represents the information in the sections of the Report for which I am responsible.

Dated: March 07, 2025

Yours faithfully,

This signature has been scanned. The author has given permission to its use for this particular document. The original signature is held on file.

SRK CONSULTING (UK) LIMITED Co. No: 01575403

("signed") ("sealed")

Eur.Geol. Robert Bowell PhD C.Chem. C.Geol

Corporate Consultant (Geochemist)

#332782, Chemist; #1007245, Geologist; 10809 PEGNFL

CONSENT OF QUALIFIED PERSON

TO: British Columbia Securities Commission
Alberta Securities Commission
Ontario Securities Commission
TSX Venture Exchange

AND TO: GoviEx Uranium Inc.

Dear Sirs/Mesdames,

RE: GoviEx Uranium Inc. (the "Company")

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I hereby confirm that I have read the News Release and that the News Release fairly and accurately represents the information in the sections of the Report for which I am responsible.

Dated: March 07, 2025

Yours truly,

PROFESSIONAL GEOSCIENTIST
PROVINCE OF BRITISH COLUMBIA
A. M. DEISS
62356

Andre Marcel Deiss, B.Sc. (Hons)
Associate Consultant (Resource Geology)
Pri.Sci.Nat (400007/97), **P.Geo.** (62356)

CONSENT OF QUALIFIED PERSON

TO: British Columbia Securities Commission
Alberta Securities Commission
Ontario Securities Commission
TSX Venture Exchange

AND TO: GoviEx Uranium Inc.

Dear Sirs/Mesdames,

RE: GoviEx Uranium Inc. (the "Company")

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Dated: March 07, 2025

Yours truly,

("signed") ("sealed")

ALAN MITCHELL CLEGG Pr.Eng Pr.CPM PMP FSAIMM FIOQ
Independent Specialist Advisor (Mining) & Competent Valuator (CV)
Engineer #20050117 Valuator #701825 Project Professional #571190 Construction Manager #D/999/2206

List of abbreviations

Abbreviation/ Unit	Unit or term
t	Tonnes
%	Percent
€	Euro
°C	Degrees Celsius
µ	Micron or microns
4G	Fourth generation wireless
4WD	Four-wheel drive vehicle
AA	Atomic absorption
AACEI	Advancement of Cost Engineering International
AC	Air core
AFR	African Energy Resources Ltd
AGIP	AGIP SpA
BOQ	Bill of quantity
CBA	Commercial bid adjudications
CCD	Counter-current decantation
cfm	Cubic feet per minute
CIL	Carbon-in-leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIX	Continuous ion exchange circuit
cm	Centimetre
cm^2	Square centimetre
cm^3	Cubic centimetre
COG	Cut-off grade
ConfC	Confidence code
CPP	Central processing plant
CPS	Counts per second
Crec	Core recovery
CSR	Corporate social responsibility
CSS	Closed-side setting
CTW	Calculated true width
CV	Coefficient of variation
DDH	Diamond drilling, diamond core drilling
DGPS	Differential global positioning system
dia.	Diameter
DTH	Down-the-hole hammer
E&I	Electrical and instrumentation
EIA	Environmental impact assessment
EIS	Environmental Impact Statement
EM	Electromagnetic
EMP	Environmental Management Plan
EPF	Environmental Protection Fund
ESG	Environmental, Social and Governance
ESIA	Environmental and Social Impact Assessment
eU	Equivalent uranium assay value; determined radiometrically
eU_3O_8	Equivalent U_3O_8; determined radiometrically
FEL	Front-end loader
FIFO	First in first out

Abbreviation/ Unit	Unit or term
FS	Feasibility study
ft	Foot (feet)
ft^2	Square foot (feet)
ft^3	Cubic foot (feet)
g	Gram
G&A	General and administration project costs
gal	Gallon
g-mol	Gram-mole
GoviEx	GoviEx Uranium Inc.
GHG	Greenhouse gas
gpm	Gallons per minute
GPS	Global positioning system
g/t	Grams per tonne
GSZ	Geological Survey of Zambia
GT	Grade thickness
GWe	Giga Watts electrical
ha	Hectares
HAZOP	Hazard and operability study
HDPE	High Density Polyethylene
hPa	Hectopascal
HPS	High pressure sodium
IAEA	International Atomic Energy Agency
ICP	Induced couple plasma
ID	Inverse distance
ID^2	Inverse-distance squared
ID^3	Inverse-distance cubed
IFC	International finance corporation
ILS	Intermediate leach solution
IX	Ion exchange
JORC	Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Mineral Council of Australia
K	Potassium
kg	Kilograms
kg/m^3	Kilograms per cubic metre
kg/t eU	Kilograms per tonne of equivalent uranium metal
km	Kilometre
km^2	Square kilometre
koz	Thousand troy ounce
kt	Thousand tonnes
ktpa	Kilotonnes per annum
ktpd	Thousand tonnes per day
ktpy	Thousand tonnes per year
kV	Kilovolt
kW	Kilowatt
kWh	Kilowatt-hour
kWh/t	Kilowatt-hour per metric tonne
L	Litre
lb	Pound
LLDPP	Linear low density polyethylene plastic
LOI	Loss on ignition
LOM	Life of mine

Abbreviation/ Unit	Unit or term
Lps	Litres per second
LVTC	Lusaka Vocational Technical College
m	Metre
M lcm	Million loose cubic metres
m.y.	Million years
m/month	Metres per month
m^2	Square metre
m^3	Cubic metre
masl	Metres above sea level
MDA	Mine Development Associates
MEL	Mechanical Equipment List
mg/l	Milligrams/litre
Mlb	Million pounds
mm	Millimetre
mm^2	Square millimetre
mm^3	Cubic millimetre
MME	Mine & Mill Engineering
MMMD	Ministry of Mines and Minerals Development of Zambia
MRE	Mineral Resource Estimate
Mt	Million tonnes
MTO	Material take off
Mtpa	Million tonnes per annum
MTW	Measured true width
m_{vert}/m_{hor}	Vertical metres per horizontal metre
MWe	Mega Watts electrical
NGO	non-governmental organization
NI 43-101	Canadian Securities Administrators National Instrument 43-101 - Standards of Disclosure for Mineral Projects
OK	Ordinary kriging
oz	Troy ounce
P&ID	Piping and instrumentation diagram
PEA	Preliminary economic assessment
PFD	Process flow diagram
PFS	Pre-feasibility study
PLC	Programmable logic controller
PLS	Pregnant liquor solution
PMF	Probable maximum flood
ppm	Parts per million
QAQC	Quality assurance/ quality control
QP	Qualified person
Ra-Grade	Radiometric-Grade
RC	Reverse circulation
RES	Remote Exploration Services
RL	Reduced level
RO	Reverse osmosis
ROM	Run-of-Mine
RPEEE	Reasonable prospects for eventual economic extraction
s	Second
SANS	South African National Standards
SCADA	Supervisory control and data acquisition
SDDR	Supplier Document and Drawing Requirements

Abbreviation/ Unit	Unit or term
SG	specific gravity
SI	Statutory Instruments
SMPP	Structural, Mechanical, Platework and Piping
SOD	Spent ore dump
SOW	Scope of work
SP	Self-potential
SPR	Single point resistance
SWSCO	Southern Water and Sewerage Company
t	Metric tonne
t eU	Tonnes of equivalent uranium metal
t/doh	Tonnes per direct operating hour
TBE	Technical Bid Evaluations
The Project	Muntanga Uranium Project
TMI	Total magnetic intensity
tpa	Tonnes per annum
tpd	Tonnes per day
tph	Tonnes per hour
tpy	Tonnes per year
t_{waste}:t_{RoM}	Tonnes of waste per tonne of run-of-mine
U	Uranium
U_3O_8	Uranium expressed as an oxide (triuranium octoxide)
USD	The United States dollar
USD/kg	USD per kilogram
USD/kg U	USD per kilogram of equivalent uranium
USD/lb U_3O_8	USD per pound of U_3O_8
USD/t	USD per tonne
USD/t_{metal}	USD per tonne of uranium metal
USD/t_{ROM}	USD per tonne of run-of-mine
USDA	United States Department of Agriculture
USD thousand	Thousand USD
USD million	Million USD
W	Watt
WD	Wagon drill (“WD”).
WRD	Waste rock dump
WRMA	Water Resources Management Authority
XRD	X-ray diffraction
XRF	X-ray fluorescence
yr	Year
ZEMA	Zambian Environmental Management Agency
ZESCO	Zambian Electricity Supply Corporation

Table of general terms and definitions

Glossary	Definition
Acceptable foreign code	means the JORC Code, the PERC Code, the SAMREC Code, SEC Industry Guide 7, the Certification Code, or any other code, generally accepted in a foreign jurisdiction, that defines mineral resources and mineral reserves in a manner that is consistent with mineral resource and mineral reserve definitions and categories set out in sections 1.2 and 1.3 of the instrument (NI43-101)
Adjacent property	means a property: a. in which the issuer does not have an interest b. that has a boundary reasonably proximate to the property being reported on and c. that has geological characteristics similar to those of the property being reported on
Advanced property	means a property that has: a. mineral reserves, or b. mineral resources the potential economic viability of which is supported by a preliminary economic assessment, a pre-feasibility study or a feasibility study
Certification code	means the Certification Code for Exploration Prospects, Mineral Resources and Ore Reserves prepared by the Mineral Resources Committee of the Institution of Mining Engineers of Chile, as amended.
CIM	means the Canadian Institute Mining's CIM Definition Standards on Mineral Resources and Reserves, which establish definitions and guidance on the definitions for Mineral Resources, Mineral Reserves, and mining studies used in Canada.
Data verification	means the process of confirming that data has been generated with proper procedures, has been accurately transcribed from the original source and is suitable to be used.
Disclosure	means any oral statement or written disclosure made by or on behalf of an issuer and intended to be, or reasonably likely to be, made available to the public in a jurisdiction of Canada, whether or not filed under securities legislation, but does not include written disclosure that is made available to the public only by reason of having been filed with a government or agency of government pursuant to a requirement of law other than securities legislation.
Early-stage exploration property	means a property for which the technical report being filed has: a. no current mineral resources or mineral reserves defined and b. no drilling or trenching proposed.
Effective date	means, with reference to a technical report, the date of the most recent scientific or technical information included in the technical report.
Exploration information	means geological, geophysical, geochemical, sampling, drilling, trenching, analytical testing, assaying, mineralogical, metallurgical, and other similar information concerning a particular property that is derived from activities undertaken to locate, investigate, define, or delineate a mineral prospect or mineral deposit.
Historical estimate	means an estimate of the quantity, grade, or metal or mineral content of a deposit that an issuer has not verified as a current mineral resource or mineral reserve, and which was prepared before the issuer acquiring, or entering into an agreement to acquire, an interest in the property that contains the deposit.
JORC Code	means the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves prepared by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Minerals Council of Australia, as amended.
Mineral project	means any exploration, development or production activity, including a royalty or similar interest in these activities, in respect of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals.
NI 43-101	Means Canadian National Instrument 43-101, which is a Canadian securities regulation that governs how companies disclose information about mining and mineral exploration. It applies to companies that trade on Canadian stock exchanges or over-the-counter markets.
PERC Code	means the Pan-European Code for Reporting of Exploration Results, Mineral Resources and Reserves prepared by the Pan-European Reserves and Resources Reporting Committee, as amended.
Preliminary economic assessment	means a study, other than a pre-feasibility or feasibility study, that includes an economic analysis of the potential viability of mineral resources.
Producing issuer	means an issuer with annual audited financial statements that disclose: a. gross revenue, derived from mining operations, of at least $30 million Canadian for the issuer's most recently completed financial year and b. gross revenue, derived from mining operations, of at least $90 million Canadian in the aggregate for the issuer's three most recently completed financial years.
Professional association	means a self-regulatory organization of engineers, geoscientists or both engineers and geoscientists that: a. is i. given authority or recognition by statute in a jurisdiction of Canada, or ii. a foreign association that is generally accepted within the international mining community as a reputable professional association b. admits individuals on the basis of their academic qualifications, experience, and ethical fitness

Glossary	Definition
	c. requires compliance with the professional standards of competence and ethics established by the organization d. requires or encourages continuing professional development and e. has and applies disciplinary powers, including the power to suspend or expel a member regardless of where the member practises or resides
Qualified person	means an individual who: a. is an engineer or geoscientist with a university degree, or equivalent accreditation, in an area of geoscience, or engineering, relating to mineral exploration or mining b. has at least five years of experience in mineral exploration, mine development or operation, or mineral project assessment, or any combination of these, that is relevant to his or her professional degree or area of practice c. has experience relevant to the subject matter of the mineral project and the technical report d. is in good standing with a professional association and e. in the case of a professional association in a foreign jurisdiction, has a membership designation that: i. requires attainment of a position of responsibility in their profession that requires the exercise of independent judgment and ii. requires: A. a favourable confidential peer evaluation of the individual's character, professional judgement, experience, and ethical fitness or B. a recommendation for membership by at least two peers and demonstrated prominence or expertise in the field of mineral exploration or mining.
Quantity	means either tonnage or volume, depending on which term is the standard in the mining industry for the type of mineral.
SAMREC Code	means the South African Code for the Reporting of Exploration Results, Mineral Resources and Mineral Reserves prepared by the South African Mineral Resource Committee (SAMREC) under the Joint Auspices of the Southern African Institute of Mining and Metallurgy and the Geological Society of South Africa, as amended.
SEC Industry Guide 7	means the mining industry guide entitled "Description of Property by Issuers Engaged or to be Engaged in Significant Mining Operations" contained in the Securities Act Industry Guides published by the United States Securities and Exchange Commission, as amended.
Specified exchange	means the Toronto Stock Exchange, the Australian Stock Exchange, the Johannesburg Stock Exchange, the London Stock Exchange Main Market, the Nasdaq Stock Market, the New York Stock Exchange, or the Hong Kong Stock Exchange.
Technical report	means a report prepared and filed in accordance with this Instrument and Form 43-101F1 Technical Report that includes, in summary form, all material scientific and technical information in respect of the subject property as of the effective date of the technical report.
Written disclosure	includes any writing, picture, map, or other printed representation whether produced, stored or disseminated on paper or electronically, including websites.
Mining studies	In this Instrument, the terms "preliminary feasibility study", "pre-feasibility study" and "feasibility study" have the meanings ascribed to those terms by the Canadian Institute of Mining, Metallurgy and Petroleum, as the CIM Definition Standards on Mineral Resources and Mineral Reserves adopted by CIM Council, as amended.
Independence	In this Instrument, a qualified person is independent of an issuer if there is no circumstance that, in the opinion of a reasonable person aware of all relevant facts, could interfere with the qualified person's judgment regarding the preparation of the technical report.
Pre-Feasibility Study (Preliminary Feasibility Study)	The CIM Definition Standards require the completion of a Pre-Feasibility Study as the minimum prerequisite for the conversion of Mineral Resources to Mineral Reserves. A Pre-Feasibility Study is a comprehensive study of a range of options for the technical and economic viability of a mineral project that has advanced to a stage where a preferred mining method, in the case of underground mining, or the pit configuration, in the case of an open pit, is established and an effective method of mineral processing is determined. It includes a financial analysis based on reasonable assumptions on the Modifying Factors and the evaluation of any other relevant factors which are sufficient for a Qualified Person, acting reasonably, to determine if all or part of the Mineral Resource may be converted to a Mineral Reserve at the time of reporting. A Pre-Feasibility Study is at a lower confidence level than a Feasibility Study.
Feasibility study	A Feasibility Study is a comprehensive technical and economic study of the selected development option for a mineral project that includes appropriately detailed assessments of applicable Modifying Factors together with any other relevant operational factors and detailed financial analysis that are necessary to demonstrate, at the time of reporting, that extraction is reasonably justified (economically mineable). The results of the study may reasonably serve as the basis for a final decision by a proponent or financial institution to proceed with, or finance, the development of the project. The confidence level of the study will be higher than that of a Pre-Feasibility Study.
Mineral Resource	A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.

Glossary	Definition
	The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.
Inferred Mineral Resource	An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
Indicated Mineral Resource	An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.
Measured Mineral Resource	A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.
Modifying factors	Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.
Mineral Reserve	A Mineral Reserve is the economically mineable part of a measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at pre-feasibility or feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified. The reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported. The public disclosure of a Mineral Reserve must be demonstrated by a Pre-Feasibility Study or Feasibility Study.
Probable Mineral Reserve	A Probable Mineral Reserve is the economically mineable part of an indicated, and in some circumstances, a Measured Mineral Resource. The confidence in the Modifying Factors applying to a Probable Mineral Reserve is lower than that applying to a Proven Mineral Reserve.
Proven Mineral Reserve	A Proven Mineral Reserve is the economically mineable part of a Measured Mineral Resource. A Proven Mineral Reserve implies a high degree of confidence in the Modifying Factors.

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1. Summary

1.1. Introduction

The Muntanga Uranium Project ("the Project") is located in the Siavonga and Chirundu Districts in the southeastern region of Zambia. The Project is controlled 100 % by GoviEx Uranium Zambia Limited, which is ultimately 100% owned and controlled by the Toronto Stock Exchange-listed exploration and development company, GoviEx Uranium Inc. ("GoviEx").

After the release on 30 November 2017 of a NI 43-101 TR on a preliminary economic assessment ("PEA") for the Project, GoviEx conducted a further drilling programme to increase the Mineral Resource at the Project and improve the classification of the Mineral Resources. Knowledge of the geology of the orebodies increased to a point where a full feasibility study ("FS") could be carried out, enabling a Mineral Reserve to be declared and a updated NI 43-101 Technical Report issued.

GoviEx appointed Ukwazi Transaction Advisory (Pty) Ltd ("Ukwazi"), SRK Consulting (UK) Limited ("SRK"), SGS Bateman (Pty) Ltd ("SGS") and Cresco Global Ltd ("Cresco") to complete technical studies to a feasibility level of confidence for the Muntanga open pit ("OP") project, process plant and associated infrastructure. This report has been prepared in accordance with the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1, collectively referred to as "NI 43-101".

1.2. Reliance on other experts

The qualified persons for this TR, Jaco Lotheringen, Robert Bowell, André Deiss and Alan Clegg, have examined the historical and current data for the Project provided by GoviEx with respect to Mineral Resources, metallurgical test work, and other project information, and have relied upon that data to support the statements and opinions presented in this report. Several other technical specialists, including GoviEx staff members, are also contributors of information in sections of this report. These contributions have been supervised and reviewed by the qualified persons and the qualified persons have taken reasonable measures to confirm the information provided by others.

1.3. Property description and ownership

GoviEx holds sole ownership of several mining and exploration licences for uranium deposits in the Siavonga and Chirundu Districts in the southeastern region of Zambia, geographically centred at 16°22'03.31"S, 28°28'51.3"E (shown in Figure 2-1). These are collectively known as the Muntanga Uranium Project ("Muntanga", or "the Project"). The Project comprises three mining licences: Muntanga, Dibbwi and Chirundu, and three exploration licences: Chirundu Extension, Nabbanda and Kariba Valley (Chisebuka), all shown in Figure 2-2. The Muntanga and Dibbwi mining licences comprise the Muntanga, Dibbwi and Dibbwi East deposits. The Chirundu mining licence contains the Njame and Gwabi deposits. There are no agreements or encumbrances on the permits currently held by GoviEx or its subsidiaries.

The northern extent of the Project, where the Gwabi and Njame deposits are situated, is located close to the town of Chirundu, near the Zimbabwe border. The prospect areas extend south towards Siavonga and along the northern edge of Lake Kariba to Kariba Valley in the southernmost extent. The northernmost deposits of Njame and Gwabi are located approximately 100 km southeast of the Zambian capital, Lusaka. Chisebuka, further south, is approximately 180 km south of Lusaka.

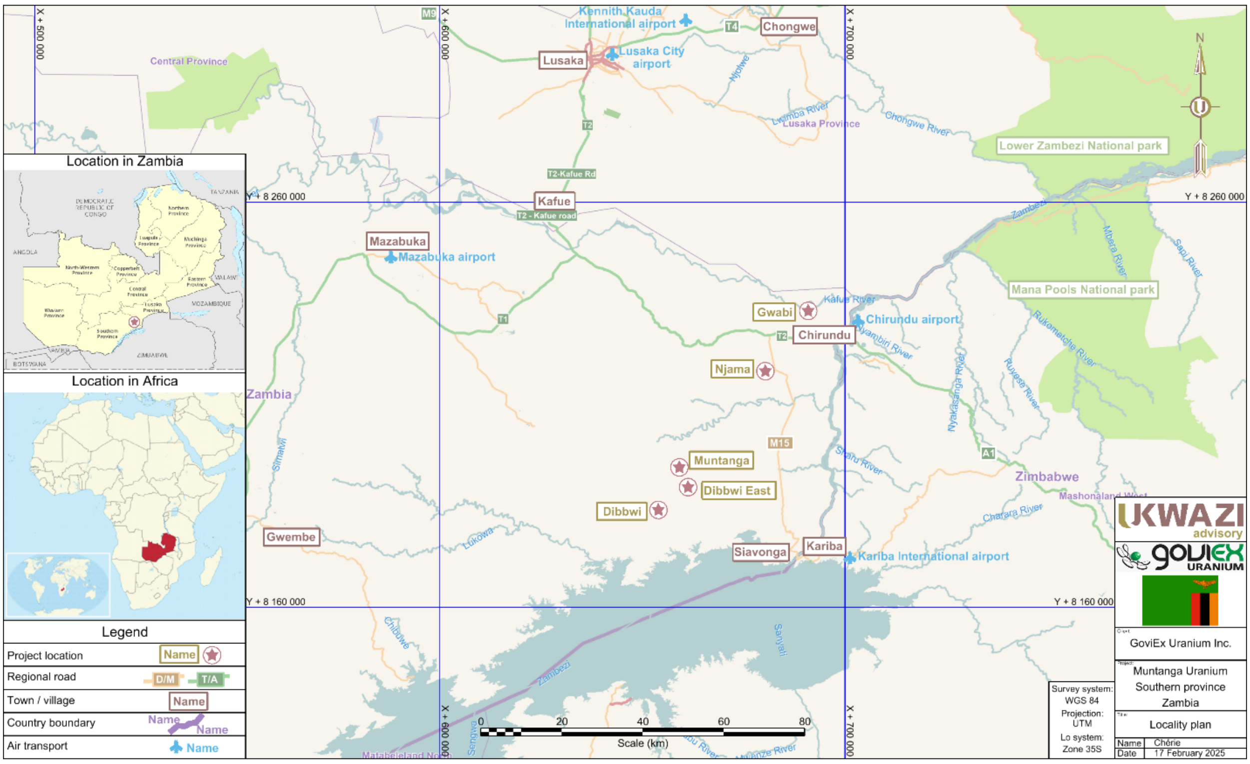

Figure 2-1: Property location map

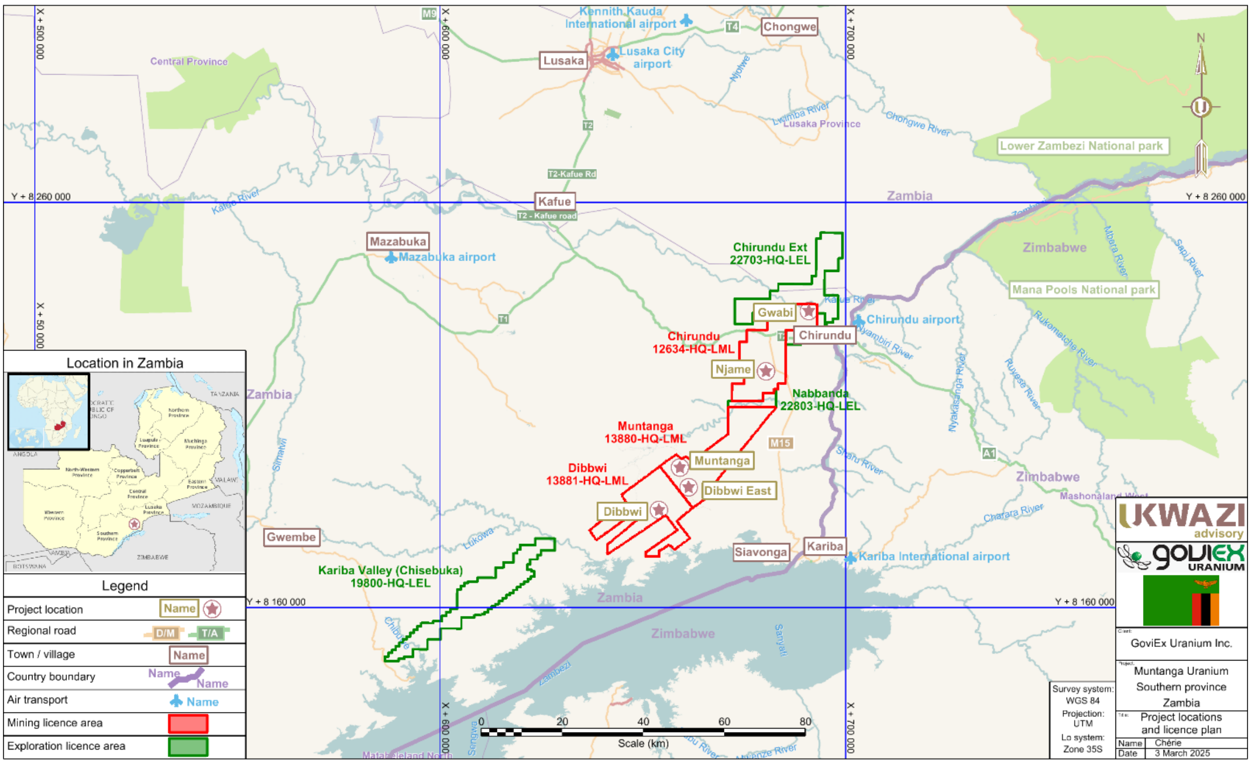

Figure 2-2: The Project site and licence boundaries

1.4. Relevant legislation, permits and approvals

The key legislation with regard to permitting a mining project in Zambia and the applicability and status with regard to the Project are detailed in the following sections.

1.4.1. The Mines and Minerals Development Act 2015

The Mines and Minerals Development Act states that all mineral rights ("MR") are vested in the President of Zambia on behalf of Zambia. This act specifies how the rights to prospect, mine and dispose of minerals can be acquired and held. It confers on the holder exclusive rights to carry on mining and prospecting operations in the mining licence area. This includes erecting the equipment needed to mine, process and transport the minerals, disposal of mining wastes, stockpiling of minerals or waste products and prospecting within the licence area. It gives preference to Zambian products, contractors and services as well as employment of citizens from construction and operation through to decommissioning.

A large-scale mining licence is granted for 25 years and the holder must maintain security and ensure that there are no illegal miners in the licence area, provide an annual audited financial statement to the Mining Cadastre Office, a return showing compliance with obligations, annual mine plans, ore recovery and production costs and produce ore resource and reserve statements every two years.

A mineral processing licence is required for mineral processing activities. However, the holder of a mining licence may construct and operate a mineral processing plant within their licence area without a mineral processing licence.

For the export of minerals, a mineral export permit issued by the Director of Mines is required. This is valid for one year and is limited to the quantities specified in the permit. For radioactive minerals, the applicant must comply with the requirements of the Ionising Radiation Protection Act 2005. GoviEx will comply with the requirements of the act and apply for an export permit for the uranium product as the project progresses.

1.4.2. Water Resources Management Act 2011

The Water Resources Management Act establishes the Water Resources Management Authority ("WRMA") and defines its function and powers. The Act provides for the protection of Zambia's water resources and that the said resources should be used, developed, conserved, managed and controlled sustainably, beneficially, reasonably and equitably for the needs of the present and future generations. It provides for the management, development and utilisation of water resources to take into account climate change adaptation.

1.4.3. Ionising Radiation Protection Act 2005

The Ionising Radiation Protection Act establishes the Radiation Protection Authority's functions and powers, and provides for the protection of the public, workers and the environment from hazards related to ionising radiation or the release of radioactive material. This act requires a licence issued by the Radiation Protection Authority which GoviEx will apply for as the project progresses.

1.4.4. Zambia Wildlife Act 2015

This act makes provision for the management and conservation of wildlife in Zambia. It provides for the implementation of the Convention on International Trade in Endangered Species of Wild Fauna and Flora, the Convention on Wetlands of International Importance especially as Waterfowl Habitat, the Convention on Biological Diversity, the Lusaka Agreement on Cooperative Enforcement Operations Directed at Illegal Trade in Wild Fauna and Flora and other international instruments to which Zambia is party. The Zambia Wildlife Regulations 2016 and Zambia Wildlife Order 2016 provide the mechanism to implement the Act. These may relate to measures specified under an Environmental Impact Assessment ("EIA") approved by the Zambian Environmental Management Agency ("ZEMA").

1.4.5. Environmental Management Act 2011

The Environmental Management Act ("EMA") is the principal piece of legislation governing environmental management in Zambia. ZEMA is mandated to ensure the sustainable management of natural resources and protection of the environment, and the prevention and control of pollution. The EMA provides for public participation in environmental decision-making and access to environmental information. In particular, section 29 of the Act states that "A person shall not undertake any project that may have an effect on the environment without the written approval of the Agency, and except in accordance with any conditions imposed in that approval". The Act provides specific regulations for pollution control, water, air, waste management, pesticides and toxic substances, noise, ionizing radiation and natural resources management.

GoviEx currently holds a licence for the management of hazardous waste.

1.5. Environmental liabilities

The Project is a greenfield exploration site with no history of previous development or industrial activity. As a result, there are no obvious current environmental liabilities. Should the Project be implemented and mining operations commence, environmental liabilities to decommission and remove infrastructure, rehabilitate disturbed areas and manage long-term effects will be incurred. A conceptual Closure Plan and cost estimate has been prepared as part of this FS as detailed in Section 20.7.

GoviEx has established a permanent exploration camp immediately adjacent to the Muntanga deposit. Should the project not progress to an active operating mine, the camp will have to be closed, and any uranium-bearing sample material appropriately disposed of. It is probable that local communities could use the camp infrastructure.

1.6. Accessibility, climate, local resources, infrastructure and physiography

1.6.1. Topography, elevation and vegetation

The Project area is located within the Zambezi Rift System in southern Zambia. The Zambezi River flows to the east of the area, following the border between Zambia, Zimbabwe and Mozambique.

Surface runoff is predominantly contour-controlled but occasionally fault-controlled. Lake Kariba is situated at 485 m above mean sea level and the Project region varies between 500 m and 960 m above sea level.

Vegetation typically consists of forest, which is predominantly miombo woodland mixed with munga and mopane, but there are also small areas of agricultural fields and degraded grassland.

1.6.2. Access to property

Proximity to Chirundu and Siavonga means that the area is relatively well-serviced with sealed roads and numerous gravel tracks, which lead to farms and villages.

Access to the Project is by the sealed main road running between Chirundu and Lusaka and the sealed road to Siavonga, then turning onto the sealed road leading to Munyumbwe, in Gwembe District. The main roads are in fairly good condition, but the actual Project area is located east of the main roads and accessed via gravel roads that require a four-wheel drive vehicle (Figure 2-1). The nearest commercial airport is in Lusaka, located 144 km by road from Chirundu.

1.6.3. Climate

The Project has a climate described as tropical wet and dry, with very distinct wet and dry seasons. Meteorological information is obtained from the nearest station at Lusitu, approximately 40 km north-east of Muntanga with a similar elevation and climate.

Annual rainfall is recorded as between 600 mm and 720 mm, and the wet season occurs in the hottest summer months between November and March. The highest rainfall generally occurs in January/ February. Maximum temperatures range from 22 °C to 46 °C and minimum temperatures range from 20 °C to 38 °C during the hottest months. The highest temperatures typically occur just prior to the onset of the rains in October. Wind speeds are greatest during this period and can range from approximately 2.5 ms^{-1} to approximately 3.6 ms^{-1}, typically from an east-southeast direction. Lightning storms can be common during the hottest months and occasionally hailstones are experienced, associated with thunderstorms. During the wettest months of October to February, the average daily sunshine hours can range from only 4.6 hours (February) to 8.8 hours (October).

During the cooler months of April to October, rainfall varies significantly spatially and temporally. Maximum temperatures range from 23 °C to 40 °C and minimum temperatures range from 6 °C to 28 °C, with lowest temperatures occurring in June and July. Winds are typically much calmer during the colder, dry months, particularly between April and August. On average, at least nine hours of daily sunshine is generally received during the drier months of May to September.

The highest maximum temperature recorded at the Project site was 46 °C and the lowest minimum temperature that has been recorded is 6°C. Evaporation typically exceeds precipitation for most of the year. Monthly relative humidity generally ranges from a minimum of 46 % in September to a maximum of 79 % in December.

1.6.4. Local resources

There are many small villages located around the Project area and approximately 10 % of the land is used for small-scale agriculture including millet and maize, sorghum, bananas, cotton and minimal animal husbandry. There are currently no industrial activities within the Project area.

According to the United States Department of Agriculture, the regional land classification indicates medium to low potential for sustainable development based upon extremely weathered and iron-rich soils. The soils are typically nutrient-deficient and not good at retaining water although they are easily worked.

1.6.5. Infrastructure

Except for the main road systems described in Section 5.2, there is limited to no infrastructure within the immediate Project area.

1.6.5.1. Roads

As described in Section 5.2, there are some sealed roads in the area which run between Lusaka, Chirundu, Siavonga and the bottom road to Munyumbwe in Gwembe District. Although they are in fairly good condition, access to the actual Project site is gravel tracks which require four-wheel drive access. Local communities rely on bicycles or carts for transport.

1.6.5.2. Power supply

There are two 88/11 kV substations located at Gotagota and Chirunda, both supplied by an 88 kV transmission line from the bigger Leopards Hill Sub Station, which is supplied via 330 kV high voltage transmission lines from the Kariba North Bank Hydroelectricity Scheme. Power lines do traverse the Project area around Njame, but most of the local villages are not connected to the national power network, and households near Muntanga and Dibbwi rely on wood for heating and cooking, and candles and kerosene lamps for lighting.

1.6.5.3. Local villages and towns

The region is sparsely populated: Chirundu, Siavonga, Kafue and Lusaka are the closest major urban areas. Lusaka has a population of 3.2 million (2023). Siavonga and Chirundu are small towns with local government and town council administration offices. The two towns have banking facilities, a post office, district hospitals and general stores. There are no defined commercial areas within the immediate vicinity of the Project and grocery stores are typically located along the sealed roads to Chirundu and Siavonga. Much of the housing in the villages is typically wooden structures covered with mud. Communities are predominantly rural, mostly subsistence farmers producing maize, cotton, millet, sorghum and vegetables; the majority of crops grown are for household consumption. Charcoal is also produced for sale and used as a main fuel source alongside wood, for heating and cooking.

1.6.5.4. Water supply and sanitation

The Project area relies on wells and boreholes for potable water and local watercourses are used as a source of irrigation. Sanitation is managed by way of pit latrines in some households. The Southern Water and Sewerage Company has a treatment plant located on the Zambezi River that supplies piped water to Siavonga, but this does not reach the Project site. GoviEx has provided 15 water boreholes to local villages.

1.6.5.5. Education and health care facilities

There are very few schools and health facilities in the Project area and typically they have insufficient staff and resources. The main challenges faced are long distances, poor staffing levels, inadequate funding and transport.

The development of local health and school facilities through sustainable development projects carried out by the Project will benefit the local communities. To date, GoviEx, through its corporate social responsibility programme, has provided clinics for the villages of Muntanga, Sikoongo, and Chizilika, and Nurses' houses at Muntanga, Chizilika and Syamwiinga. Temporary schools have been constructed at Muntanga and Mutuba to help the local population have access to education near their locality. These temporary schools will be demolished when the mine proceeds into development and rebuilt in the new relocation areas. The company pioneered the development of schools at Hachibozu, Chizilika and Njame villages by constructing classroom blocks. At Chaanga, two laboratory classrooms were built, leading to the upgrading of the school from the primary to the secondary level. Staff houses for teachers have been constructed by GoviEx at Hachibozu, Chizilika and Muntanga.

GoviEx has also supported these sectors with the provision of solar power to facilitate the delivery of information and communication technology lessons at the Hachibozu and Muntanga schools. At Muntanga clinic, GoviEx has provided a solar system for refrigeration of medical supplies. Piped water, using solar systems, has been supplied to Hachibozu School and village, and Muntanga Clinic. Muntanga Clinic shares the facility with the adjacent Muntanga School. GoviEx is supporting three educational support programmes intended to aid GoviEx's policy of local employment and development by creating of pool of local skilled labour it can tap from as the project goes into mine development.

1.6.5.6. Telecommunications

Telecommunications are provided to the Muntanga area by Airtel, MTN and Zamtel. Airtel and MTN provide 4G services for internet connectivity.

1.6.6. Physiography

The topography is defined by geology and consists of gentle, low escarpment-type hills with steep and/or craggy scarp northwest slopes and gently sloping southeast dipping slopes.

1.7. History

Uranium was first identified in the area in 1957 by ground survey which located five anomalous areas in the vicinity of Bungua Hill, west of Siavonga. In 1958 and 1959 Chartered Exploration found low-grade uranium mineralisation that could be followed for over 800 m of strike extent.

The main exploration took place between the late 1970s and mid 1980s initially by the Geological Survey of Zambia, followed by AGIP SpA, an Italian petroleum company. The AGIP exploration campaign included a regional ground radiometric surveying programme which highlighted numerous radiometric anomalies along the northern shores of Lake Kariba including Dibbwi and Chisebuka. Several of the anomalies were investigated via more detailed ground radiometric surveying and subsequent drilling. Their campaign predominantly focused on the Muntanga and Dibbwi deposits, and in 1983/4 a small uneconomic resource was outlined at Njame but AGIP ceased work in 1985.

Numerous historical Mineral Resource Estimates were prepared by a variety of companies and consultants using several different methodologies. Considering the successive exploration drilling completed at the project, all estimates, in general, compare favourably and demonstrate similar U_3O_8 grades and tonnages.
There has been no uranium production from any of the Project licence areas.

1.8. Geological setting and mineralisation

1.8.1. Regional geology

The Project area is situated within the Karoo Supergroup, which comprises thick, carboniferous to late Triassic age, terrestrial sedimentary strata and is widespread across much of what is now southern Africa. The Karoo Supergroup was deposited within an extensive foreland basin created when compression and accretion along the southern margin of Gondwana resulted in the formation of the Cape Fold Belt to the south. To the north, crustal extension due to thermal doming following the assembly of the Pangean supercontinent around 320 million years ago, resulted in the formation of a northeasterly trending series of rift basins (Yeo, 2010). The rifting is believed to have been associated with the breakup of Gondwanaland during the Permian Period, followed by the opening of the proto-Indian Ocean in the Jurassic; with a final episode related to the development of the East African Rift system in the late Cretaceous and early Tertiary times.

During the Cenozoic, the East African Rift System propagated south-westerly across the continent and led to the reactivation of the Karoo rift basins as well as the formation of new fault depressions, such as the Okavango Rift (Laletsang et al., 2007; Kinabo et al., 2007), the southeastern extension of the mid-Zambezi and Luangwa rift systems.

The Karoo Supergroup in the Project area consists of three formations within the Lower Karoo; the Siankondobo Sandstone Formation, overlain by the Gwembe Coal Formation, which itself is overlain by the Madumabisa Mudstone Formation. The Siankondobo Sandstone Formation consists of fine clastic sediments with a basal diamictite and conglomerate overlain by siltstones and sandstones. The Gwembe Coal Formation is comprised of carbonaceous mudstones and siltstones interspersed with coal seams and sandstones, while the Madumabisa Mudstone Formation consists of a thick sequence of non-carbonaceous grey mudstones with calcareous bands. The Madumabisa Formation is unconformably overlain by the Upper Karoo which consists of four formations; the Escarpment Grit is overlain by the Interbedded Sandstone and Mudstone Formation, followed by Red Sandstone which is finally capped by the Jurassic Bakota Basalt Formation. The Escarpment Grit comprises a 400 m thick series of continental arenaceous silici-clastic sediments with interbedded mudstones. Although locally referred to as Escarpment Grits, this group is a correlative of the Beaufort Group elsewhere in the Karoo Supergroup and contains interbedded mudstones and fine-grained sandstones, as well as grits and conglomerates.

The Project is situated in the mid-Zambezi Rift Valley. In the region, known uranium mineralisation typically occurs within the Upper Karoo whereas the Lower Karoo hosts much of the coal reserves of Zambia, Zimbabwe and South Africa. At the Project, all of the known uranium mineralisation occurs within the Escarpment Grit. Similar sandstone-hosted uranium mineral deposits occur in many of the Karoo rift basins including Letlhakane in the Kalahari Basin of Botswana and Kayelekera in the Rukuru Basin of Malawi. The underlying Madumabisa Mudstone appears to have acted as an impermeable barrier controlling the base of the mineralisation. The Escarpment Grit itself shows a wide variation in lithology which is typical of continental sediments. Uranium mineralisation appears to have been introduced after sedimentation (epigenetic) and occurs as fillings into pore spaces, fractures, joints, coatings on sand grains and occasionally along steeply dipping cross beds.

The Escarpment Grit Formation consists of coarse to very coarse-grained sandstones that are locally conglomeratic and fine upwards into more fine-grained sandstones and intercalated mudstones. Silicified wood is abundant locally. AGIP geologists historically distinguished two informal members in the Escarpment Grit suggesting a change in fluvial style. A lower “Braided Facies” member is characterised by relatively poorly sorted sandstones and pebbly sandstones with mudclasts and thin discontinuous mudstones, and an overlying “Meandering Facies” member is characterised by well-sorted upward-fining sandstones (i.e., point bar deposits) with mudclasts and pebble-lag layers, interbedded with laterally extensive mudstones.

Stratabound uranium mineralisation in the Escarpment Grit is known in the lower part of the “Meandering Facies” at Njame, and the upper part at Dibbwi. Association with boundaries between sandstone-dominated stratigraphic units suggests that permeability contrast is a factor controlling uranium mineralisation. Widespread soft-sediment folds suggest syn-depositional seismic activity and fault re-activation, with potential seismic pumping of diagenetic fluids contributing to the mineralisation event.

A geological map of the Dibbwi-Muntanga area is shown in Figure 2-3.

1.8.2. Regional geological structures

The mineralised zones are offset and impacted by various faults and fractures, but the mineralisation itself does not appear to have any significant structural controls.

Regionally, the Muntanga uranium deposit and other uranium occurrences in southern Zambia, lie near the northwest margin of the Mid-Zambezi Graben. This structure is essentially a half-graben, with its faulted footwall against the Precambrian crystalline rocks on the northwestern Zambian side, and passive onlap on crystalline basement rocks on the southeastern Zimbabwean side. The Mid-Zambezi Graben is subdivided into two major sub-basins by the northeast-trending Kamativi - Chizarira - Matusadona basement block. The north sub-basin is fault-bounded on both its margins and is, hence, a true graben. Cyclic upward fining of Karoo strata (Catuneanu et al., 2005) reflects episodic, fault-controlled subsidence in the graben.

At Muntanga, Dibbwi and Dibbwi East, northeast-trending faults likely controlled deposition of the Escarpment Grit “Braided Facies”, and fault-related folds may control blind mineralisation in the Dibbwi and Dibbwi East area. The Muntanga area of the Mid-Zambezi Valley is characterised by a series of northeast-trending, fault-bounded cuestas or fault blocks, uplifted to the northwest and dipping to the southeast. Three major northeast-trending anastomosing fault systems can be distinguished in the Muntanga area: the Lusitu, Dibbwi and Bungua Mountain fault zones. There are numerous minor faults of limited extent trending northwest to north.

Minor north- to northwest-trending faults, with extents of less than four kilometres, crosscut the major fault systems. In contrast with the major faults, they appear to be normal faults. These minor faults likely formed in response to differential uplift on the major faults. One of these extends southerly into the Dibbwi East mineral deposit.

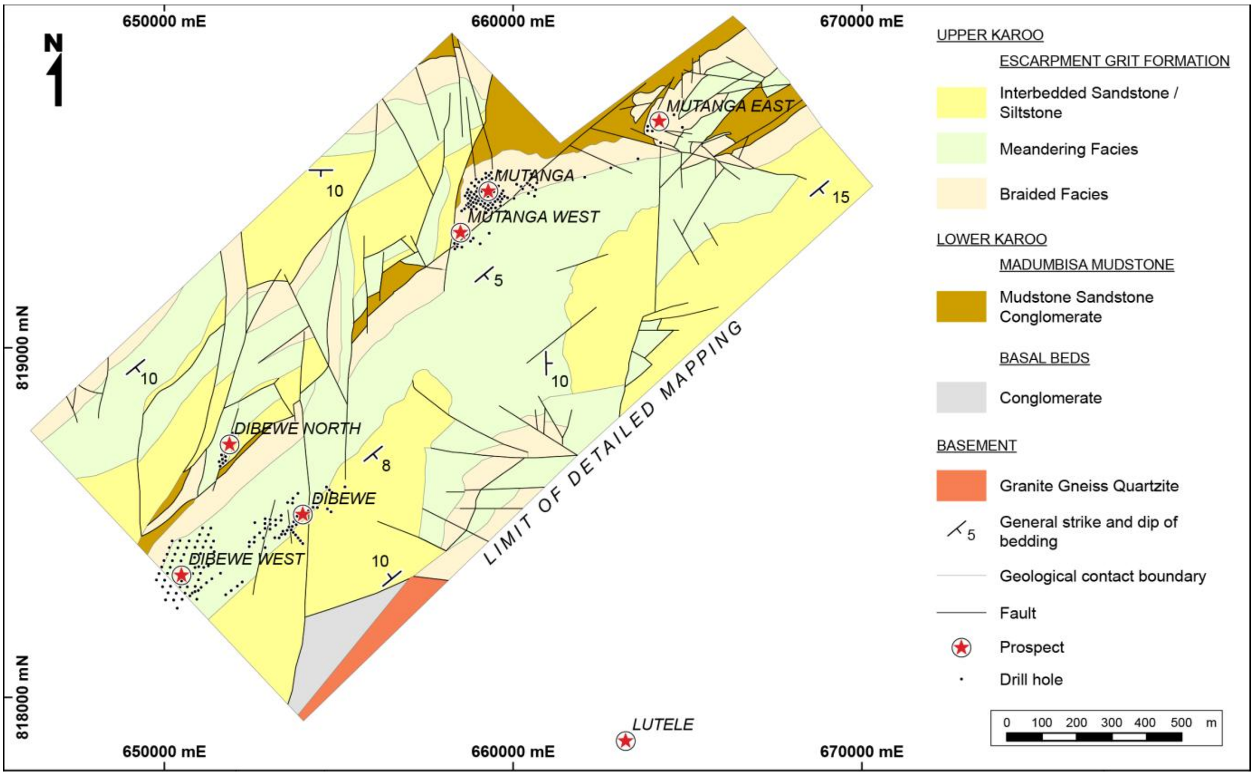

Figure 2-3: Geological map of the Dibbwi-Muntanga area (Source: CSA, 2013)
Note: This map contains historical spelling. “Dibewe” = Dibbwi, “Mutanga” = Muntanga

The Njame uranium deposit consists of Escarpment Grit exposed on a gentle dip slope which faces to the southeast. In the northwest, the slope is a much steeper scarp controlled by the position of a northwest dipping normal fault. This fault is downthrown several hundred metres to the northwest, representing one of a number of faults that have caused imbrication in the Kariba Rift. The sequence is also cut by several smaller strike-parallel normal faults, which have caused northwest block-down displacements of up to 25 m. Similarly, the eastern limit of the Njame mineralisation is a major southeast trending wrench fault that truncates the slope and the stratigraphy. The sequence is cut by several smaller strike-parallel normal faults, which have caused down displacements of the northwest block.

Gwabi uranium mineralisation forms a broadly tabular body that dips very gently to the southeast and occurs at very shallow depths of between 3 m and 29 m below surface. In the northwest, the slope is a much steeper scarp controlled by the position of a northwest dipping normal fault. Minor post-mineralisation faulting has locally caused metre-scale offsets to the mineralisation and may have truncated the mineralisation along its southern boundary.

1.8.3. Mineralisation

At Muntanga, Dibbwi and Dibbwi East, uranium mineralisation appears to be later than at least some of the normal faults which cut the Escarpment Grit Formation. This is evident from the good correlation of the radiometric logging data between adjacent holes within the Muntanga deposit separated by interpreted faulting (Lusambo, 2011).

The source of the uranium is believed to be the surrounding proterozoic gneisses and plutonic basement rocks. Having been weathered from these rocks, the uranium was dissolved, transported in solution and precipitated under reducing conditions in siltstones and sandstones. Post-lithification fluctuations in the groundwater table caused dissolution, mobilisation and redeposition of uranium in reducing, often clay-rich zones and along fractures. Mineralisation is not strictly associated with a particular unit in the stratigraphic section. It is observed to occur in both the fine-grained and coarser material and in mudstones, especially where fractures and mud balls occur. Some mineralisation occurs in association with manganese oxide or disseminated with pyrite. Mineralisation in some bore holes is seen to occur where there was a grey alteration, limonite and feldspar alteration and in dark grey mudstones (Sakuwaha, 2011). The strata dip in the south-easterly direction and mineralisation seems to occur along dip.

Uranium mineralisation occurs in a number of different associations, namely disseminated uranium mineralisation, uranium mineralisation associated with mudstones and siltstones, fracture-hosted uranium mineralisation and primary uranium mineralisation.

At Njame, the uranium mineralisation occurs at the interface between siltstones and sandstones at redox boundaries. Approximately 25 % of the Njame mineralisation is siltstone hosted, with the balance in coarser-grained sandstones and grits. Drilling identified two main mineralised horizons; the thickest, most consistent and highest grade is the lower horizon within the second sequence from the base. Drilling was carried out along the entire length of the 5 km long system, with uranium mineralisation encountered along the entire length. The siltstone horizons are generally laterally continuous for hundreds of metres, except where younger grit/ sandstone channels have cut through them. There is a clear stratigraphic control on mineralisation at the deposit scale, although structural control may be present on a larger scale.

Similarly to Njame, the uranium mineralisation at Gwabi is related to the redox front; there is one main mineralised horizon which appears to be controlled by both lithology and the redox boundary. It is hosted by the coarse-grained sediments that are interpreted to be the along-strike continuation of the Escarpment Grits which host the Njame uranium mineralisation. Uranium mineralisation at the Gwabi deposit occurs in red, oxidised, coarse-grained sandstones, grits and pebble conglomerates which overlie a green, non-mineralised, reduced silty-shale horizon. This is interpreted to represent a major redox boundary and maybe the regional unconformity between the upper and lower Karoo.

1.8.4. Deposit types

The primary uranium mineralisation in the Karoo rocks of the Project conforms to a sandstone-hosted fluvial channel-type deposit. Sandstone uranium deposits are contained within medium to coarse-grained sandstones deposited in a continental fluvial or marginal marine sedimentary environment. Impermeable shale or mudstone units are interbedded in the sedimentary sequence and often occur immediately above and below the mineralised horizon. Uranium is mobile under oxidizing conditions and precipitates under reducing conditions, and thus the presence of a reducing environment is essential for the formation of uranium deposits in sandstones.

Only one Karoo uranium deposit, Lotus Energy's Kayelekera deposit in Malawi, has been developed. Kayelekera is on care and maintenance, but in October 2024 Lotus Energy released an accelerated restart plan with an eight-to-ten-month timeline to first uranium production. Other deposits have economic potential.

These deposits have some key features in common:

- All are hosted in fluvial arkosic sandstones that have undergone post-depositional faulting and uplift (tectonic inversion)
- All lie at or near the surface and hence, typically have strong surface radiometric expression
- All appear to have tabular geometry; no classic roll-front deposits have been convincingly demonstrated
- Most feature a range of mineralisation styles, including primary uranium oxides and silicates in relatively reduced sandstones, secondary uranyl phosphates or vanadates in more strongly iseoxidised sandstones and secondary mineralisation remobilised into surficial calcretes
- Mineralisation is commonly associated with stratigraphic contacts indicative of a marked drop in stream energy.

1.9. Exploration

1.9.1. Muntanga, Dibbwi, and Dibbwi East

The earliest phase of exploration for uranium in the area covering the Muntanga and Dibbwi deposit areas was conducted by AGIP in the late 1970s to the mid-1980s. AGIP carried out systematic exploration, comprising outcrop mapping, ground radiometric surveys, air-borne photographic and geophysical surveys, trenching and pitting. Regional exploration drilling was carried out in the broad Muntanga-Dibbwi area.

During 2006, a detailed aeromagnetic and radiometric survey was carried out by OmegaCorp which confirmed the position and tenor of the existing uranium prospects and identified additional targets, based on interpreted radiometric signatures.

During August and September 2013, Geotech Ltd. carried out a helicopter-borne geophysical survey of the Project. Principal geophysical sensors included a versatile time domain electromagnetic ("VTEMplus") system, and horizontal magnetic gradiometer. Ancillary equipment included a global positioning system ("GPS") navigation system and a radar altimeter. A total of 1 903 line kilometres of geophysical data were acquired during the survey. In-field data quality assurance ("QA") and preliminary processing were carried out daily during the acquisition phase. Preliminary and final data processing, including the generation of final digital data and map products, was undertaken from the office of Geotech Ltd.

Geological mapping of the Muntanga property was undertaken during August and September 2014 by Remote Exploration Services of Cape Town, South Africa. A total of 324 line kilometres of mapping traverses were completed including 1 815 mapping stations. Field mapping data were integrated with airborne geophysical data, satellite imagery and previous geological maps and interpretations to produce a revised geological map for the Muntanga property.

The Project area was covered with soil geochemical and radon surveys from 2013 to 2015. The objective of the surveys was to delineate any significant exploration targets outside of the drill-defined uranium deposits. Previous drilling had largely focused on testing airborne radiometric anomalies and the soil geochemical and radon approach allowed for possible detection of blind or buried mineralisation, particularly in areas of thick or transported regolith. Surveys were carried out in the dry months between May and November. Coincident soil and radon stations were 100 m apart on 800 m spaced northwest-southeast survey lines. Survey data and results were stored in an Access database. Prior to the implementation of the surveys, calibration exercises were conducted over known mineralisation to establish optimal methodologies.

The soil geochemical and radon surveys produced numerous anomalies across the Project area and new exploration targets were defined for follow-up. The soil geochemical and radon methods utilised adequately detected the drill-defined mineralisation and showed a reasonable correlation with radiometric anomalies, thereby confirming this exploration approach. The new exploration targets were defined based on combinations of anomalous soil uranium, soil uranium pathfinders, radon and soil radioactivity. In some cases, the targets corresponded with surficial cover (thicker soils) alluding to a buried source. Targets located over prospective geology and structure were prioritised for follow-up. Figure 2-4 shows the gridded soil uranium results.

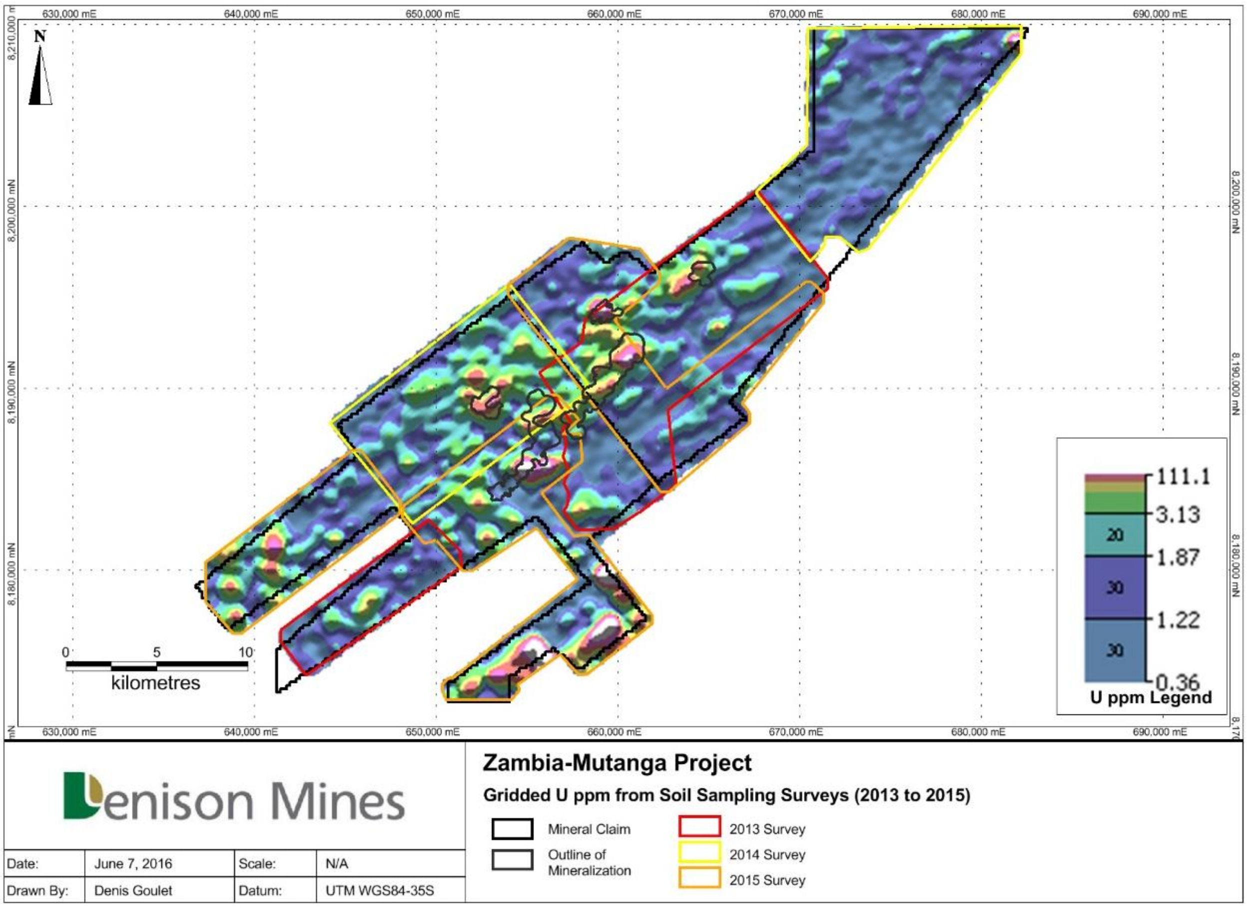

Figure 2-4: Gridded soil uranium results

Trenching was undertaken over priority targets to test for additional mineralised horizons outside of the drill-defined uranium deposits. The trenching provided a cost-effective follow-up methodology, before any drilling, to test targets generated from the soil geochemistry and radon surveying. Trenches provided a means of accessing the fresh bedrock, or otherwise saprock, for the in-situ determination of geology and mineralisation. Trenches were typically located along, and parallel to, the soil and radon survey lines which were roughly perpendicular to stratigraphic strike and known mineralisation. The soil and radon anomalies tended to follow stratigraphic strike parallel trends. Trenches were designed to cover the entire anomaly and to extend into the background by $^1/_3$ to $^1/_2$ of the anomaly width in each direction, and sampling was undertaken over intervals where elevated gamma readings were encountered.

Weak mineralisation was encountered in the majority of the trenches and a few distinct mineralised horizons were discovered. Leaching at the regolith-bedrock interface where trench samples were collected may be the reason higher grades were not encountered.

1.9.2. Gwabi and Njame

In the late 1970s to the mid-1980s, AGIP completed a major regional programme of ground radiometric surveying which identified numerous radiometric anomalies in the area along the northern shores of Lake Kariba. A number of these anomalies were evaluated with more detailed ground radiometric surveying and a small number were subsequently tested with rotary percussion drilling, wagon drilling ("WD") and in some cases diamond drilling.

Albidon (Zambia) Limited acquired the Mugoto PLLS.250 tenement in June 2005 as part of their Munali nickel project tenement holding. The tenement was subsequently transferred to Albidon Exploration Limited in 2006 with Ministerial approval. In October 2005, Albidon Exploration Limited signed a joint venture agreement with African Energy Resources Ltd ("AFR") under which the latter would explore the eastern part of the Mugoto PLLS for uranium, coal and coal bed methane. This is the area in which both the Gwabi and Njame deposits are located.

AFR undertook a major exploration programme from 2006 to 2007, which included:

- Drilling at the Njame deposit which identified additional uranium mineralisation to that defined by AGIP
- An airborne radiometric survey identified a significant uranium anomaly at Gwabi; this was tested with surface radiometric surveying, soil sampling and
- Subsequent drilling at Gwabi which outlined uranium mineralisation.

Through 2008 and 2009, AFR then completed a series of infill drilling programmes, comprising reverse circulation ("RC") and diamond drilling ("DD") to define the extent of both the Njame and Gwabi deposits, and tighten the drilling patterns to improve confidence in the geological and Mineral Resource models.

1.9.3. Photogrammetry and light detection and ranging

In 2022, Rocketmine from South Africa were contracted to carry out a photogrammetry and light detection and ranging ("LIDAR") survey using a drone platform. The areas selected for surveying covered each of the deposit areas at Dibbwi, Dibbwi East-Muntanga, Njame and Gwabi. The LIDAR data have been used in the current MRE to define the ground surface.

1.10. Drilling

Drilling at the Dibbwi East, Dibbwi, and Muntanga deposits was completed in three major phases. Historically, drilling was conducted by AGIP and the Zambian Geological Survey (1973 to 1984), followed later by OmegaCorp and Denison (2006 to 2012), and most recently by GoviEx between 2021 and 2024, which was predominately comprised of infill drilling at Dibbwi East and limited confirmation drilling at the Muntanga and Dibbwi deposits. In 2024, the drilling consisted mostly of sterilisation drilling around the proposed infrastructure and relocation sites.

Drilling at the Gwabi and Njame deposits was managed by AFR and completed between 2006 and 2009. GoviEx conducted limited drilling at Njame and Gwabi from 2022 to 2024.

Summaries of annual drilling completed on the main deposit areas are provided in Table 2-1. The drilling techniques used on the Project include diamond core drilling, RC, down-the-hole ("DTH") hammer, air core ("AC") and percussion WD. 954 DD holes totalling 73 209 m and 2 284 percussion holes totalling 157 358 m were drilled. Further drilling of 989 holes of all types totalling 68 369 m was completed on areas adjacent to the main deposits for sterilisation and infrastructure geotechnical purposes.

Table 2-1: Project deposit drilling summary

Deposit	Period	DDH holes	DDH metres	Percussion holes	Percussion metres
Dibbwi East	1980 to 2024	174	21 569	508	59 978
Dibbwi	1980 to 2024	222	20 193	204	16 762
Muntanga	1980 to 2024	350	21 484	612	30 711
Njame	2006 to 2024	162	8 115	671	36 899
Gwabi	1980 to 2024	46	1 848	289	13 008
Total		**954**	**73 209**	**2 284**	**157 358**

1.10.1.Muntanga, Dibbwi and Dibbwi East deposits

Prior to 2006, AGIP and the Zambian Geological Survey undertook drilling across the Muntanga and Dibbwi licence areas (circa 1980). Several hundred drill holes were completed, and the main known deposits were identified, along with a number of prospects. However, due to insufficient historical records being available to verify the reliability of these data, all drill hole information from the time frame has been excluded from the MRE process.

During the OmegaCorp/ Denison tenure (2006 to 2012), RC and DD were the principal methods of exploration and delineation drilling after initial geophysical surveys.

In 2006, OmegaCorp drilled DDH to twin previous drilling at the Muntanga deposit. Results confirmed the broad tenor of the earlier mineralised intercepts. From 2007 to 2008, Denison completed work on the Muntanga deposits, focussing on the Muntanga and Dibbwi areas. The work included an appraisal of all available data and from this information, Denison produced several databases covering Muntanga along with other prospects.

A two-phase drilling campaign resumed in April 2011. Phase 1 drilling on Dibbwi East and Muntanga targets commenced in April and ended in July 2011. The results for Phase 1 confirmed the continuity of uranium mineralisation identified in the 2008 drilling programme at Dibbwi East, with a northeast-southwest strike length greater than 2.5 km.

Based on the encouraging results obtained with the Phase 1 drilling over the Dibbwi East area, a Phase 2 drilling programme was completed between August to October 2011. This drilling programme discovered primary mineralisation at depth and increased the strike length to 4.0 km. In 2012, the primary targets for drilling were the Dibbwi East, Dibbwi and Muntanga deposit areas, to further delineate and infill within the deposit footprints.

During the 2021 to 2023 drilling campaigns, GoviEx carried out drilling mostly on the Dibbwi East deposit to infill the existing drill pattern to a 100 m line spacing with drill holes at 50 m between holes. Selected areas were drilled at a closer spacing of 25 x 25 m to assess the continuity of mineralisation for MRE purposes. Uranium grade data were determined using a downhole gamma probe. DDH made up approximately 10 % of the total drilling meterage, with several holes drilled to collect metallurgical samples, and others drilled to twin historical holes for data validation purposes. DDH were drilled on all deposits by GoviEx during the 2021 and 2022 drilling campaigns.

The 2023 drilling programme was driven by the success of the 2021 to 2022 exploration efforts and the updated MRE reported on July 17, 2023. During the second half of the year, a total of 15 835 m of infill drilling was conducted across 160 drill holes, primarily at the Dibbwi East and Muntanga deposits. The focus was on upgrading Inferred Mineral Resources into the Indicated category by improving drill hole spacing and expanding the Dibbwi East open pit Mineral Resource. DD was undertaken to validate gamma and radon corrections in downhole logs, with twin holes drilled to match earlier percussion drilling. Additionally, 14 geotechnical holes were completed across all deposits to optimise pit wall geometry. Hydrogeological work included 29 holes to assess dewatering needs, along with nine water bores and pump testing to support future water supply planning. Geotechnical investigations involved the digging of 119 shallow test pits to evaluate soil characteristics for infrastructure design, including leach pads and waste dumps.

Building on the progress made in 2023, the 2024 drilling programme focused on sterilisation drilling around proposed pit areas, mine plant locations, and resettlement zones. The drilling was done to ensure the identification of non-mineralised zones suitable for infrastructure placement and community relocation. The company did hydrogeological drilling to secure water supply for the processing plant. These activities further refine project planning and align with the broader pre-construction objectives.

All DD holes were logged for lithology, structure, alteration, mineralisation and geotechnical characteristics. In 2009, data were entered into DHLogger software on laptops in the field and then transferred into a Fusion database. Hard copies of drill logs are stored at the site. In 2021 and 2022, the DDH core data were collected using the Seequent MX Deposit Application, with data stored directly in the cloud. Most of the core mark-ups and photography were done on the drill pad so that the quality of the core was not lost during transport to the core farm. The core was then logged geologically using the descriptions outlined above and then marked up for sampling.

Prior to core logging, down-hole geophysical probe information was reviewed, with the major lithological contacts, structures and mineralised horizons being inferred from the gamma and conductivity readings. These inferences are then reviewed alongside the core. The core was then measured and metre marked, and the core recovery, longest piece and scintillometer readings were recorded

Down-hole geophysical logging was done with the use of down-hole geophysical probes, which measure the electrical properties of the rock from which lithologic information can be derived and natural gamma radiation, from which an indirect estimate of uranium content can be made. Parameters measured by the down-hole geophysical probes are conductivity, resistivity, self-potential, single-point resistance, deviation, and natural gamma.

Data from the 2006 to 2012 drilling programme was converted by Denison used an in-house developed computer program known as GAMLOG to convert the measured cps of the gamma rays into an equivalent per cent U_3O_8 ("eU_3O_8%"), while down-hole gamma data collected by GoviEx from 2021 to 2024 were converted into eU_3O_8 using the ALT Wellcad software supplied by an external geophysical contractor, Terratec Geophysical Services.

1.10.2.Njame and Gwabi deposits

Drilling was carried out by AFR using a combination of DDH, RC and AC techniques. The AC method was only used at the early-stage exploration at Njame in 2006, and all subsequent drilling at the Njame and Gwabi deposits was completed by RC and DDH techniques.

RC drilling was used for obtaining suitable samples for MRE at these deposits and was carried out along drill lines spaced between 25 m and 50 m apart along prospective anomalies.

The majority of the DDH drilling was completed in 2008 and was carried out by Capital Drilling (Zambia) Limited.

Collar positions for all holes were initially established using handheld GPS. Drill sites and access were cleared using a bulldozer when required and the drill position was re-marked using handheld GPS. Upon hole completion, each drill hole was left with a Polyvinyl chloride ("PVC") collar tube cut at ground level. The collar coordinates were re-checked using handheld GPS. Subsequently, most drillhole collars were surveyed with a differential GPS by a professional surveyor and Lusaka-based Rankin Engineering.

AFR used well-documented procedures for RC and DDH sample logging. In general, RC chips were logged immediately after drilling whereas the core was logged after being carefully joined up and marked on a V-trough. The information recorded included lithological, structural, geotechnical, weathering/ oxidation and mineralogical logs. For cored holes, the mineralised zones of each were selected at the discretion of the logging geologist.

GoviEx completed three drill holes on each of the Njame and Gwabi deposits in 2022 for data confirmation and geometallurgical sampling. Logging and sampling procedures used for these holes are consistent with the procedures described above for drilling completed on the Muntanga, Dibbwi and Dibbwi East deposit drilling campaigns.

1.11. Data verification

1.11.1.Data verification by previous companies

Limited down-hole radiometric quality assurance/ quality control ("QAQC") data are available to support the historical drilling completed prior to 2006, however Denison's drilling campaigns, which represent the majority of historical data for the Muntanga, Dibbwi and Dibbwi East deposits, used a variety of systematic checks and standards for routine checking and calibration of down-hole radiometric logging tools.

CSA Global ("CSA") conducted data verification exercises in 2009 and 2012 to support the historical MRE updates completed by CSA. The following items were included in their data verification process, including exploration protocols used by Denison:

- Core sampling, sample preparation and assaying
- QAQC procedures
- Drill hole collar and down-hole deviation surveys
- Down-hole radiometric logging procedures and results and
- Database validation.

No material issues were identified by CSA regarding data collected by Denison.

AFR completed twin hole drilling of RC and DDH to confirm AC holes, as well as DDH to confirm RC holes. A total of 23 twins were completed and compared versus the original holes during the exploration programmes at Njame and Gwabi. Although some of the holes were not directly comparable due to extra sampling requirements, the results indicate that the comparison between twin holes is generally acceptable.

1.11.2.Data verification by SRK

As part of the 2021 and 2022 drilling campaigns, check surveys were conducted on a limited number of historical drill hole collars to verify the location and relative position of the historical collars to drill holes completed by GoviEx. Through this verification exercise, it was determined that the UTM WGS84 drill hole collar coordinates for the historical drill holes were on average approximately 7.25 m off in the easting coordinate and 0.15 m off in the northing coordinate. Therefore, all historical collar coordinates for drill holes located on the Muntanga, Dibbwi and Dibbwi East deposits were shifted to align with the 2021 to 2023 survey locations.

In addition, all drill hole collar elevations were adjusted to align with the 2023 LIDAR survey conducted on the Project area in in Quarter (“Q”) 2023. All drill hole collar adjustments were completed in preparation for Mineral Resource estimation purposes.

SRK conducted a review of the Project drill hole assay database, comparing database entries to the original Lab assay certificates. Approximately 10 % of historical assay database entries and 85 % of recent assay database entries were validated against the original Lab assay certificates, and no errors were noted.

SRK reviewed the down-hole radiometric and eU_3O_8 profiles for all 2021 and 2023 drill holes, and where radon contamination was identified, adjusted (corrected) the eU_3O_8 profiles to produce a more robust eU_3O_8 grade profile.

1.12. Mineral processing and metallurgical testing

Metallurgical testwork on Muntanga has been conducted since the 1980s by various groups. Heap leaching of Chirundu ore has previously been evaluated, in the late 1980s testing by the government laboratory. The testwork resulted in uranium recoveries up to 90 % at low sulfuric acid consumption rates of less than 5 kg/t ore leached. Similar results were reported by Denison for Muntanga and reported in the previous PFS (SRK, 2016).

From historic and recent geometallurgical and mineralogical studies, two key issues were assessed:

- Efficiency of natural liberation of uranium from the matrix of quartz from the conglomerate phases; and
- The occurrence and mineralogy of uranium phases, including grain size, association and liberation.

The studies identified that most of the primary uranium occurs as uraninite (over 80 %) with autunite and uranophane occurring in the oxide ore. In addition, trace coffinite was also identified as well as fine grained rutile-uraninite intergrows. The vast majority (more than 90 %) of the U-bearing mineral particles studied in the test programme were liberated to whilst less than 10 % remained unliberated. The U-bearing minerals in the latter category were predominantly attached to the quartz boundaries.

Between 50 % to 60 % of the U-bearing particles in the test programme were associated with quartz, but the average grain size was small so that the proportion of the total deportment was low at approximately 2 %. The U-bearing mineral autunite was associated within the pores of the host rock (sandstone), not within the clay cement. The data suggests that the timing of the U mineralisation was post-depositional, which is supported by the low association between the U-bearing minerals and the quartz grains and clay cement.

In the FS study reported here, metallurgical test work was performed on five composites, from all the prospects, with Muntanga and Dibbwi East accounting for approximately 80 % of M&I Resources. The samples were sourced from existing drillcore from the 2023 drilling programme. They were:

- Composite 1: Dibbwi East Oxidised; coarse oxidized sandstone containing visible secondary uranium minerals of autunite, umhoite and carnotite
- Composite 2: Dibbwi East Reduced; black reduced siltstone-sandstone mixed containing finer grained groundmass with some lithoclasts
- Composite 3: Muntanga + Njame; Both samples were fine grained grey to green sandstone-siltstone mix with some graphitic and pyritic material
- Composite 4: Gwabi Oxidised; medium grained sized oxidized siltstone
- Composite 5: Dibbwi Main Oxidised; coarse oxidized sandstone containing visible secondary uranium minerals of autunite, umhoite and carnotite.

The application of curing acid optimisation test protocols provided for a substantial reduction in the leach cycle in the columns to between 20 and 60 days. Most of the uranium dissolves in the curing stage and washes out during initial irrigation. The application of curing acid provides a means to introduce the acid rapidly and evenly throughout the bed and avoid acid limitation within zones of poor solution contact.

High uranium dissolutions of 90 % and above were achieved in the columns which contained Muntanga and Dibbwi East ores. Only the Gwabi ore yielded lower dissolutions of 75 %.

Slumping was minimal (less than 1.7 %) as measured by a decrease in the height of bed. Slumping was also reduced by compacting the agglomerates during loading to between 1.46 t/m^3 to 1.49 t/m^3 by hammering on the sides of the column. This prevents further compaction during leaching on account of agglomerates “settling” to a higher bulk density during wetting. Moreover, no permeability restraints were observed during irrigation in the columns at 10 L/m^2/h. Compared with test heaps, columns are also known to provide “wall support” which is absent in heaps.

The good permeability may be attributed to a low percentage of silt plus clay (-75 µm) material in the feed solids. The silt plus clay fraction is known to block pores if it is present in amounts greater than 10 % to 14 %. Even though the samples contained a large percentage of sand, normally classified as -2 mm or -4 mm, this is not associated with poor permeability, although it is associated with increased surface area and moisture content. During the leach, however, decrepitation resulted in an increase in the fines (-75 µm) content to 15 %, close to the recommended limit, although this did not translate into permeability restraints in the columns. Hydrodynamic column tests on the residues indicated that the columns operated close to saturation on account of the high fines content generated during the leach.

Uranium dissolutions by one-metre section and by size class were uniform, suggesting that there was no reagent limitation down the height of the column, nor is uranium finely disseminated/ locked in coarser rocks. In other words, the crush size has little effect on the uranium dissolution, as the uranium is liberated, even in the coarser rocks.

The resulting flowsheet will comprise the following stages:

- Primary crushing and ore sorting at satellites (if mined), sorted ore trucked to Central plant using a road-going hauling fleet
- Primary crushing of ore from Central pits
- Secondary and tertiary crushing of combined ore
- Agglomeration with sulfuric acid and stacking
- Heap leaching to produce a pregnant leachate solution containing uranium, iron and other impurities followed by ripeos (spent ore) reclamation and disposal
- Recovery of uranium from pregnant leachate solution using ion exchange, with barren solution recycled to the heap
- Concentration of ion exchange eluate by nanofiltration, with recycling of recovered acid to ion exchange elution
- Neutralization of excess acid in NF concentrate using lime
- Precipitation of iron with sodium hydroxide
- Precipitation of uranium using hydrogen peroxide
- Calcining and packaging to produce U_3O_8.

1.13. Mineral Resource estimates

The Mineral Resource model prepared by SRK considered 2 366 historical drill holes drilled between 2005 and 2012, and 468 drill holes drilled by GoviEx from 2021 to 2023. The MRE work was completed by André Marcel Deiss, Pr.Sci.Nat., P.Geo. an “independent qualified person” as this term is defined in NI 43-101. The effective date of the Mineral Resource statement is January 31, 2024.

In the opinion of SRK, the MREs reported herein are reasonable representations of the global uranium Mineral Resources found in the Project at the current level of sampling. The Mineral Resources have been estimated in conformity with the generally accepted Canadian Institute of Mining, Metallurgy and Petroleum “Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines” dated November 29, 2019, and “Definition Standards for Mineral Resources and Mineral Reserves” published May 10, 2014, and are reported in accordance with the Canadian Securities Administrators’ NI 43-101 standards of disclosure for mineral projects. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resource will be converted into Mineral Reserves.

The database used to estimate the Project MRE was audited by SRK. SRK believes that the current drilling information is sufficiently reliable to interpret with confidence the boundaries for uranium mineralisation and that the sample data are sufficiently reliable to support Mineral Resource estimation. Figure 2-5 shows a location map of the Project’s uranium deposits.

Figure 2-5: Location map of the Muntanga uranium deposits

The Mineral Resource evaluation methodology involved the following procedures:

- Database compilation and verification
- Review of Njame and Gwabi's historical MRE
- Construction of grade shell wireframe models for the boundaries of uranium mineralisation for the Muntanga, Dibbwi and Dibbwi East deposits
- Data conditioning (compositing and capping) for geostatistical analysis and variography
- Block modelling and grade interpolation
- Mineral Resource classification and validation
- Assessment of “reasonable prospects for economic extraction” and selection of appropriate cut-off grades (“COG”)
- Preparation of the Mineral Resource statement.

The Mineral Resource drill hole database for the Project contains 2 834 drill holes totalling 191 751 m of drilling; 468 of these drill holes were drilled by GoviEx between 2021 and 2023 totalling 52 924 m of drilling. The database contains 33 280 uranium (U_3O_8) assays and 114 364 m of down-hole radiometric probe data converted in equivalent U_3O_8 (eU_3O_8) grade data for MRE purposes.

For the Gwabi and Njame deposits, mineralisation domains were generated using the three-dimensional (“3D”) software package Gemcom Surpac® (“Surpac”). Uranium mineralisation occurs in fine to coarse-grained sedimentary units consisting of siltstone, sandstones, pebbly/gritty sandstones, and grits-to-pebble conglomerates. Mineralised lenses occur as sub-parallel layers with shallow dips of 2° to 5° to the southeast at Njame, and to the east-northeast at Gwabi, and were defined using a 100 ppm U_3O_8 COG.

At Njame, the main concentration of uranium mineralisation occurs at the contact between sedimentary sequences where there is rapid change from fine to coarse sediments. At Gwabi, the main concentration of uranium mineralisation is hosted in a 10 m to 20 m thick coarse-grained sandstone located above a thick siltstone/ mudstone unit.

For the Muntanga, Dibbwi and Dibbwi East deposits, mineralisation domains used were defined based on grade shells generated using a 100 ppm eU_3O_8 COG with an 80 ppm eU_3O_8 cut-off low-grade halo. The updated mineralisation domain models incorporate additional drill hole information and database QAQC conducted since the previous MREs were completed in 2009 for Muntanga and Dibbwi (CSA, 2009); and 2012 for Dibbwi East (RPA, 2012); 2017 for all three deposits (SRK); 2023 for Muntanga and Dibbwi (SRK, 2023) and 2023 for Dibbwi East (SRK, 2023). 3D grade shells were generated using Leapfrog software predicated on eU_3O_8 grade data obtained from down-hole radiometric probing.

The resulting block model quantities and grade estimates were reviewed to determine the portions of the MRE having "reasonable prospects for eventual economic extraction" ("RPEEE") from an open pit mine. SRK considers that the blocks located within the conceptual pit envelopes show RPEEE and can be reported as a Mineral Resource as reported in Table 2-2.

Table 2-2: Mineral Resource statement the Muntanga Uranium Project, Zambia, effective date, January 31, 2024.

Category	U_3O_8 cut-off [ppm]	Deposit	Tonnes [Mt]	U_3O_8 Grade [ppm]	U_3O_8 Metal [Mlb]
Measured	110	Gwabi	1.1	254	0.6
	90	Njame	2.5	358	2.0
Indicated	90	Muntanga	8.6	369	7.0
	90	Dibbwi	3.2	253	1.8
	90	Dibbwi East	31.3	372	25.7
	110	Gwabi	2.7	374	2.2
	90	Njame	1.0	306	0.7
Total M&I			**50.4**	**359**	**40.0**
Inferred	90	Muntanga	3.4	278	2.1
	90	Dibbwi	1.0	213	0.5
	90	Dibbwi East	7.1	252	3.9
	110	Gwabi	0.2	272	0.1
	90	Njame	1.1	329	0.8
Total inferred			**12.8**	**263**	**7.4**

*Notes

1. The effective date of the Mineral Resource statement is January 31, 2024. The QP for the estimate is André Deiss, Pr.Sci.Nat., P. Geo., Associate Consultant (Resource Geology) of SRK Consulting (Canada) Inc.
2. Mineral Resources are prepared in accordance with CIM Definition Standards (CIM, 2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (CIM, 2019).
3. Mineral Resources are constrained within an optimised pit shell using a uranium price of USD 100 /lb U_3O_8, mining costs of USD 3.30 /t, processing costs of USD 9.00 t ore, additional mining costs of USD 0.55 /t, G&A costs of USD 1.50 /t, Transport costs of USD 1.50 and a royalty of 5%.
4. Mineral Resources are reported at a U_3O_8 COG within the optimised pit shell and are inclusive of Mineral Reserves.
5. Mineral Resources are inclusive of mineralisation in the 80 ppm halo but reported above the relevant cut-off and classed as Inferred Resources. This mineralisation represents approximately 5 % of the total Mineral Resources metal (Mlb).
6. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources will be converted into Mineral Reserves in the future.
7. All figures have been rounded to reflect the relative accuracy of the estimate.

1.14. Mineral Reserve estimate

The QP accepting the professional responsibility for the open pit Mineral Reserve estimates section is Mr Jaco Lotheringen Pr Eng, member of the SAIMM (Registration number: 401237) and registered as a professional engineer at the ECSA (Registration number: 20030022). This section relates specifically to the Muntanga and Dibbwi East Mineral Reserve estimates completed for this TR and based on the Mineral Resource models and estimates as reported in Section 1.13. Project base case economic analysis shows that the Project life of mine ("LOM") plan, used to estimate the Mineral Reserves, provides a positive net present value of the free cash flow, confirming that the Mineral Reserves are economically viable, and that economic extraction can be justified. The author is not aware of any additional mining, metallurgical, infrastructure, permitting, or other factors not presented in this report that could materially affect the Mineral Reserve estimate.

During the course of the FS, financial analysis showed that the mining schedules for Dibbwi, Gwabi and Njame are value accretive, but would not earn sufficient returns under current market conditions. Based on this, the simplified project development strategy was developed to focus on the Muntanga and Dibbwi East deposits only. The LOM plan and hence the Mineral Reserve estimate thus exclude these satellite deposits.

To conform with NI 43-101 standards, the Mineral Reserve estimate was derived from Measured and Indicated (“M&I”) Mineral Resources only. The M&I MRE are listed in Table 2-2 are reported inclusive of the associated Mineral Reserve.

The conversion of the Mineral Resource estimate to a Mineral Reserve estimate followed a conventional approach, commencing with open pit optimisation techniques incorporating economic parameters and other modifying factors. The ultimate (optimal) pit outlines (shells) were used to create practical and detailed open pit designs accounting for the inclusion of batters, berms and haul roads.

These pit designs then provided the ore and waste mining inventories for a detailed production schedule that demonstrates viable open pit mining. This schedule, which in turn provides the physical basis for cash flow modelling, is described in Section 22.

The resulting Mineral Reserve estimate for the Project is shown in Table 2-3.

Table 2-3: Muntanga Mineral Reserve statement, in-pit inventory on January 1, 2025

Mineral Reserve class	Tonnes [Mt]	U_3O_8 Grade [ppm]	U_3O_8 Contained [Mlb]	Contribution [%]
Muntanga pit				
Proven	-	-	-	0
Probable	8.4	331	6.12	100
Subtotal	**8.4**	**331**	**6.12**	
Dibbwi East pit				
Proven	-	-	-	0
Probable	31.2	317	21.86	100
Subtotal	**31.2**	**317**	**21.86**	
Total project				
Proven	-	-	-	0
Probable	39.6	320	27.99	100
Total Mineral Reserve	**39.6**	**320**	**27.99**	

Notes:
1. All figures are rounded to reflect the relative accuracy of the estimate and have been used to derive sub-totals, totals and weighted averages. Such estimates inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, Ukwazi does not consider them to be material.
2. The Concession is wholly owned by, and exploration is operated by GoviEx.
3. The standard adopted in respect of the reporting of Mineral Reserves for the Project, following the completion of required technical studies, is in accordance with the NI 43-101 guidelines and the 2014 CIM Definition Standards, and have an Effective Date of January 1, 2025.
4. The Open Pit Mineral Reserves are reported with engineered pit designs using a cut-off grade per area varying between 70.1 ppm U_3O_8 and 85.1 ppm U_3O_8, which is based on a selling price of USD80 /lb U_3O_8, reference mining cost of USD 3.30 /t rock, additional ore mining cost of USD 0.55 /t ore, additional ore hauling cost of USD 0.18 /t ore-km, incremental depth mining cost of USD 0.05 /t/10m bench, processing cost of USD9.00 /t ore, royalty of 5 %, G&A of USD1.50 /t ore, port costs of 1.50 /lb U_3O_8 and recoveries varying per location between 74.6 % and 93.3 %.
5. The Open Pit Mineral Reserves are derived from a regularized block models of 5 m x 5 m x 2.5 m (Muntanga) and 10 m x 10 m x 2.5 m (Dibbwi East) and include an additional dilution and 5 % mining loss.
6. Jaco Lotheringen of Ukwazi is an appropriate “independent qualified person” as defined in National Instrument 43-101 and has completed site inspections of the deposit

1.15. Mining methods

For context, the mine site layout is shown in Figure 2-6. The main features shown are:

- The Muntanga and Dibbwi East open pits
- The Muntanga and Dibbwi East waste dumps
- Surface haul routes
- The Central processing complex, including run of mine (“ROM”) tip, crushers and conveyors, heap leach facility (“HLF”), processing plant, offices and mining workshops and offices
- Spent ore dump
- Stockpile area.

Mining follows conventional drill and blast, shovel and truck mining practice. The sequence of mining activities is conventional and is generally as follows:

- Grade control drilling delineates the ore zones
- A grade control model will developed as basis for the design of blast limits and digging blocks

- Ore, waste or mixed blocks will be blasted to design, according to layouts based on hole patterns and powder factors to suit ground conditions
- Trim blasts and perimeter blasting techniques will be used to ensure pit wall profiles are cut to the correct angle and to minimise wall damage
- Diesel/ hydraulic excavators will load the blasted rock onto a fleet of articulated dump trucks (”ADT”) of 25 m³ capacity
- Ore was scheduled to be hauled directly to the ROM crusher, and waste material will be hauled to the surface dumps or dumped in-pit once sufficient pit floor space is available.

The mining schedule was based on a ROM production rate of 3.5 million tonnes per annum (“Mtpa”). ROM production was scheduled to commence at the Muntanga deposit, due to its low 1.21 stripping ratio (“S/R”) and progress to the Dibbwi East pit (with a 4.29 S/R). On depletion of the Muntanga pit, Dibbwi East will serve as the sole ROM production feed.

The Muntanga mine design and schedule were based on a cutback mining approach starting with a north-eastern boxcut and progressing westwards in a series of 40 m-wide mining benches. In-pit dumping will take place once sufficient void space is created. Since the ore body outcrops at the northern side of the pit and dips at a haulable gradient of approximately 6°, pit access will be gained via the outcrop or by means of temporary in-pit ramps with a minimum 120 m allowance between any two mining benches.

While Muntanga initially produces the bulk of the ore, Dibbwi East makes up for the shortfall as the Muntanga pit depletes to sustain the overall mining production profile. Dibbwi East was scheduled based on three pushbacks. Mining will commence at the first pushback, with scheduled pre-stripping of the second pushback to support a sustainable production profile during the Muntanga pit depletion.

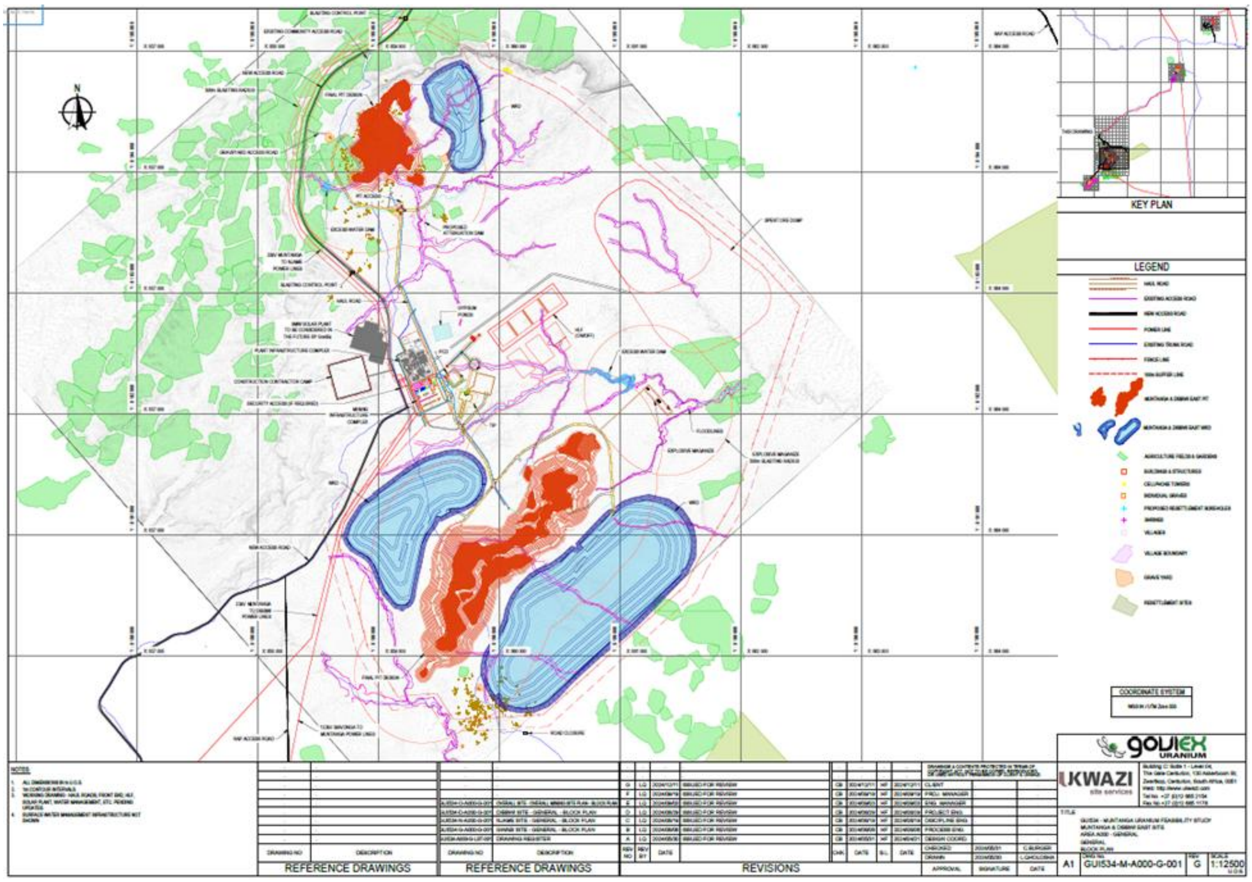

Figure 2-6: Mine site layout

The equipment selected for the development of the mining schedule includes the following items of major equipment (the number required for steady-state mining in brackets):

- Drilling: Sandvik DI650i drill rigs (three to four required depending on schedule requirements)
- Excavators: Caterpillar 395 (seven to eight required)

- Dump trucks: Cat 745 ADT (45 to 49 required).

Appropriate support equipment includes dozers, front-end loaders (“FELs”), graders, water and diesel bowsers, secondary rock breakers, water pumps, mobile lighting plants, tractors loaders backhoe (“TLBs”), site-use buses and light delivery vehicles (“LDVs”).

Specifications from the original equipment manufacturers (“OEMs”) were used to estimate the number of units required, operating cost and replacement schedule.

The total tonnage mined each year is shown in Figure 2-7. Over the LOM, a total of 183.8 million tonnes (“Mt”) of material was schedule, comprising 39.6 Mt of ore at a grade of 320 ppm U_3O_8 and 144.2 Mt of waste.

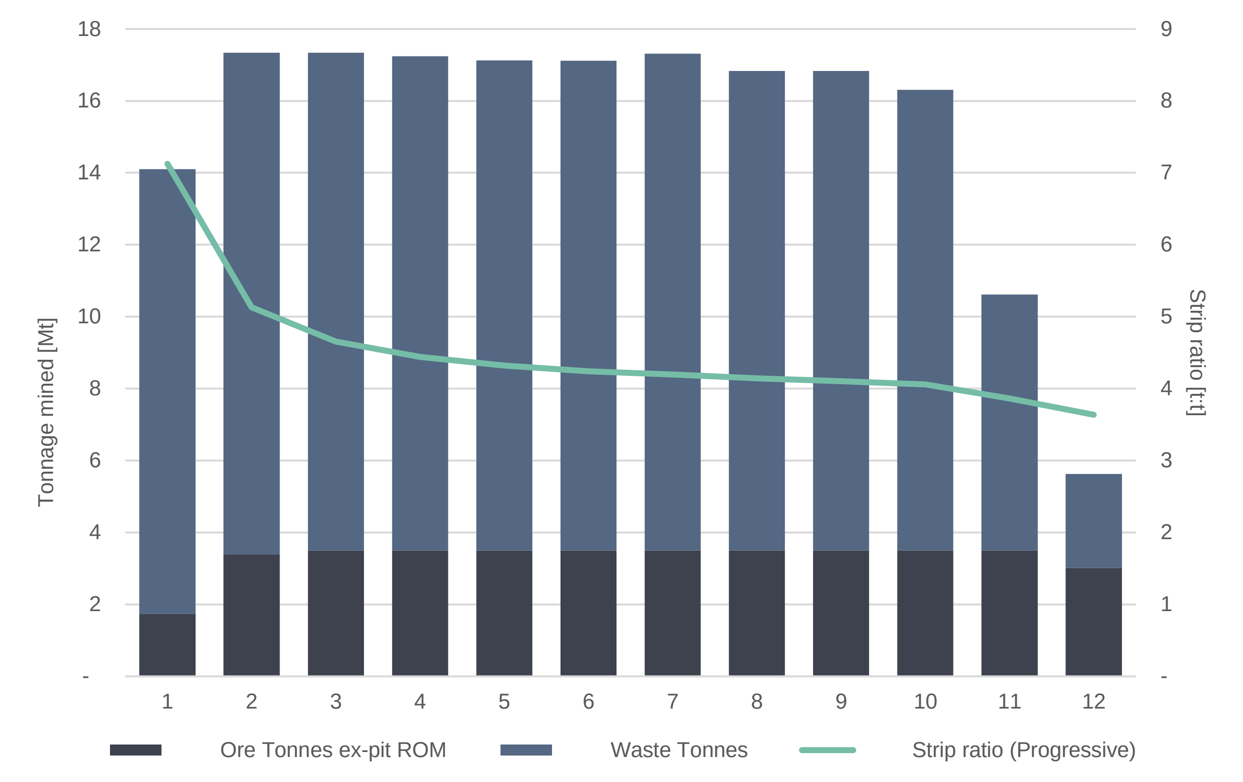

Figure 2-7: LOM schedule annual material movements and progressive strip ratio

1.16. Recovery methods

The central processing plant (“CPP”) was designed to handle a total of 3.5 Mtpa of ROM material sourced from the central Muntanga and Dibbwi East mining sites and, if mined in the future, sorted ore from the Dibbwi, Gwabi and Njame satellite mining sites. The mix of ore from the respective pits will vary over time.

Processing of the ROM ore to produce a saleable U_3O_8 product takes place in three stages:

1. **Ore preparation:** ROM ore hauled from the pits is placed into the ROM tipping bin and enters three stages of crushing, before undergoing agglomeration in preparation for leaching
2. **Heap leach:** The agglomerated ore is placed on the HLF for leaching. The pregnant leach solution is pumped to the uranium recovery and purification plant, and the spent ore is placed on the spent ore dump after rinsing
3. **Uranium recovery and purification:** Uranium recovery by ion exchange (“IX”) is followed by the recovery of U_3O_8 by eluting the uranium-loaded resin using a sulfuric acid solution. The eluate undergoes a nanofiltration process facilitating sulfuric acid recovery for recycling to the elution process. Following this, the concentrated solution is dosed with hydrated lime and sodium hydroxide to neutralise residual acid and remove deleterious minerals such as iron, after which it is dosed with hydrogen peroxide, leading to uranyl peroxide precipitation. The precipitate is dewatered and calcined, and the final product and packed into drums as U_3O_8 or yellowcake.

The overall process flow sheet is shown in Figure 2-8.

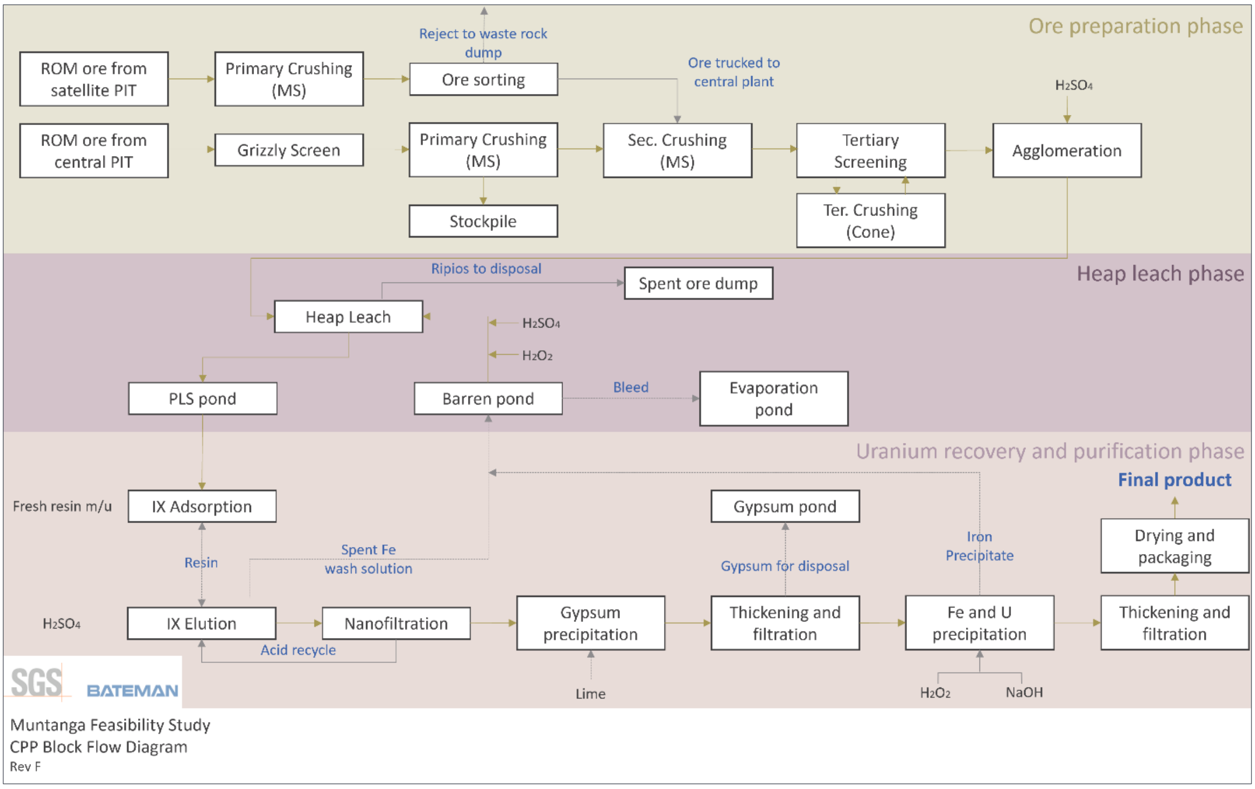

Figure 2-8: Block diagram of the CPP

Because of their different geochemical properties, the U_3O_8 recovery rate differs from orebody to orebody. The recovery for Dibbwi East oxide ore was estimated at 91.3 %, 89.7 % was estimated for the Dibbwi East reduced ore, and the Muntanga reduced ore recovery was estimated at 93.0 %. Over the life of mine, the 39.6 Mt of ore fed to the plant yields 25.3 Mlb of U_3O_8.

1.17. Project infrastructure

Primary access roads: The Project's primary access roads ("PARs") connect plant and mine sites to the nearest national road. They will be used during construction and operation and will be used by local traffic. Figure 2-9 shows the PARs for the project. The main PAR for Muntanga and Dibbwi East joins the national D500 road to the Central site and requires a bridge to be built over the Machinga River.

General Central complex infrastructure: The process plant design includes offices, changerooms, dining facilities and other infrastructure required by the general departments of the Project not directly involved in production activities.

Mining infrastructure: Infrastructure to support all aspects of the mining operation was developed. This includes hauls roads, a mining infrastructure complex, offices, workshops, change houses, ROM pads and waste rock dumps ("WRDs").

Water management: Infrastructure to manage all water-related requirements for the Project was designed. This includes stormwater, surface water, groundwater, potable and process water, pit dewatering (both in-pit and interception dewatering), and water quality management and monitoring

Bulk power supply: Power supply is required at the various plant, mine and accommodation sites. Muntanga will connect via a new, dedicated connection to the Siavonga 330 kV/132 kV/33 kV substation, which is adjacent to the Kariba Dam requiring 11 kV switchgear supplied by the grid feeding the site's electrical distribution system, from whence it is part of the design of each area.

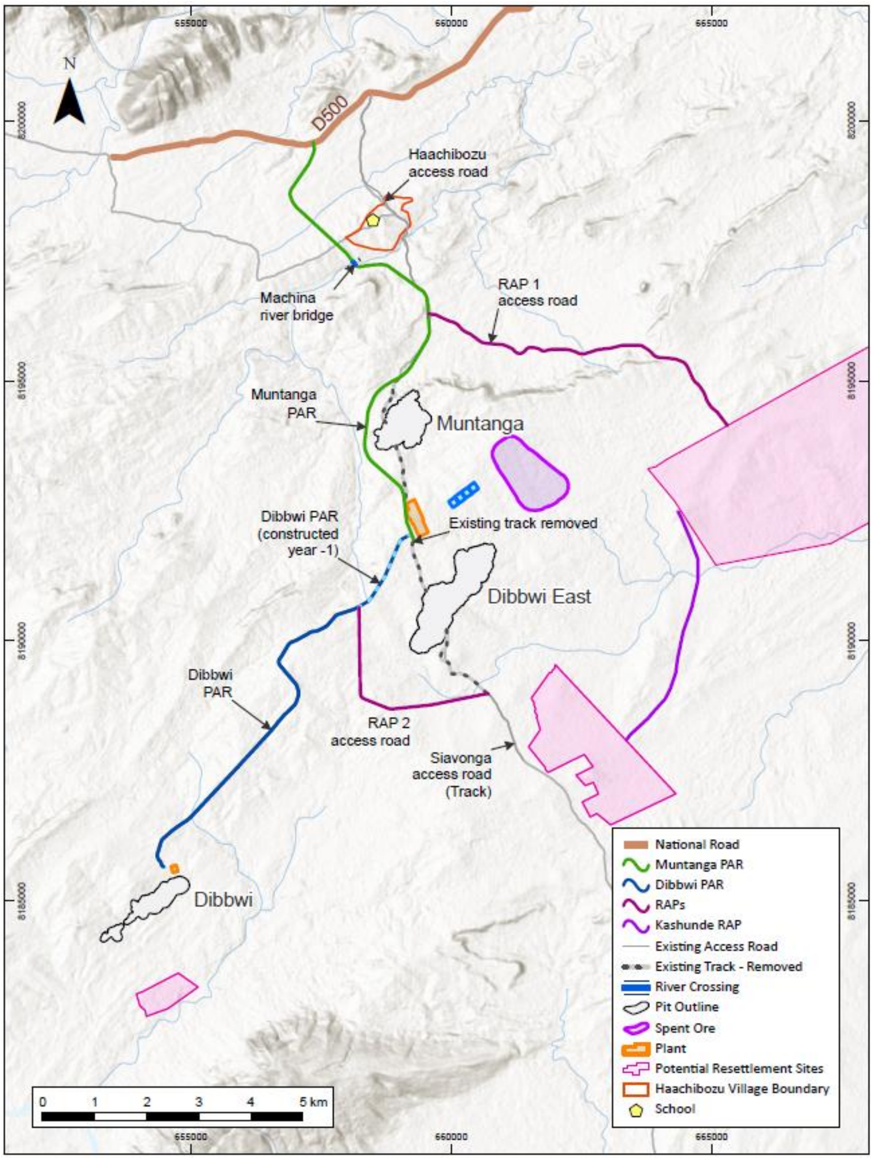

Figure 2-9: Access roads for Muntanga, Dibbwi East and Dibbwi

1.18. Marketing studies and contracts

This section provides an overview of the fundamental principles of the uranium market and how the derived U_3O_8 is sold into the market; transported; and transformed for use in nuclear reactors. As such the following elements will be described:

- Understand the position and role of uranium within the nuclear fuel cycle
- Analyse U_3O_8 demand with particular reference to the U_3O_8 requirements of the world's reactors
- Explain the transformation of U_3O_8 into uranium hexafluoride and the role of the conversion facilities that provide such a service
- Summarise the requirements for transportation of U_3O_8 from the Project to the conversion facilities
- Examine the contractual relationship between GoviEx as the uranium producer and the conversion facilities.

Since 2011 the key impact on primary uranium demand was excess inventories throughout the supply pipeline. Increasing nuclear energy production and primary uranium supply constraints have resulted in declining inventories. The uranium miners have reduced their inventories to just-in-time levels through supply reductions, sell down of surplus inventories, on-market purchases and in the case of Kazatomprom, the sale of its surplus inventory to the financial fund Yellow Cake.

Utility inventories have been declining as long-term contracts have unwound, and utilities have undertaken active inventory control. This has been compounded by the uncertainty associated with geo-political factors, especially affecting the United States of America ("USA"), including the Iranian sanctions, Russia Suspension Agreement and Section 232/Nuclear Fuel Working Group. During 2020 and into the start of 2021, utilities were affected by COVID-19, and nuclear energy generation decreased by approximately 4 % in 2020, resulting in a 20 % to 30 % decline in annual purchases.

In late 2021, the activity of Sprott Physical Uranium Trust, and in 2022, the disturbances in the Russian sphere of influence have dramatically focussed the industry's attention on the security of fuel supply issues and have increased the uncertainty faced by buyers and sellers alike.

Inventories on conversion and enrichment material have been declining, as highlighted by the rising price and increasing concerns on conversion and enrichment capacity in the medium to long term.

The increasing supply constraint and declining inventories have already been noted by the improving uranium price. Based on history alone, uranium prices can make swings when future production levels are uncertain due to the long lead times required to bring new projects online. Since the actions taken by Cameco and Kazatomprom to constrain supply, and the recent market impacts of Sprott Physical Uranium Trust and conflicts in the Russian sphere of influence, the uranium price has responded positively.

In July 2023, the military coup in Niger resulted in political instability in the region and saw international and regional sanctions imposed on the military junta. Consequently, the mining and shipping of Nigerien uranium has been seriously curtailed. Niger has been supplying over 20 % of European Union uranium in recent years.

In early 2024, the announcement by Kazatomprom that their forecast production targets would not be met caused more uncertainty in the market and put upward pressure on long-term contract prices, which rose from United States Dollar ("USD") 56 /lb in June 2023 to USD 80 /lb in June 2024.

Coupled with this was the legislation enacted in the United States of America in 2024 to ban imports of Russian nuclear fuel, subject to possible waivers in the next three years 2025 to 2027. This heightened uncertainty has put further upward pressure on nuclear fuel prices.

The QPs view is that for Muntanga a base price of USD 90 /lb should be applied to the valuation, with sensitivity analysis at USD 80 /lb and USD 70 /lb to the downside and USD 100 /lb and USD 110 /lb to the upside as this will cover the widest range of potential scenarios.

1.19. Environmental studies, permitting, and social or community impact

An environmental impact assessment ("EIA") was prepared for the Chirundu (Njame and Gwabi) sites in 2008. This was based on baseline data collected between March 2007 and February 2008 (AFR, 2008). Similarly, an environmental impact study was prepared for the Project in 2009 by African Mining Consultants ("AMC") as part of the Denison Feasibility Study (MDM, 2009).

As of March 2025, AMC is in in the final stages of a full environmental and social impact assessment ("ESIA") process that builds on the earlier studies but includes a comprehensive update of the baseline studies and assessment of the impacts based on the new project design. GoviEx is committed to developing the Project to International Finance Corporation standards and the ESIA process has been scoped to achieve this. The Project will result in the resettlement of a number of villages and accordingly AMC are developing a Resettlement Action Plan("RAP").

The potential environmental impacts of the Project are being systematically assessed using the source-pathway receptor framework. An environmental management plan ("EMP") will form part of the AMC deliverable. AMC plans to finalise the ESIA in Q1 2025 and submit the report for regulatory comment and approval towards the end of Q1. The regulatory consultation process for the ESIA and Resettlement Action Plan ("RAP") is expected to take approximately 6 -12 months.

The ESIA report will describe what are effectively two phases in the project life:

1. Phase one focuses on the central Muntanga and Dibbwi East area.
2. Phase two describes the development of the satellite deposits at Dibbwi, Njame and Gwabi.

The Project will require the resettlement of local communities at all five project sites. AMC are in the process of finalising the detailed resettlement plan. Full community baseline and household surveys have been completed and the asset inventory cut-off date, November 2024, has been communicated to the people affected by Phase 1 of the Project, which involves those impacted by mining at Muntanga and Dibbwi East.

Phase 2 will impact the communities and farmers at Dibbwi, Njame and Gwabi. These groups have been included in all communications to date and have been comprehensively surveyed. These areas may only be disturbed five to seven years into the project life and resettled later in the life of the mine.

The total number of people subject to physical resettlement and Muntanga and Dibbwi East is 958. The phasing of the resettlement programme is described in more detail in Section 20.3.2.

1.20. Capital and operating costs

Capital and operating costs for the Project were derived by the technical teams working in each aspect of the Project. In general, capital costs were obtained by deriving bills of quantities ("BOQs") based on the designs and issuing requests for quotations to the market in packages comprising the BOQs and/or a pricing schedule, along with detailed specifications. The responses were evaluated for financial and technical merit and used as a basis for the capital cost estimate. If responses were not received on a package, database rates from similar projects were used. Mobile equipment capital costs were based on quotations received from the original equipment manufacturers or their agents.

Operating costs were based on some common factors such as diesel and electricity prices, which were obtained from suppliers and applied to each component of the project. Labour rates were obtained from Zambian mining industry benchmarking in a report by Align Advisors: "Benchmark Salary Report, Zambian Mining Industry 2024" and applied to all labour in the Project. Each technical team drew up detailed labour schedules for their component of the Project. Operating costs were generally estimated using first principles, for example calculating the usage per tonne/ year/ pound of a consumable and multiplying it by the appropriate unit price.

Initial capital expenditure ("Capex") is the expenditure required to purchase the initial mining fleet, develop the processing plant and build all roads and infrastructure, up to the point where mining production can commence and revenue is received. The total initial Capex is USD 282 million as shown in Table 2-4.

Sustaining capital is required thereafter to maintain production levels at the target throughout the LOM, including equipment purchases and replacement, and expansion of facilities such as the HLF, waste and spent ore dumps. This totals USD 101 million over the LOM, of which 93 % is for the replacement of primary mining equipment.

Total LOM capital is USD 383 million.

Table 2-4: Initial development capital

Development Capital [USD '000]	Total	2025	2026	2027
Mining equipment	36 887	0	0	36 887
Mining infrastructure	14 099	570	7 657	5 872
Processing plant	137 721	143	44 753	92 825
Heap leach pads	24 200	2 663	12 497	9 040
Heap leach stacking and reclaiming	25 592	0	11 028	14 564
Power	20 020	934	11 829	7 257
Roads	9 658	6 843	1 770	1 045
Water management	5 824	0	971	4 854
General & administration	4 061	385	1 183	2 493
Resettlement action plan	3 885	647	3 237	0
Total initial Capex	**281 948**	**12 185**	**94 926**	**174 837**

The LOM Opex is shown in on a total, unit cost per ROM tonne, unit cost per U_3O_8 pound and percentage basis. Mining and processing costs make up 87 % of the operating costs.

Table 2-5: Operating cost summary

Opex [USD '000]	USD/ ROM t	USD/ lb U_3O_8	% of Total
Mining	9.55	14.94	46.4
Mining infrastructure	0.19	0.29	0.9
Processing	8.37	13.09	40.7
Stacking	0.85	1.34	4.2
Reclaiming	0.35	0.55	1.7
G&A	0.42	0.66	2.1
Power rebate	-0.13	-0.20	-0.6
Product transport	0.93	1.46	4.5
Closure	0.05	0.07	0.2
Total Opex	**20.58**	**32.20**	**100**

1.21. Economic analysis

The economic analysis was conducted by building a discounted cash flow model for the project, using the financial assumptions detailed in Table 2-6 and the production, Capex, Opex and project implementation schedule discussed above. The model is built in real terms, based on January 1, 2025 USD.

Table 2-6: Financial assumptions applied in valuation

Parameter	Units	Value	Comment
Uranium price	USD/ lb U_3O_8	90	
Corporate income tax rate	%	30	Percent of taxable income, sourced from Zambian tax legislation
Government royalties	%	5	Percent of revenue, sourced from Zambian mining legislation
Discount rate	% p.a.	8	See derivation below
Valuation base date	Date	January 1, 2025	
Tax depreciation rate	Years	5	
Capital expenditure contingency	%	10	Percent of initial capital expenditure

The cashflows for the project over its life are shown in Figure 2-10. The Project returns USD 672 million in free cash flow, resulting in a net present value ("NPV") (at 8 %) of USD 243 million and an internal rate of return ("IRR") of 20.8 %. On a cash basis, the Project pays back by October 2031, within 3.5 years of first revenue.

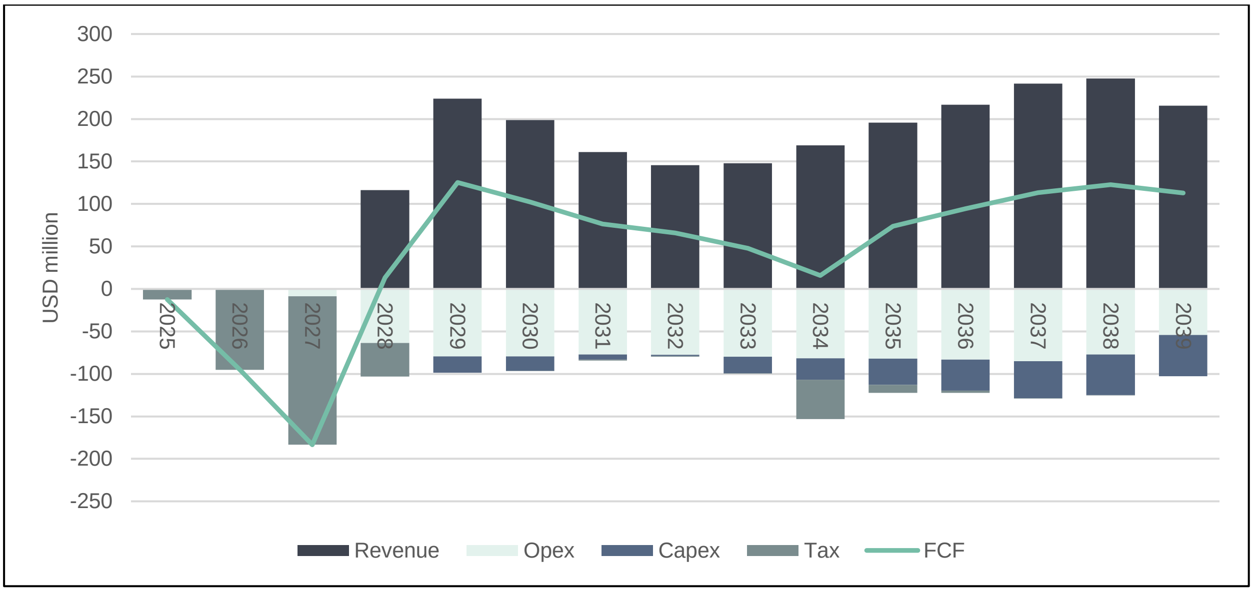

Figure 2-10: Project cash flows

Table 2-7 presents a summary of the financial performance of the Project, including a unit cost analysis.

Table 2-7: Financial performance summary

Item	LOM [USD million]	Product unit [USD/ lb U_3O_8]	ROM unit [USD/t ROM]
Revenue			
U_3O_8 Revenue	**2 279.8**	**90.00**	**57.51**
Opex costs			
Mining	378.5	14.94	9.55
Processing*	379.4	14.98	9.57
Other costs	57.9	2.29	1.46
Royalties	114.2	4.51	2.88
Total Opex	**930.0**	**36.71**	**23.46**
Corporate income tax			
Tax	294.8	11.64	7.44
Capex			
Mining	144.2	5.69	3.64
Processing	193.8	7.65	4.89
Infrastructure	36.7	1.45	0.93
G&A	4.1	0.16	0.10
RAP	3.9	0.15	0.10
Total Capex	**382.6**	**15.10**	**9.65**
Financial performance			
Free cash flow	**672.4**	**26.55**	**16.96**
Net present value @ 8%	**242.6**		
Internal rate of return	**20.8%**		

* Processing cost includes HLF stacking and reclaiming

The sensitivity of NPV to changes in the U_3O_8 price, Capex and Opex is shown in Figure 2-11.

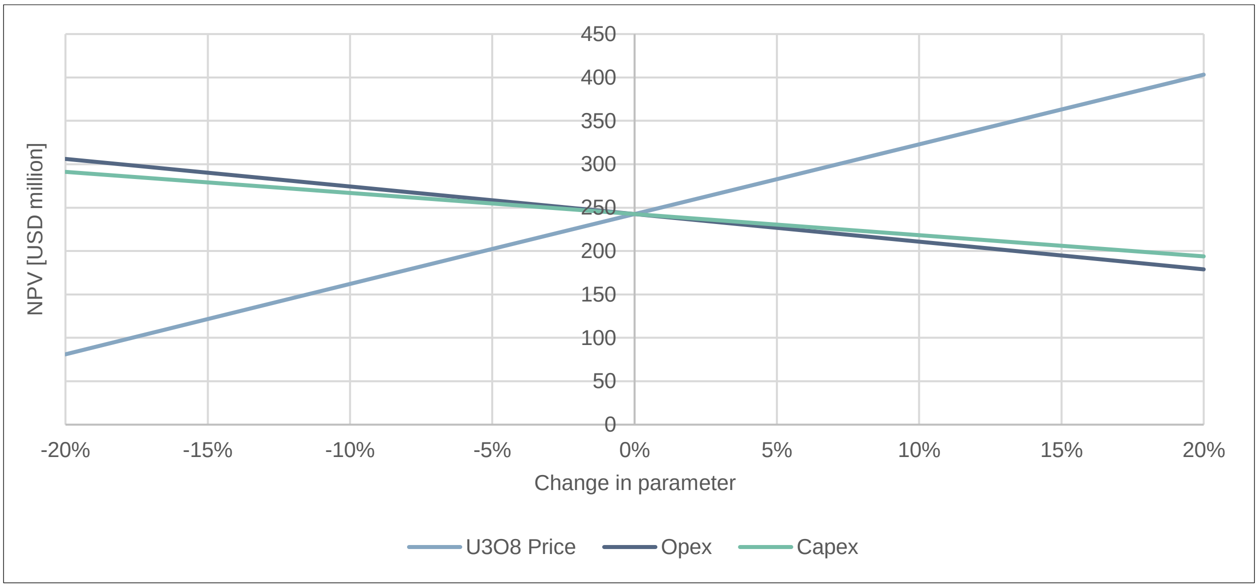

Figure 2-11: NPV sensitivity to changes in U_3O_8 price, Opex and Capex

1.22. Adjacent properties

There are no mining properties immediately adjacent to the Project licences.

1.23. Other relevant data and information

1.23.1. The satellite pits: Dibbwi, Gwabi and Njame

The scope of work for this FS included five pits: the central pits, Muntanga and Dibbwi East; and the satellite pits, Dibbwi, Gwabi and Njame. The FS work was done to the same level for all five pits, including mine schedules, heap leach and processing plants, waste rock and spent ore dumps, infrastructure, power supply, water management, access roads, environmental and resettlement plans. However, the small scale, higher capital requirements and operating costs of the satellite operations relative to the Central pits detracted from the financial performance of the overall project, and the Project was reduced a Central-only operation.

However, under the right market conditions, the satellite pits have the potential to be economically attractive, and the FS work has developed these projects to a point at which they could be implemented. The satellite pits have the following characteristics:

1. The satellites would be mined at 0.5 Mtpa
2. There will be no process plant or HLF at the satellites
3. The satellites would be equipped with a radiometric ore sorting system, “Rados”, which would reject ore at a grade of less than 90 ppm U_3O_8 and reduce the volume of plant feed to 0.35 Mtpa while increasing the feed grade
4. The sorted ore would be trucked with a road-capable side-tipper fleet to the Central plant, where it would be fed into the second crushing stage. The Rados reject will be placed on the WRDs at the satellite pits.
5. Dibbwi would be mined first, given its proximity to Central and hence reduced trucking costs, followed by Gwabi and then Njame

Table 2-8 shows a summary of the LOM production for the satellite pits. A total of 25.0 Mt of material was scheduled, of which 6.5 Mt is ore, at an average grade of 300 ppm. Radiometric sorting by the Rados system reduces this to 4.6 Mt at an increased grade of 408 ppm. After processing in the Central uranium processing and refining plant with an average recovery of 89.1 %, the high-grade ore could yield 3.4 Mt of saleable U_3O_8 product.

The ROM production scheduled from the satellite pits was not included in the Mineral Reserve estimate as part of the Project and did not form part of the Project valuation.

Table 2-8: Satellite pit production summary

Production parameter	Units	Gwabi	Njame	Dibbwi	Total
ROM					
Annual steady-state ROM	Mtpa	0.5	0.5	0.5	**0.5**
Waste	Mt	6.2	11.2	1.0	**18.4**
Ore	**Mt**	**3.4**	**2.3**	**0.9**	**6.5**
Total mined	Mt	9.6	13.5	1.9	**25.0**
Stripping ratio	t:t	1.8	4.9	1.1	**2.8**
ROM ore grade	**ppm U_3O_8**	**322**	**300**	**220**	**300**
Contained U_3O_8	Mlb	2.4	1.5	0.4	**4.3**
Mining duration	years	7.3	5.1	1.8	**13.8**
Rados sorted ore					
Mass pull	%	70	70	70	**70**
U_3O_8 recovery	%	90	90	90	**90**
Annual high-grade ore	Mtpa	0.35	0.35	0.35	**0.35**
High-grade sorted ore volume	**Mt**	**2.4**	**1.6**	**0.6**	**4.6**
Sorted ore grade	**ppm U_3O_8**	**436**	**408**	**298**	**408**
Contained U_3O_8	Mlb	2.3	1.4	0.4	**4.1**
Uranium processing and refining					
U_3O_8 recovery	%	73.1	93.0	92.2	**81.9**
Saleable U_3O_8 product	**Mlb**	**1.7**	**1.3**	**0.4**	**3.4**

At the FS base price of USD 90/ lb, the saleable production volumes from Table 2-8 result in the LOM revenue of USD 303.5 million. Applying the LOM costs to the revenue gives the earnings before interest and tax (“EBIT”) shown in Table 2-9.

The average EBIT margin is 28 %, with both Gwabi and Njame exceeding 30 %. Dibbwi shows a negative margin due to the high Capex relative to ore volume (only 14 % of satellite ROM production). Value engineering approaches such as reducing infrastructure by operating out of Central or trucking directly to Central (as the distance to Central is far shorter than Gwabi and Njame) without implementing Rados can be explored.

Table 2-9: Satellite EBIT

Satellite EBIT	Units	Gwabi	Njame	Dibbwi	Total	USD/t ROM	USD/ lb U_3O_8 product
U_3O_8 product	M lb	1.7	1.3	0.4	**3.4**		
U_3O_8 sales price	USD/lb	90.00	90.00	90.00	**90.00**		
Revenue	USD million	**150.1**	**119.5**	**33.9**	**303.5**	**46.38**	**90.00**
Capex	*USD million*	*-23.7*	*-19.2*	*-18.0*	***-60.9***	*-9.31*	*-18.07*
Opex	*USD million*	*-80.2*	*-59.9*	*-18.9*	***-159.0***	*-24.30*	*-47.16*
Total costs	USD million	**-103.9**	**-79.1**	**-36.9**	**-220.0**	**-33.62**	**-65.23**
FCF before tax	USD million	**46.2**	**40.4**	**-3.1**	**83.5**	**12.76**	**24.77**
FCF before tax margin	%	31	34	-9	**28**	28	28

1.23.2. Inferred material in the mining schedule

Waste material in the mining schedule presented in Figure 2-7 includes mineralised material from Inferred Mineral Resources. As the schedule stands, this material will be blasted, loaded and then hauled to the WRD. At Dibbwi East this mineralised material comprises 5.4 Mt at an average grade of 217 ppm, and at Muntanga 465 kt at an average grade of 283 ppm, giving a total for the Project of 5.8 Mt at a grade of 222 ppm.

Please note that this mineralised material contains Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as Mineral Reserves, and there is no certainty that any value will be realised from them.

1.23.3. Inferred material out of the mining schedule

To investigate the potential contribution to the Project of the Inferred Resources, the Whittle pit shell optimisation was run using the same parameters as the Measured- and Indicated Mineral Resources-only case. This produced a bigger pit shell (the “Inferred pit shell”) which completely encloses that used to develop the selected Mineral Reserve mine schedule.

The resulting additional mineralised material was obtained by subtracting material from the selected pit shell (designed pit) from the Inferred pit shell. This has the potential to bring an additional 6.6 Mt of mineralised material from Indicated and Inferred Mineral Resources (of which 40 % is Indicated and 60 % Inferred Mineral Resources) at an average grade of 278 ppm into the mining schedule.

Please note that this mineralised material contains Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as Mineral Reserves, and there is no certainty that any value will be realised from them.

1.24. Interpretation and conclusions

Ukwazi and SRKs interpretations of the geology, Mineral Resources and feasibility level studies of mining, infrastructure and processing options for the Project are as follows:

- Ukwazi, SRK and SGS have completed technical studies to a feasibility level of confidence for the Project. The Project development plan envisions mining a total of 39.6 Mt of ore at an average grade of 320 ppm U_3O_8, with the average process recovery of 90.5 % yielding a total of 25.3 Mlb of saleable yellowcake product over the 12-year LOM
- Initial capital costs were estimated at USD 282 USD million and sustaining capital at an additional USD 101 million. The total average operating cost (excluding royalties) over the life of the operation was estimated at USD 20.6 /ROM t or USD 32.2 /lb saleable U_3O_8
- A long-term uranium price of USD90 /lb U_3O_8 was applied in the base case financial analysis. The DCF model for the project shows a total LOM net free cash of USD 673 million, which at a discount rate 8 % gives an after-tax NPV of USD 243 million, with an IRR of 20.8%.

The conclusion of this FS is that the Project demonstrates technical and economic feasibility and is in a position to advance to the next stage of project development. A total Mineral Reserve of 39.6 Mt has been stated.

1.25. Recommendations

The following recommendations are provided to advance the understanding of the geology, mineralisation controls, Mineral Resources (and possibly the Mineral Reserves) for the Project:

- Continue development of litho-structural models for the Project deposits, incorporating major fault interpretations within the vicinity of active mine areas or proposed future project infrastructure
- Continue infill drilling to support the conversion of Inferred to Indicated Mineral Resources
- Continue further exploration of other potential orebodies in the Goviex licence areas
- Additional assay sampling to support further refinement of the Ra-grade correlation used to convert down-hole probe data into equivalent uranium grades
- Continue to assess for radon contamination within future drilling programmes and correct down-hole gamma signatures accordingly to mitigate the potential for over-estimation of grade due to radon
- Additional density analysis should be conducted on future drill programmes to refine tonnage estimates.

There are several components of the process design that can be optimised by future testwork.

- The control of iron leaching in the heap, and hence peroxide consumption, can be optimised by recirculating solutions continuously through a number of cycles using small lab columns
- The final product precipitation process can be optimised with respect to impurity deportment, particularly iron. These tests can be done using PLS produced by the small column tests described above, using lime sourced from Zambia
- Finally, rheology work can be done using the gypsum and uranium slurries produced above, to finalise parameters for sizing the various thickening and filtration equipment.

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2. Introduction

2.1. Issuer

This NI 43-101 Technical Report was prepared for GoviEx Uranium Inc., a company registered in British Columbia, Canada, with company registration number C0904287, domiciled at Suite 606, 999 Canada Place, Vancouver, BC, V6C 3E1, and listed on the Toronto Stock Exchange Venture Exchange under the symbol "GXU".

2.2. Terms of reference for and purpose of the technical report

Subsequent to the issuance of the report "NI 43-101 TECHNICAL REPORT ON A PRELIMINARY ECONOMIC ASSESSMENT OF THE MUNTANGA URANIUM PROJECT IN ZAMBIA", effective date November 30, 2017, GoviEx conducted drilling campaigns in the Muntanga project area to develop the Mineral Resources to a sufficient level to allow for a feasibility study ("FS") to be developed and Mineral Reserves to be stated. Subsection 4.2 (1) (j) (ii) of NI 43-101 obliges an issuer to file a TR if "a change in Mineral Resources, Mineral Reserves or the results of a preliminary economic assessment from the most recently filed TR if the change constitutes a material change in relation to the issuer".

Ukwazi Transaction Advisory (Pty) Ltd ("Ukwazi") and SRK Consulting UK Limited ("SRK") were appointed by GoviEx as the qualified persons ("QPs") to conduct the feasibility-level study work required to produce a FS and report with updated Mineral Resource statement, Mineral Reserve statement and a valuation of the project, in accordance with the disclosure and reporting requirements outlined in the Toronto Stock Exchange Manual, National Instrument 43-101 (2011) - Standards of Disclosure for Mineral Projects ("NI 43-101"), Companion Policy 43-101CP to NI 43-101, and Form 43-101F1 of NI 43-101.

Neither Ukwazi nor SRK (including their directors and employees) have or hold:

- Any vested interests in any concessions held by GoviEx, or any adjacent concessions
- Any rights to subscribe to any interests in any of the concessions held by GoviEx either now or in the future or
- Any right to subscribe to any interests or concessions adjacent to those held by GoviEx either now or in the future.

The only financial interest Ukwazi, SRK and other sub-consultants have in the Project is the right to charge professional fees at normal commercial rates, plus normal overhead costs, for work carried out in connection with the investigations reported here. Payment of professional fees was not dependent either on project success or project financing.

2.3. Qualified persons

The QPs responsible for oversight of the technical work and signoff of the FS report are:

1. **Jacobus Johannes Lotheringen**, B Eng (Mining Engineering), South African Institute of Mining and Metallurgy ("SAIMM") – Member (Reg no 701237) and Professional Engineer registered at the Engineering Council of South Africa ("ECSA") (Reg no 20030022). Jaco Lotheringen is employed by Ukwazi as a principal mining engineer. He has had 25 years' experience in mining projects throughout South Africa, Southern Africa, and elsewhere in the world.

2. **Robert J. Bowell,** BSc (Geochemistry), Hons, PhD (Geochemistry), Royal Society of Chemistry – Chartered Chemist (Memb. no 332782), Professional Geologist for the province of Newfoundland and Labrador. (Reg no 10809), Geological Society of London - Chartered Geologist (Reg no 1007245), Institute of Mining, Metallurgy and Materials – Fellow. Robert Bowell is a Principal Geochemist with SRK, with 21 years' experience in applied geochemistry, data analysis and qualification, exploration, exploration management, and mining project evaluation. He has had four years direct experience with uranium exploration, geochemical analysis, mineralogy and evaluation of uranium deposits for project development.

3. **André Marcel Deiss,** B.Sc.(Hons) Geology, registered at the South African Institute of Mining and Metallurgy (SAIMM) - Member Reg. no. 705005, South African Council for Natural Scientific Professions (SACNASP) as Professional Natural Scientist (Geological Science) - Reg. no. 400007/97 and Engineers and Geoscientists of British Columbia as a Professional Geoscientist - P.Geo. Reg. no. 62356, employed by SRK Consulting (Canada) Inc. as an Associate Consultant (Resource Geology).

4. **Alan Mitchell Clegg,** B.Sc. (Mining Engineering), Fellow of the SAIMM (Reg no 701825) and Professional Engineer registered at the ECSA (Reg no 20050117), employed as an independent valuation expert.

Table 2-1 lists the sections of this report and which QP/s were responsible for each, where:

- JL = Jaco Lotheringen
- RB = Robert Bowell
- AD = André Deiss
- AC = Alan Clegg

Table 2-1: Responsibility of QPs for report sections

Section	Item	Responsible QP
1	Summary	All
2	Introduction	JL, RB
3	Reliance on other experts	JL, RB
4	Property description and location	JL, RB
5	Accessibility, climate, local resources, infrastructure and physiography	JL, RB
6	History	JL, RB
7	Geological setting and rnineralisation	AD
8	Deposit types	AD
9	Exploration	AD
10	Drilling	AD
11	Sample preparation, analyses and security	AD
12	Data verification	AD
13	Mineral processing and metallurgical testing	RB
14	Mineral Resource estimates	AD
15	Mineral Reserve estimates	JL
16	Mining methods	JL
17	Project recovery methods	RB
18	Project infrastructure	JL, RB
19	Marketing studies and contracts	JL, RB, AC
20	Environmental studies, permitting, and social or community impact	RB
21	Capital and operating costs	AC
22	Economic analysis	AC
23	Adjacent properties	JL, RB
24	Other relevant data and information	All
25	Interpretation and conclusions	All
26	Recommendations	All

2.4. Sources of information

The sources of information for the FS include:

1. The report “NI 43-101 TECHNICAL REPORT ON A PRELIMINARY ECONOMIC ASSESSMENT OF THE MUNTANGA URANIUM PROJECT IN ZAMBIA”, and the database of all supporting information for this study, including studies, costings, models and reports held by SRK and Goviex
2. The database of drilling information to date for the Muntanga project area, owned and provided by Goviex, was used to develop the geological block model to determine Mineral Resources, Mineral Reserves and the mine schedule
3. Historical project information owned and provided by Goviex
4. Testwork conducted by Mintek on the drilling samples
5. Technical studies, proprietary technical information, databases, capital and operating costs, and other data owned by the contributing consultants for the FS
6. Design, costings, quotes and recommendations from suppliers of mining, uranium processing and other equipment suppliers
7. Relevant mining, environmental, tax and corporate legislation and regulations of the Republic of Zambia

2.5. Property inspections

Mr. Lotheringen, accompanied by Mr. Neil Rossouw (mining engineer) and Dr. Christine Vivier (environmental expert) of Ukwazi, visited the Muntanga project from 22 to 23 April 2024. The purpose of the visit was to experience the topography and flora in the proposed mining areas to inform the mine design and surface infrastructure layout. As part of the visit, they observed drilling, core and drill chip library, sample preparation, and data collection. Ukwazi can confirm that the description of mineralisation, exploration methods, storage and sample information in the reports is a fair reflection of observations made in the field.

Dr. Bowell visited the Chirundu project from 3 to 7 May 2011 as part of a due diligence for a third party. During the visit, he observed drilling, core and drill chip library, sample preparation, and data collection. He can confirm that the description of mineralisation, exploration methods, storage and sample information in reports by African Energy Resources Ltd (“AFR”) as well as their consultants is a fair reflection of observations made in the field. In addition he visited the site for the Muntanga project from 7 to 11 May, 2022 to view all prospects in the FS and examine new core.

Mr. Deiss did not complete a site visit. Mr. Deiss worked directly with Mr. Revering on the project until early 2024 until Mr. Revering was no longer in SRK’s employ. Subsequently, Mr. Deiss took over the role of QP for to ensure the continuation and consistency in the technical work. Mr. Revering who visited the Muntanga project twice in 2022, from May 8 to May 11, and October 17 to October 20. During the site visits, he observed drilling and down-hole logging activities, core and drill chip logging and data collection, and assay sampling and chain of custody protocols. He can confirm that the description of the geology, mineralisation and mineralisation controls, and the drilling, logging, sampling and data collection techniques described are consistent with observations made in the field during these site visits. The QP, Mr. Deiss, is satisfied that the site visits completed by Mr. Revering, while they were both directly engaged on the project, have verified the information upon which the Mineral Resource is based.

3. Reliance on other experts

The QPs from Ukwazi and SRK for this TR, Jaco Lotheringen, Robert Bowell, André Deiss and Alan Clegg, have examined the historical and current data for the Project provided by GoviEx with respect to resources, metallurgical test work, and other project information, and have relied upon that basic data to support the statements and opinions presented in this TR. A full list of documents reviewed is included in Section 27. In the opinion of the authors, the project data is presented in sufficient detail to provide an accurate representation of the Muntanga Uranium Project.

The authors of this report have relied upon the following documents and experts (who are not QPs), and in this regard, the authors disclaim responsibility for the information provided in the following:

- Mintek – Process test results
- SGS Bateman (Pty) Ltd – Process design and engineering
- AMC Africa – Environmental and social studies
- Cresco Global – Financial analysis.

4. Property description and location

The Project is located in the southeastern region of Zambia in the Siavonga and Chirundu Districts and is geographically centred at 16º22'03.31"S, 28º28'51.3"E. The Project comprises three mining licences and two exploration licences, and one recently granted mining licence, with a total combined area of 1 136 km². The three mining licences – Muntanga, Dibbwi and Chirundu – encompass 719 km². The prospect areas extend from Chirundu southwards towards Siavonga and along the northern edge of Lake Kariba to Kariba Valley in the southernmost extent. (Figure 4-1).

The Muntanga and Dibbwi mining licences, which comprise the Muntanga, Dibbwi and Dibbwi East deposits, are located 180 km south of Lusaka, north of Lake Kariba in Siavonga District.

The Chirundu mining licence, which contains the Njame (north and south) and Gwabi uranium deposits, is located close to the town of Chirundu, near the Zimbabwe border and is located approximately 150 km southeast of the Zambian capital, Lusaka. Chisebuka, further south, is approximately 180 km south of Lusaka.

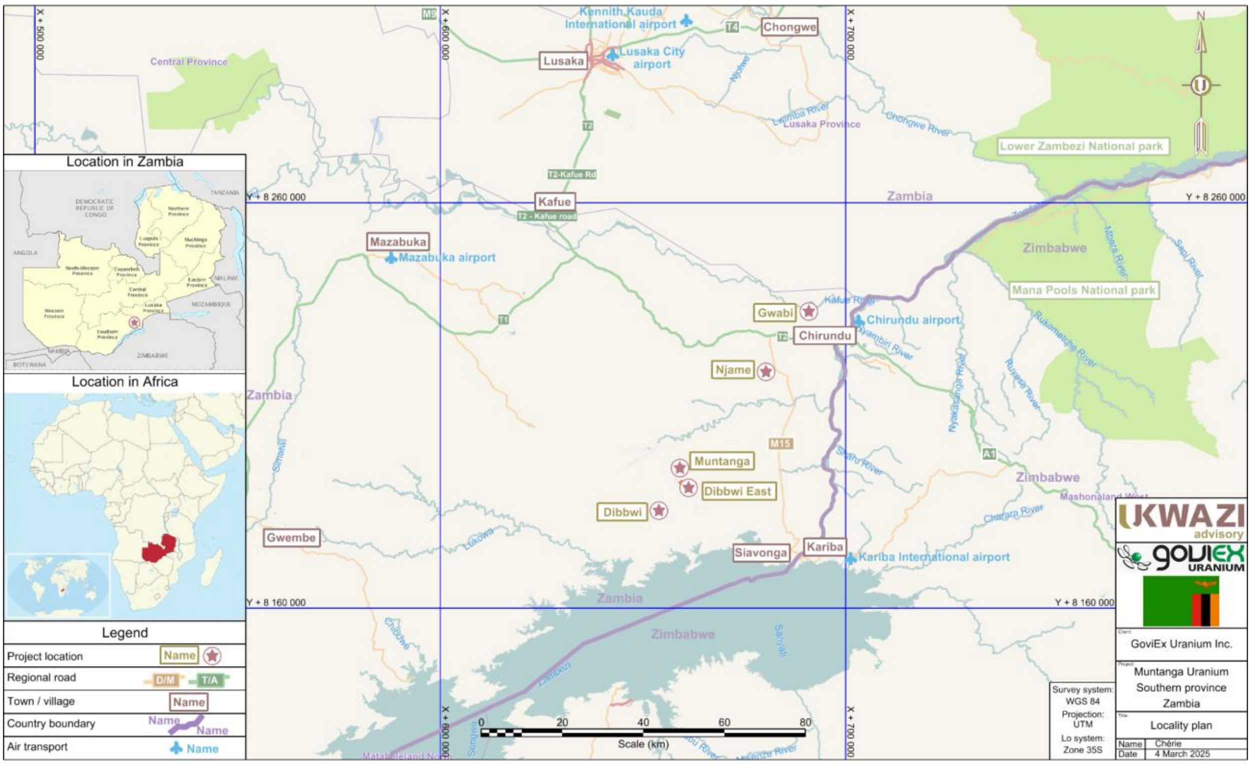

Figure 4-1: Property location map

4.1. Mineral tenure

The Project encompasses three mining licences – Muntanga (Licence no. 13880-HQ-LML), Dibbwi (Licence no. 13881-HQ-LML), and Chirundu (Licence no. 12634-HQ-LML), covering 719 km², (Table 4 1; Figure 4 2) that are located approximately 200 km south of Lusaka, north of Lake Kariba. Additionally, the Company holds two exploration licences for Nabbanda (Licence no. 22803-HQ-LEL) and Chirundu Extension (Licence no 22075-HQ-LEL), and a recently granted mining licence for Kariba Valley (License no. 38555-HQ-LML), which expands the total combined area to 1 136 km². The Mineral Resources reported in this TR are contained within these licences.

100% of the Muntanga and Dibbwi mining licences, which comprise the Muntanga, Dibbwi and Dibbwi East deposits, was acquired by GoviEx in a share purchase agreement from Rockgate Capital Corporation, a wholly-owned subsidiary of Denison Mines Corporation on June 13, 2016. 100 % of the Chirundu mining licence, which contains the Njame (north and south) and Gwabi deposits, and the Kariba Valley (Chisebuka) exploration licence, was acquired from AFR, on October 31, 2017.

The Nabbanda exploration licence, acquired by GoviEx on February 5, 2019, was successfully renewed and approved in 2023. The Chirundu Extension exploration licence, a new GoviEx application, was granted in 2023. In 2024, GoviEx Uranium Zambia Limited applied for the conversion of the Kariba Valley exploration licence to a mining licence. The application has been validated was granted final approval from the Mining Licence Committee in December 2024.

In 2008, the Zambian Government introduced the Mines and Minerals Development Act of 2008, to which all tenements are required to conform. In 2015, the Government repealed the 2008 Act and enacted the current Mines and Minerals Development Act of 2015. according to the Act, exploration licences can have a maximum size of 2 000 km^2 and licence corners must conform to a six-arc-second graticular grid. Each company is allowed a total holding area of 10 000 km^2.

Table 4-1: Muntanga Project Mineral Tenements

Licence Name	Licence number	Area [km^2]	Date First granted	Date expiry	Commodity group	Current status
Muntanga	13880-HQ-LML	233.6	26 March 2010	25 April 2035	Uranium, Coal, Sand, Clay, Gravel and Limestone	Granted
Dibbwi	13881-HQ-LML	237.5	26 March 2010	25 April 2035	Uranium, Coal, Sand, Clay, Gravel and Limestone	Granted
Chirundu	12634-HQ-LML	248.0	09 October 2009	08 October 2034	Uranium	Granted
Chirundu_Ext Nabbanda	22075-HQ-LEL 22803-HQ-LEL	212.9 11.9	18 Jul 2023 5 Feb 2019	17 Jul 2027 Renewal approved	Uranium, Coal Uranium, Coal, Sand, Clay, Gravel and Limestone	Granted
Kariba Valley	38555-HQ-LML	192.2	02 December 2024	01 December 2044	Uranium	Granted

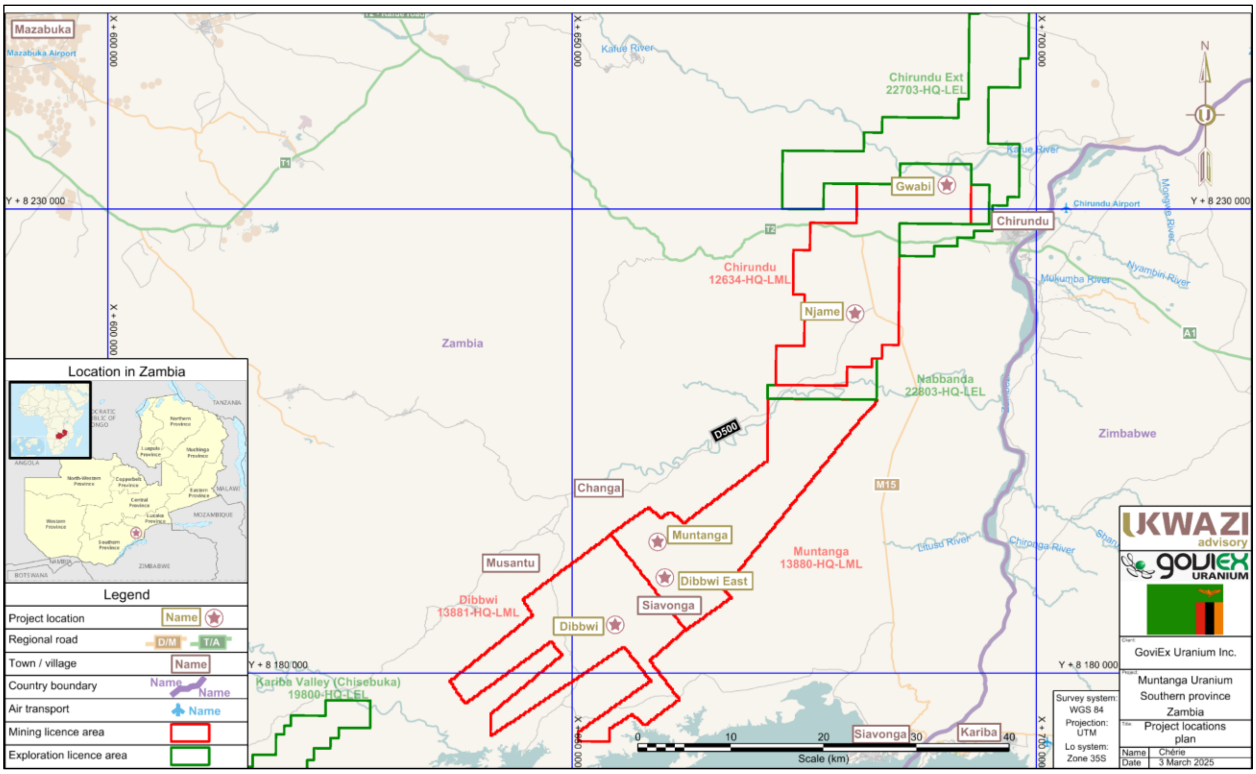

Figure 4-2: The Project site and licence boundaries

4.2. Relevant legislation, permits and approvals

The key legislation with regard to permitting a mining project in Zambia and the applicability and status with regard to the Project are detailed in the following sections.

4.2.1. The Mines and Minerals Development Act 2015

The Mines and Minerals Development Act states that all mineral rights ("MR") are vested in the President of Zambia on behalf of Zambia. This act specifies how the rights to prospect, mine and dispose of minerals can be acquired and held. It confers on the holder exclusive rights to carry on mining and prospecting operations in the mining licence area. This includes erecting the equipment needed to mine, process and transport the minerals, disposal of mining wastes, stockpiling of minerals or waste products and prospecting within the licence area. It gives preference to Zambian products, contractors and services as well as employment of citizens from construction and operation through to decommissioning. Notable sections related to the Project include:

- For the granting of an exploration licence, the following is considered: whether the applicant has the financial resources and technical ability to do the work; if the land is in a protected area, whether the applicant has written consent from the appropriate authority; and the exploration programme makes proper provision for environmental protection
- An exploration licence is valid for four years and can be renewed for two further periods not exceeding three years each. The maximum period from initial grant of the licence shall not exceed ten years. At each renewal, 50% of the exploration licence shall be relinquished. As such, it is understood that the Nabbanda exploration licence has one further renewal and expires in February 2029, and the Chirundu Extension exploration licence has two renewals remaining and will expire in 2033
- Exploration operations can only begin once the holder submits to the Mining Cadastre Office a decision letter in respect of the environmental project brief approved by the Zambia Environmental Management Agency ("ZEMA")
- The holder of an exploration licence can apply, no later than six months before the exploration licence expiry, for a mining licence. A mining licence is required for large-scale mining, and the applicant must meet the following requirements
  - A mine plan, an environmental plan, a financial plan, and a decision letter in respect of the environmental project brief or environmental impact assessment approved by ZEMA
  - A local business development plan and a proposal for the employment and training of citizens of Zambia; and
  - The FS must be bankable
- The environmental plan details the proposals for the prevention of pollution, the treatment of wastes and the rehabilitation of land and water resources. Conditions can be included in the mining right or imposed separately by means of written notice to ensure the protection or conservation of the environment; the rehabilitation of land; the filling in or sealing of excavations, shafts and tunnels; and payment of a cash deposit into an Environmental Protection Fund ("EPF") administered by the EPF Committee appointed by the Minister
- A large-scale mining licence is granted for 25 years and the holder must maintain security and ensure that there are no illegal miners in the licence area, provide an annual audited financial statement to the Mining Cadastre Office, a return showing compliance with obligations, annual mine plans, ore recovery and production costs and produce ore resource and reserve statements every two years
- A mineral processing licence is required for mineral processing activities. However, the holder of a mining licence may construct and operate a mineral processing plant within their licence area without a mineral processing licence
- For the export of minerals, a mineral export permit issued by the Director of Mines is required. This is valid for one year and is limited to the quantities specified in the permit. For radioactive minerals, the applicant must comply with the requirements of the Ionising Radiation Protection Act 2005. GoviEx will comply with the requirements of the act and apply for an export permit for the uranium product as the project progresses
- Storage, transport, or mining of radioactive minerals must also be done in accordance with the provisions of the Ionising Radiation Protection Act 2005. This requires a licence issued by the Radiation Protection Authority which GoviEx will apply for as the project progresses
- In terms of surface rights, the holder of a mining licence shall not mine at a dedicated place of burial, land containing monuments defined in the National Heritage Conservation Commission Act, or land within 90m of any building or dam owned by the State without written consent from the appropriate authority. In addition, the licence holder requires written consent of the owner or legal occupier of land within 180m of an inhabited, occupied or temporarily uninhabited house, within 45m of land used to farm crops, within 90m of any cattle dip tank, dam or private water as defined by the Water Resources Management Act 2011, upon land occupied by a village or other land under customary tenure without written consent of the chief or any land in a protected area without complying with the Zambia Wildlife Act 2015. The holder of the mining right who requires the exclusive use of the exploration or mining area may acquire a lease of the land or other right to use the land by agreeing on terms with the landowner or occupier. GoviEx presently has no surface rights over the project area. GoviEx intends to secure the required surface rights as part of the resettlement planning and permitting process that will accompany the FS and Environmental and Social Impact Assessment ("ESIA"). The process of obtaining surface rights in Zambia requires applicants to apply to the Ministry of Lands and in the case of traditional land, GoviEx will obtain approval and recommendation from the traditional leaders and local councils of Siavonga and Chirundu.

4.2.2. Water Resources Management Act 2011

The Water Resources Management Act established the Water Resources Management Authority ("WRMA") and defined its function and powers. The Act provides for the protection of Zambia's water resources and that the said resources should be used, developed, conserved, managed and controlled sustainably, beneficially, reasonably and equitably for the needs of the present and future generations. It provides for the management, development and utilisation of water resources to take into account climate change adaptation.

4.2.3. Ionising Radiation Protection Act 2005

The Ionising Radiation Protection Act established the Radiation Protection Authority's functions and powers, and provides for the protection of the public, workers and the environment from hazards related to ionising radiation or the release of radioactive material.

4.2.4. Zambia Wildlife Act 2015.

This act makes provision for the management and conservation of wildlife in Zambia. It provides for the implementation of the Convention on International Trade in Endangered Species of Wild Fauna and Flora, the Convention on Wetlands of International Importance especially as Waterfowl Habitat, the Convention on Biological Diversity, the Lusaka Agreement on Cooperative Enforcement Operations Directed at Illegal Trade in Wild Fauna and Flora and other international instruments to which Zambia is party. The Zambia Wildlife Regulations 2016 and Zambia Wildlife Order 2016 implement it. As the Chirundu Extension exploration licence is within the Chiawa Game Management Area, Part IV of the act is of specific relevance as it relates to mining rights within game management areas. Mining can occur in this area if prior written notice is given to the Director of National Parks and Wildlife and subject to compliance with any conditions the Minister may impose. These may relate to measures specified under an Environmental Impact Assessment ("EIA") approved by ZEMA.

4.2.5. Environmental Management Act 2011

The Environmental Management Act ("EMA") is the principal piece of legislation governing environmental management in Zambia. ZEMA is mandated to ensure the sustainable management of natural resources and protection of the environment, and the prevention and control of pollution. The EMA provides for public participation in environmental decision-making and access to environmental information. In particular, section 29 of the Act states that "A person shall not undertake any project that may have an effect on the environment without the written approval of the Agency, and except in accordance with any conditions imposed in that approval". The Act provides specific regulations for pollution control, water, air, waste management, pesticides and toxic substances, noise, ionizing radiation and natural resources management.

GoviEx currently holds a licence for the management of hazardous waste, details of which are included in Table 4-2.

Table 4-2: Summary of GoviEx ZEMA licences

Permit	Date awarded	Duration	Expiry date
Hazardous waste licence	9 August 2022	3 years	8 August 2025

Prior to commencing mining operations, other licences granted by ZEMA that must be applied for include, but are not limited to:

- Air pollution monitoring permits
- Water effluent and discharge licences
- Waste management licence.

A summary of the relevant regulations and their subsidiary statutory instruments ("SI") is shown in Table 4-3.

Table 4-3: Zambian Regulations (Source MDM, 2009)

Institution of Legislation	Act	Regulations
Mining	The Mines and Minerals Development Act (Act No. 11 of 2015)	The Mines and Minerals (Environmental) Regulations (SI No. 29 of 1997) The Mines and Minerals Development (Prospecting, Mining and Milling of Uranium Ores and Other Radioactive Mineral Ores) Regulations, 2008 (SI No. 7 of 2008) Mines and Minerals (Environmental Protection Fund) Regulations (SI No. 102 of 1998)
Environment	The Environmental Management Act No 12 of 2011	The Environmental Protection and Pollution Control (Environmental Impact Assessment) Regulations (SI No. 28 of 1997) Waste Management (Licensing of Transporters of Wastes and Waste Disposal Sites) Regulations (SI No.71 of 1993) Hazardous Waste Management Regulations (SI No. 125 of 2001) Water Pollution Control (Effluent & Waste Water) Regulations (SI No. 72 of 1993) Pesticides and Toxic Substances Regulations (SI No. 20 of 1994) Air Pollution Control (Licensing and Emission Standards) Regulations (SI No. 141 of 1996)
Ionising Radiation	The Ionizing Radiation Protection Act, 2005 (SI No. 16 of 2005)	
Energy	The Energy Regulation Act, No 12 of 2019 The Electricity Act, No 11 of 2019 The Petroleum Exploration and Production Act, No 10 of 2008	
Wildlife and National Heritage	The National Heritage Conservation Commission Act, 1989 (SI No 23 of 1989) The National Parks and Wildlife Act, 1991 (SI No. 10 of 1991) The Zambia Wildlife Act 2015(SI No. of 1998) The Pneumoconiosis Act (SI No. 124 of 1965 and amendments) The Forests Act No 4 of 2015	
Health	The Pneumoconiosis Act (SI No. 124 of 1965 and amendments) Public Health Act CAP 295	
Employment	The Employment Code Act No 3 of 2019;	
Road Transport	Roads and Road Traffic Act (Act No. 11 of 2002 and all amendments)	
Taxes	The Zambia Revenue Authority Act (SI No. 28 of 1993 and all amendments) Customs and Excise Act No 45 of 2021; and all amendments and subsidiary legislation) Value Added Tax Act (SI No. 4 of 1995 and all amendments)	

4.2.6. International agreements

The Republic of Zambia is a member of 44 international organisations, one of which is the International Atomic Energy Agency ("IAEA"). Some of the commitments made through these organisations are:

- The Rio Convention on Biological Diversity
- The official Convention on Climate Change signed in Rio
- The Climate Change Kyoto Protocol
- International Convention on Desertification
- The Ramsar Convention related to the Wetlands of International Importance and particularly recognised as habitats for wilderness
- The International Convention for the Protection of Fauna and Flora in Africa
- The Convention on Endangered Species
- International Convention on the Protection of the Ozone Layer
- The International Convention on Hazardous Wastes
- The Law of the Sea and
- The United Nations Framework Convention on Climate Change.

4.2.7. Royalties and agreements

The licences are wholly owned (100 %) by GoviEx Uranium Inc through its local subsidiaries GoviEx Uranium Zambia Ltd, Chirundu JV Ltd and Muchinga Energy Ltd. There are no agreements or encumbrances on the permits currently held by GoviEx or its subsidiaries.

4.3. Environmental liabilities

The Project is a greenfield exploration site with no history of previous development or industrial activity. As a result, there are no obvious current environmental liabilities. Should the Project be implemented and mining operations commence, environmental liabilities to decommission infrastructure rehabilitate disturbed areas and manage long-term effects will be managed in line with the conceptual Closure Plan. A summary of the closure planning activities and associated costs is included in Section 20.8.

GoviEx has established a permanent exploration camp immediately adjacent to the Muntanga deposit. Should the project not progress to an active operating mine, the camp will have to be closed, and any uranium-bearing sample material appropriately disposed. It is probable local communities could use the camp infrastructure.

4.4. Qualified person comment

The QPs do not know of any significant factors or risks affecting access, title or the right or ability to perform work on the property.

5. Accessibility, climate, local resources, infrastructure and physiography

5.1. Topography, elevation and vegetation

The Project area is located within the Zambezi Rift System in southern Zambia. The Zambezi River flows to the east of the area, following the border between Zambia, Zimbabwe and Mozambique.

Surface runoff is predominantly contour-controlled but occasionally fault-controlled. Lake Kariba is situated at 485 m above mean sea level and the Project region varies between 500 m and 960 m above sea level.

Vegetation typically consists of forest, which is predominantly miombo woodland mixed with munga and mopane, but there are also small areas of agricultural fields and degraded grassland. The dominant vegetation is as follows:

- Commiphora – Kirkia thicket on lower Karoo sands. Frequently occurs as lake basin chipya, semi-evergreen thicket or termite moulds
- Colophospermum mopane woodland on heavy clay soils. Dominant vegetation type that is frequently pure or almost pure in mopane woodlands, mopane munga and mopane miombo. It occurs on munga and mopane termitaria in deciduous thicket
- Southern Isoberlinia – Brachystegia woodland on escarpment soils. Highly favoured for fuelwood production, especially charcoal
- Acacia woodland on clay soils. Vegetation that favours dry areas; is important for soil improvement, livestock and game, gum exudation, timber and traditional medicine.

The wild bushland experiences only minor disturbance including dry season fires, human cutting for building materials or fuel, and human clearing for agriculture, grazing or settlements.

5.2. Access to property

Proximity to Chirundu and Siavonga means that the area is relatively well-serviced with sealed roads and numerous gravel tracks, which lead to farms and villages.

Access to the Project is by the sealed main road running between Chirundu and Lusaka and the sealed road to Siavonga, then turning onto the sealed road leading to Munyumbwe, in Gwembe District. The main roads are in fairly good condition, but the actual Project area is located east of the main roads and accessed via gravel roads that require a four-wheel drive vehicle. The nearest commercial airport is in Lusaka, located 144 km by road from Chirundu.

5.3. Climate

The Project has a climate described as tropical wet and dry, with very distinct wet and dry seasons. Meteorological information is obtained from the nearest station at Lusitu, approximately 40 km north-east of Muntanga with a similar elevation and climate. The meteorological station operated from 1995 to 2005 and since 2005 weather data has been measured at the site but is not considered to be sufficiently reliable.

Annual rainfall is recorded as between 600 mm and 720 mm, and the wet season occurs in the hottest summer months between November and March. The highest rainfall generally occurs in January/ February. Maximum temperatures range from 22 °C to 46 °C and minimum temperatures range from 20 °C to 38 °C during the hottest months. The highest temperatures typically occur just prior to the onset of the rains in October. Wind speeds are greatest during this period and can range from approximately 2.5 ms^{-1} to approximately 3.6 ms^{-1}, typically from an east-southeast direction. Lightning storms can be common during the hottest months and occasionally hailstones are experienced, associated with thunderstorms. During the wettest months of October to February, the average daily sunshine hours can range from only 4.6 hours (February) to 8.8 hours (October).

During the cooler months of April to October, rainfall varies significantly spatially and temporally. Maximum temperatures range from 23 °C to 40 °C and minimum temperatures range from 6 °C to 28 °C, with lowest temperatures occurring in June and July. Winds are typically much calmer during the colder, dry months, particularly between April and August. On average, at least nine hours of daily sunshine is generally received during the drier months of May to September.

The highest maximum temperature recorded at the Project site was 46 °C and the lowest minimum temperature that has been recorded is 6 °C. Evaporation typically exceeds precipitation for most of the year. Monthly relative humidity generally ranges from a minimum of 46 % in September to a maximum of 79 % in December.

Weather data taken from Lusaka airport and corrected for the altitude difference at the Project site indicates that the mean station level barometric pressure for Muntanga is 951 hectopascal (“hPa”).

5.4. Local resources

There are many small villages located around the Project area and approximately 10 % of the land is used for small-scale agriculture including millet and maize, sorghum, bananas, cotton and minimal animal husbandry. There are currently no industrial activities within the Project area.

According to the United States Department of Agriculture), the regional land classification indicates medium to low potential for sustainable development based upon extremely weathered and iron-rich soils. The soils are typically nutrient-deficient and not good at retaining water although they are easily worked.

5.5. Infrastructure

Except for the main road systems described in Section 5.2, there is limited to no infrastructure within the immediate Project area.

5.5.1. Roads

As described in Section 5.2, there are some sealed roads in the area which run between Lusaka, Chirundu, Siavonga and the bottom road to Munyumbwe in Gwembe District. Although they are in fairly good condition, access to the actual Project site is gravel tracks which require four-wheel drive access. Local communities rely on bicycles or carts for transport.

5.5.2. Power supply

There are two 88 kV/11 kV substations located at Gotagota and Chirundu, both supplied by an 88 kV transmission line from the bigger Leopards Hill Sub Station, which is supplied via 330 kV high voltage transmission lines from the Kariba North Bank Hydroelectricity Scheme. Power lines do traverse the Project area around Njame. Most of the local villages are not connected to the national power network and households near Muntanga and Dibbwi rely on wood for heating and cooking, and candles and kerosene lamps for lighting.

5.5.3. Local villages and towns

The region is sparsely populated: Chirundu, Siavonga, Kafue and Lusaka are the closest major urban areas. Lusaka has a population of 3.2 million (2023). Siavonga and Chirundu are small towns with local government and town council administration offices. The two towns have banking facilities, a post office, district hospitals and general stores. There are no defined commercial areas within the immediate vicinity of the Project and grocery stores are typically located along the sealed roads to Chirundu and Siavonga. The rural areas are administered by four Chiefdoms, which include Chief Simamba, Chief Sikoongo, Chief Munyumbwe and Chief Sinadambwe. Much of the housing in the villages is typically wooden structures covered with mud. Communities are predominantly rural, mostly seasonal peasant farmers producing maize, cotton, millet, sorghum and vegetables; the majority of crops grown are for household consumption. Charcoal is also produced for sale and used as a main fuel source alongside wood, for heating and cooking.

5.5.4. Water supply and sanitation

The Project area relies on wells and boreholes for potable. Sanitation is managed by way of pit latrines in some households. The Southern Water and Sewerage Company has a treatment plant located on the Zambezi River that supplies piped water to Siavonga, but this does not reach the Project site. GoviEx has provided 15 water boreholes to local villages.

5.5.5. Education and health care facilities

There are very few schools and health facilities in the Project area and typically they have insufficient staff and resources. The main challenges faced are long distances, poor staffing levels, inadequate funding and transport.

The development of local health and school facilities through sustainable development projects carried out by the Project will benefit the local communities. To date, GoviEx, through its corporate social responsibility programme, has provided clinics for the villages of Muntanga, Sikoongo, and Chizilika, and Nurses' houses at Muntanga, Chizilika and Syamwiinga. Temporary schools have been constructed at Muntanga and Mutuba to help the local population have access to education near their locality. These temporary schools will be demolished when the mine proceeds into development and rebuilt in the new relocation areas. The company pioneered the development of schools at Hachibozu, Chizilika and Njame villages by constructing classroom blocks. At Chaanga, two laboratory classrooms were built, leading to the upgrading of the school from the primary to the secondary level. Staff houses for teachers have been constructed by GoviEx at Hachibozu, Chizilika and Muntanga. At Dibbwi Primary School, GoviEx recently funded the rehabilitation of a 1 x 2 classroom block.

GoviEx has also supported these sectors with the provision of solar power to facilitate the delivery of ICT lessons at the Hachibozu and Muntanga schools. At Muntanga clinic, GoviEx has provided a solar system for refrigeration of medical supplies.

Piped water, using solar systems, has been supplied to Hachibozu School and village, and Muntanga Clinic. Muntanga Clinic shares the facility with the adjacent Muntanga School.

In addition, GoviEx is supporting three educational support programmes, namely:

- The Back to School Project is an adult education initiative run in partnership with the District Education Board Secretaries for the Siavonga and Chirundu Districts. It focuses on providing educational opportunities for adults who may not have had previous access to formal education
- The Trainee Programme funds the tuition, boarding and upkeep of an initial six students from three communities in the areas around the Project. The students started three-year courses in auto-mechanics, power electrical and plumbing at the Lusaka Vocational Technical College (“LVTC”) in May 2023. This is in addition to three graduate community health assistants that GoviEx sponsored at Mwachisompola College of Health Sciences in Chibombo
- The Skills Training Programme is an initiative piloted in conjunction with the LVTC to provide fast-track skills education to local eligible students. The company is exploring the possibility of establishing a Skills Training Centre along the Bottom Road (D500) for ease of access, including beyond the life of mine (“LOM”).

All the above educational programmes are meant to support GoviEx’s policy of local employment and development. GoviEx wants to create of pool of local skilled labour it can tap from as the project goes into mine development.

5.5.6. Telecommunications

Telecommunications are provided to the Muntanga area by Airtel, MTN and Zamtel. Airtel and MTN provide fourth-generation wireless (“4G”) services for internet connectivity.

5.6. Physiography

The topography is defined by geology and consists of gentle, low escarpment-type hills with steep and/or craggy scarp northwest slopes and gently sloping southeast dip slopes.

6. History

6.1. Introduction

Uranium was first identified in the area in 1957 by ground survey which located five anomalous areas in the vicinity of Bungua Hill, west of Siavonga. In 1958 and 1959 Chartered Exploration found low-grade uranium mineralisation that could be followed for over 800 m of strike extent.

The main exploration took place between the late 1970s and mid 1980s initially by the Geological Survey of Zambia ("GSZ"), followed by AGIP SpA ("AGIP"), an Italian petroleum company. The AGIP exploration campaign included a regional ground radiometric surveying programme which highlighted numerous radiometric anomalies along the northern shores of Lake Kariba including Dibbwi and Chisebuka. Several of the anomalies were investigated via more detailed ground radiometric surveying and subsequent drilling. Their campaign predominantly focused on the Muntanga and Dibbwi deposits, and in 1983/4 a small uneconomic resource was outlined at Njame but AGIP ceased work in 1985.

6.2. Property ownership and exploration activity: Dibbwi East, Dibbwi and Muntanga

Known prior ownership and work undertaken in the Muntanga area are summarised below:

- Owner unknown – 1957: Ground survey located five anomalous areas in the vicinity of Bungua Hill, west of Siavonga
- Chartered Exploration – 1958 and 1959: Found low-grade uranium mineralisation that could be followed for over 800 m of strike extent
- Chartered Exploration – 1974: Confirmation of this uranium mineralisation was further defined in two campaigns after regional airborne magnetic and radiometric surveys had been flown over the area by Geometrics
- GSZ – 1973 to 1977: Ground investigation
- AGIP – 1974 to 1984: Exploration ground campaign, including investigation of the Muntanga and Dibbwi uranium deposits
- Period of inactivity – 1984 to 2004
- Okorusu Fluorspar Pty Ltd – 2004 to 2006: Exploration unknown
- OmegaCorp Minerals Limited acquired Okorusu Fluorspar exploration licence – 2006: 11 holes (649 m) at the Muntanga mineral deposit to confirm the uranium deposit identified by AGIP
- Denison acquired OmegaCorp Limited in August 2007. Denison is a publicly owned, uranium exploration and development company listed on the Toronto (Canada) and NYSE MKT. OmegaCorp became a wholly owned subsidiary of Denison
- The prospecting licences were converted to two mining licences in 2010 that were held by Denison's wholly owned subsidiary Denison Mines Zambia Limited
- GoviEx acquired Denison Mines Zambia Limited in June 2016.

6.2.1. Historical Mineral Resource estimates

Numerous historical Mineral Resource Estimates ("MRE") have been prepared by a variety of companies and consultants using several different methodologies. Considering the successive exploration drilling completed at the project, all estimates, in general, compare favourably and demonstrate similar U_3O_8 grades and tonnages.

A summary of the historical MREs is provided in Table 6-1 from the 1970s through to 2012. Table 6-3 provides a summary of the most recent historical Mineral Resources as at September 12, 2013. SRK does not consider the historical estimates to be relevant or reliable, as additional drilling and data analysis have been completed as part of the 2021 and 2022 work campaigns. The QP has not completed sufficient work to classify the historical estimates as current Mineral Resources and as such GoviEx is not treating these estimates as current.

Table 6-1: Historical Muntanga MREs

Company Name / Year of MRE	Category	Cut-off [ppm U_3O_8]	Tonnes [Mt]	Grade [ppm U_3O_8]	U_3O_8 [Mlbs]
AGIP (1970s)	Unclassified*	700	2.40	1 000	5.30
AGIP (1970s)	Unclassified*	600	3.20	870	6.10
AGIP (1970s)	Unclassified*	500	4.30	740	7.00
AGIP (1970s)	Unclassified*	400	4.90	600	6.50
AGIP (1970s)	Unclassified*	300	7.80	530	9.10
AGIP (1970s)	Unclassified*	200	9.70	480	10.30
CRM Apr 2005 (Muntanga)	Unclassified*	200	7.00	400	6.20
CRM Apr 2005	Unclassified*	200	0.90	400	0.80
CRM Nov 2005 (Muntanga)		200	6.50	375	5.40
Muntanga East	Unclassified*	200	0.30	400	0.29
Muntanga West	Unclassified*	200	0.65	350	0.53
Dibbwi	Unclassified*	200	5.00	430	4.70
Total			**12.45**	**396**	**10.92**
CSA (June 2006)					
Muntanga	Inferred**	200	7.00	400	6.20
Dibbwi	Inferred**	200	8.20	370	6.60
Total			**16.40**	**380**	**13.70**
Denison-RPA (March 2012)					
Dibbwi East	Inferred	100	39.8	322	28.27

* Reported internally only, unclassified under CIM
** Reported to JORC (2004)

Table 6-2: CSA 2013 Summary Mineral Resources (Source: (CSA, 2013))

CIM compliant Mineral Resource inventory – the Project [as at September 12, 2013]										
Deposit	U_3O_8 lower cut-off	Measured			Indicated			Inferred		
		Tonnes [Mt]	U_3O_8 [ppm]	U_3O_8 [Mlbs]	Tonnes [Mt]	U_3O_8 [ppm]	U_3O_8 [Mlbs]	Tonnes [Mt]	U_3O_8 [ppm]	U_3O_8 [Mlbs]
Muntanga	100	1.88	481	2.0	8.40	314	5.8	7.20	206	3.3
Muntanga Extensions	200	-	-	-	-	-	-	0.50	340	0.4
Muntanga East	200	-	-	-	-	-	-	0.20	320	0.1
Muntanga West	200	-	-	-	-	-	-	0.50	340	0.4
Dibbwi	100	-	-	-	-	-	-	17.00	234	9.0
Dibbwi East	100	-	-	-	-	-	-	39.80	322	28.2
Total		**1.88**	**481**	**2.0**	**8.40**	**314**	**5.8**	**65.20**	**287**	**41.4**

6.3. Property ownership and exploration activity: Gwabi and Njame

The earliest known exploration for uranium occurred in the late 1970s to the mid-1980s as part of the AGIP campaign. AGIP ceased its work in Zambia in 1985, and no further exploration for uranium was undertaken in this area until AFR commenced work in 2005.

In October 2005, Albidon Exploration Limited signed a joint venture agreement with AFR for them to explore the eastern part of the Mugoto PLLS250 tenement that had been previously acquired by Albidon as part of their Munali nickel project tenement holding. The area under exploration by AFR was named the Chirundu Uranium JV and covered the Gwabi and Njame deposits.

In 2006 and 2007 AFR carried out a major exploration programme at the Chirundu site and a pre-feasibility study ("PFS") to evaluate the commercial viability of mining and processing uranium ores at Njame and Gwabi was undertaken from 2007 to 2008. Drilling at the Njame deposit led to the delineation of an Inferred Mineral Resource that was larger than the one initially identified by AGIP, and an airborne radiometric survey conducted at Gwabi revealed a significant uranium anomaly that was subsequently investigated by surface radiometric surveying and soil sampling and outlined as an Inferred Mineral Resource. In March 2008 AFRs equity was increased to 70 % when the

PFS reported an Indicated Mineral Resource, and this was subsequently increased to a 100 % interest in the Chirundu and Kariba Valley Projects in March 2011.

In October 2017 GoviEx acquired the Chirundu and Kariba Valley Projects from AFR.

6.3.1. Historical Mineral Resource Estimates

An MRE for the Njame and Gwabi deposits and the Chirundu Project as a whole (now part of the Project) was conducted in 2009 (Table 6-3). GoviEx is not treating the estimate as current because additional work has been undertaken as detailed in Section 14.6.

Table 6-3: Historical MRE, AFR Projects (Source: AFR, 2009)

Deposit	Mineral Resources							
	Measured		Indicated		Inferred		Contained U_3O_8	
	Tonnes [Mt]	Grade [ppm U_3O_8]	Tonnes [Mt]	Grade [ppm U_3O_8]	Tonnes [Mt]	Grade [ppm U_3O_8]	Tonnes	Mlb
Njame North	2.7	350	2.2	252	1.5	223	1 815	4.0
Njame East	-	-	0.6	291	0.5	233	305	0.7
Njame Central	-	-	0.9	222	0.2	219	240	0.5
Njame South	-	-	-	-	4.4	237	1 040	2.3
Njame total	**2.7**	**350**	**3.7**	**252**	**6.6**	**233**	**3 400**	**7.5**
Gwabi total	**1.3**	**237**	**3.6**	**313**	**0.8**	**178**	**1 575**	**3.5**
Chirundu project total	**4.0**	**313**	**7.3**	**282**	**7.4**	**227**	**4 975**	**11.0**

Note: All reported using a 100 ppm U_3O_8 cut-off grade envelope with appropriate rounding applied
AFR JORC accredited resource statement as of November 18, 2009 (AFR, 2009)

6.3.2. Kariba Valley (Chisebuka)

Radiometric anomalies were previously identified in the Kariba Valley area by AGIP, but very limited follow-up exploration was undertaken.

AFR and Albidon Exploration established a second joint venture, the Kariba Valley JV which contained the Chisebuka and Namakande prospects. AFR had an initial 30 % equitable interest which was later increased to 100 % holding. Their investigations included ground radiometric surveys, geochemical assessments of soil and rock-chip plus RC percussion drilling which revealed significant uranium mineralisation at Chisebuka and Namakande.

6.4. Production history

There has been no uranium production from any of the Project licence areas.

7. Geological setting and mineralisation

7.1. Regional geology

The Project area is situated within the Karoo Supergroup, which comprises thick, carboniferous to late Triassic age, terrestrial sedimentary strata and is widespread across much of what is now southern Africa. The Karoo Supergroup was deposited within an extensive foreland basin created when compression and accretion along the southern margin of Gondwana resulted in the formation of the Cape Fold Belt to the south. To the north, crustal extension due to thermal doming following the assembly of the Pangean supercontinent around 320 million years ago, resulted in the formation of a northeasterly trending series of rift basins (Yeo, 2010). The rifting is believed to have been associated with the breakup of Gondwanaland during the Permian Period, followed by the opening of the proto-Indian Ocean in the Jurassic; with a final episode related to the development of the East African Rift system in the late Cretaceous and early Tertiary times.

During the Cenozoic, the East African Rift System propagated south-westerly across the continent and led to the reactivation of the Karoo rift basins as well as the formation of new fault depressions, such as the Okavango Rift (Laletsang et al., 2007; Kinabo et al., 2007), the southeastern extension of the mid-Zambezi and Luangwa rift systems.

The Karoo Supergroup in the Project area consists of three formations within the Lower Karoo; the Siankondobo Sandstone Formation, overlain by the Gwembe Coal Formation, which itself is overlain by the Madumabisa Mudstone Formation. The stratigraphy of the Karoo Supergroup in southern Zambia is shown in Figure 7-1. The Siankondobo Sandstone Formation consists of fine clastic sediments with a basal diamictite and conglomerate overlain by siltstones and sandstones. The Gwembe Coal Formation is comprised of carbonaceous mudstones and siltstones interspersed with coal seams and sandstones, while the Madumabisa Mudstone Formation consists of a thick sequence of non-carbonaceous grey mudstones with calcareous bands. The Madumabisa Formation is unconformably overlain by the Upper Karoo which consists of four formations; the Escarpment Grit is overlain by the Interbedded Sandstone and Mudstone Formation, followed by Red Sandstone which is finally capped by the Jurassic Bakota Basalt Formation (Figure 7-1). The Escarpment Grit comprises a 400m thick series of continental arenaceous silici-clastic sediments with interbedded mudstones. Although locally referred to as Escarpment Grits, this group is a correlative of the Beaufort Group elsewhere in the Karoo Supergroup and contains interbedded mudstones and fine-grained sandstones, as well as grits and conglomerates.

The Project is situated in the mid-Zambezi Rift Valley. In the region, known uranium mineralisation typically occurs within the Upper Karoo whereas the Lower Karoo hosts much of the coal reserves of Zambia, Zimbabwe and South Africa. At the Project, all of the known uranium mineralisation occurs within the Escarpment Grit. Similar sandstone-hosted uranium mineral deposits occur in many of the Karoo rift basins including Letlhakane in the Kalahari Basin of Botswana and Kayelekera in the Rukuru Basin of Malawi (Figure 4-1). The underlying Madumabisa Mudstone appears to have acted as an impermeable barrier controlling the base of the mineralisation. The Escarpment Grit itself shows a wide variation in lithology which is typical of continental sediments. Uranium mineralisation appears to have been introduced after sedimentation (epigenetic) and occurs as fillings into pore spaces, fractures, joints, coatings on sand grains and occasionally along steeply dipping cross beds.

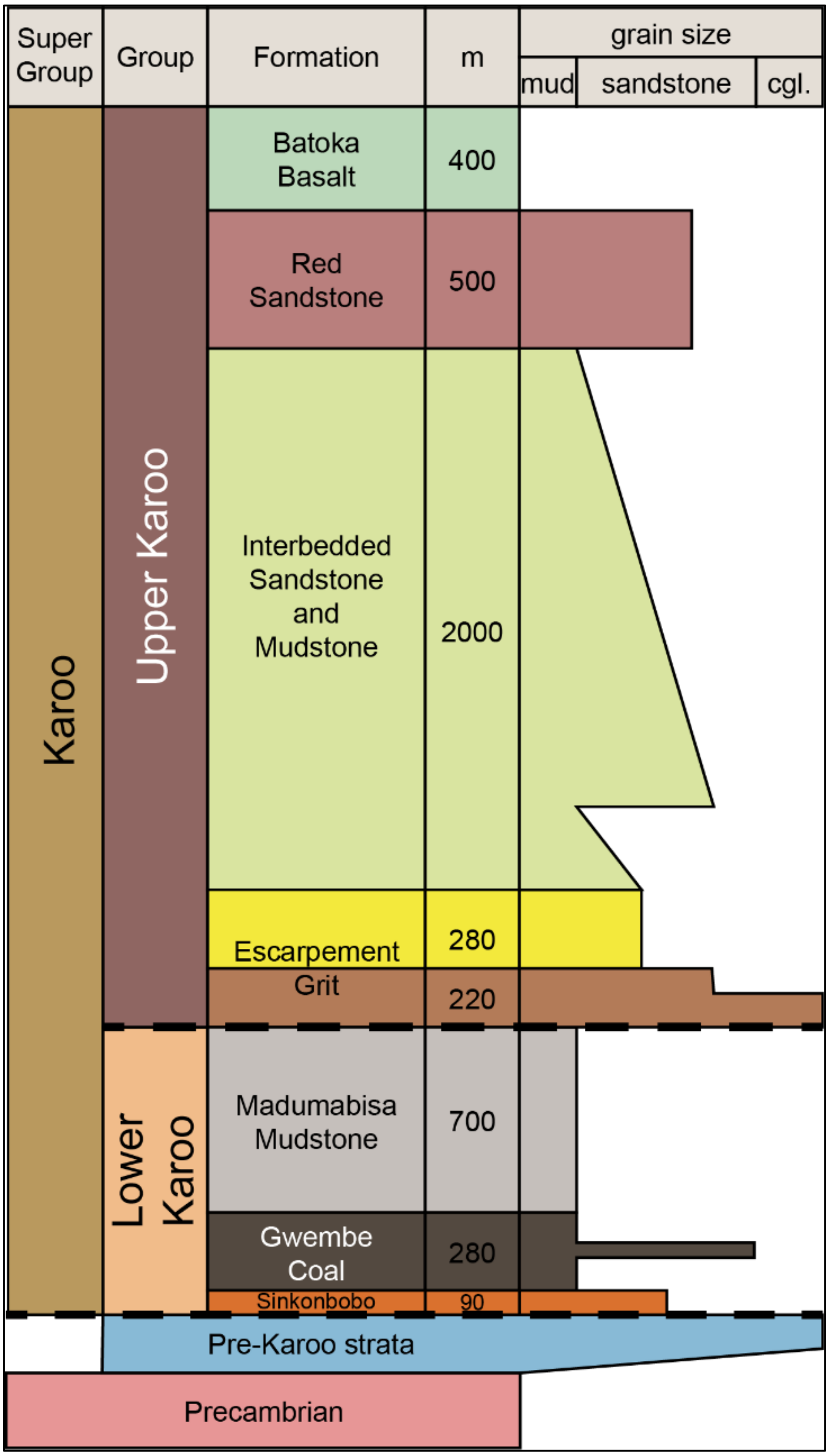

Figure 7-1: Karoo Supergroup stratigraphy in Southern Zambia (Source: (Nyambe and Utting, 1997 within CSA, 2013))

7.1.1. Madumabisa Mudstone

The Madumabisa Mudstone Formation in the mid-Zambezi Valley comprises up to 640 m of non-carbonaceous, alternating massive, poorly stratified, homogeneous mudstone and laminated silty mudstone and siltstone, with minor interbedded calcilutite, sandstone and irregular concretionary calcareous beds (Nyambe and Utting, 1997). The massive mudstone beds have a hackly conchoidal fracture and are predominantly grey to green, silty mudstone with minor, but common, concretionary cacilutite beds up to 1.2 m thick. The laminated mudstone/siltstone units comprise green to grey (greyish white to khaki weathering) parallel laminated to small-scale cross-laminated mudstone and medium bedded siltstone/mudstone with minor calcilutite and sandstone interbeds. Pinkish-grey to dark grey colours are common in the medium-bedded (coarser) and thinly laminated (finer) units. Ellipsoidal concretionary calcilutite beds have variable lateral persistence and contain up to 30 % ostracods, bivalves and fish scales. Thin, dark, bituminous calcilutites and mudstone conglomerate are locally present. Bioturbation is common.

7.1.2. Escarpment Grit Formation

The Escarpment Grit Formation, and its correlatives in the northern Karoo rift basins, lie immediately above the Permian-Triassic boundary and are characterised by extensive braided river deposits. Such deposits are typical of Precambrian fluvial basins, but uncommon in the Phanerozoic (Ward et al., suggesting that these widespread braided river deposits resulted from the die-off of plants during the Permian-Triassic extinction event.

The Escarpment Grit Formation consists of coarse to very coarse-grained sandstones that are locally conglomeratic and fine upwards into more fine-grained sandstones and intercalated mudstones. Silicified wood is abundant locally. AGIP geologists historically distinguished two informal members in the Escarpment Grit suggesting a change in fluvial style. A lower “Braided Facies” member is characterised by relatively poorly sorted sandstones and pebbly sandstones with mudclasts and thin discontinuous mudstones, and an overlying “Meandering Facies” member is characterised by well-sorted upward-fining sandstones (i.e., point bar deposits) with mudclasts and pebble-lag layers, interbedded with laterally extensive mudstones.

In areas of poor exposure, the “Braided Facies” can be distinguished from the “Meandering Facies” by the presence of abundant quartz pebbles at the surface. The thickness of these members is variable, and they appear to thin towards the rift axis. Paleocurrents in the “Braided Facies” are predominantly south-westerly, subparallel to the axis of the mid-Zambezi Rift, whereas paleocurrents in the “Meandering Facies” are highly variable.

A petrographic study of the Escarpment Grit (Prasad and Lehtonen, 1977) in the Bungua Hill area south of Dibbwi reported that the sandstones are texturally immature and range from arkosic to sub-arkosic and sublithic arenites and wackes. Arenites predominate. Feldspar content averages 22 % (4 % to 39 %) and is mainly microcline, with minor oligoclase and albite. Both fresh and kaolinised feldspars may be present in the same sample, suggesting a mixture of fresh and weathered source material rather than diagenetic alteration. Rock fragments average 2.9 % (0 % to 12.2 %), including quartzite, sericitic quartzite, siltstone, chert and jasper ranging up to 12 % of the sandstones.

Muscovite is common and fresh-looking, whereas biotite is less abundant and typically kaolinised and altered to iron oxides. Other accessory minerals comprise less than 0.5 % of the sandstones. They include zircon, tourmaline, epidote, rutile, apatite, sphene, garnet and possibly augite. Matrix (grains less than fine sand size) averages 9.1 % (0 % to 23.4 %) and includes mica, feldspar, quartz and chlorite, recrystallised from clay. Cement include iron oxide, silica and carbonate. The sandstones range from moderately well to poorly sorted with an average porosity of 6.7 %. They are interpreted to be derived from nearby gneisses and granitic rocks of the Katanga Supergroup and Basement Complex.

Stratabound uranium mineralisation in the Escarpment Grit is known in the lower part of the “Meandering Facies” at Njame, and the upper part at Dibbwi. Association with boundaries between sandstone-dominated stratigraphic units suggests that permeability contrast is a factor controlling uranium mineralisation. Widespread soft-sediment folds suggest syn-depositional seismic activity and fault re-activation, with potential seismic pumping of diagenetic fluids contributing to the mineralisation event.

7.1.2.1. Interbedded Sandstone and Mudstone Formation

The Interbedded Sandstone and Mudstone Formation in the mid-Zambezi Valley consists of typically upward-fining very coarse- to very fine-grained sandstone grading into mudstone (Nyambe and Utting, 1997). Mudclasts are a dominant feature in these sandstones. The sandstone to mudrock units are interpreted as mainly channel-fill deposits to overbank fines deposited during floods in braided streams transitional to meandering stream systems. The contact between this formation and the Escarpment Grit Formation is gradational and is placed at the base of a sandstone unit underlying the mudstone interbeds. There is approximately 10 m of greyish-green muddy siltstone and silty mudstone overlain by 10 m of fining upward sandstones. The mudstone/siltstone beds range from 8 cm to 12 cm thick and become thicker towards the top of the sequence. The thin beds are predominantly horizontally laminated with small-scale ripple lamination better developed in the thick beds towards the top of the unit. Kaolinite is abundant, but illite and mixed-layer clays are present in minor amounts. Calcite is present in the lower part of the formation.

(Prasad and Lehtonen, 1977) interpreted the sandstones of the Interbedded Sandstone and Mudstone Formation to be less arkosic than those of the Escarpment Grit, but the average feldspar content of 25.6 % (0.3 % to 37.9 %) reported is higher. Considering the wide range of values, the difference is probably not statistically significant (Yeo, 2011). Rock fragments average 4 % (0 % to 11.1 %), which is also higher than in the Escarpment Grit. The major compositional difference between the sandstones of the Escarpment Grit and overlying Interbedded Sandstone and Mudstone formations appears to be in matrix content, which is twice as high in the latter at 19 % (6.7 % to 38.8 %).

The Interbedded Sandstone and Mudstone Formation, which overlies the Escarpment Grit, contains a Scythian–Anisian age assemblage (Nyambe and Utting, 1997); hence the Escarpment Grit was deposited early in the Scythian epoch (very early Triassic). In the Muntanga area, the contact between the Escarpment Grit and the Madumabisa Mudstone is a paraconformity (Prasad and Lehtonen, 1977). Towards the mid-Zambezi rift margin, the Escarpment Grit oversteps the Lower Karoo to directly overlie basement gneisses, pegmatites and amphibolites. The known uranium mineral deposits in the mid-Zambezi Basin of southern Zambia are all restricted to the Escarpment Grit.

7.1.3. Depositional sequences

The Karoo Supergroup comprises at least six regional depositional sequences (Catuneanu et al., 2005), which reflect broadly synchronous episodes of basin subsidence and climate change but vary considerably in detail from one sub-basin to another. Karoo strata typically overlie Precambrian crystalline basement rocks.

1. **Sequence 1:** Comprises glacial deposits (for example, Dwyka tillite and equivalents) capped by post-glacial lacustrine mudstones laid down in a temperate climate.
2. **Sequence 2:** Comprises coal deposits and associated clastic strata accumulated in a warm humid climate (e.g. Gwembe Coal Formation in Zambia).
3. **Sequence 3:** Comprises fluvial sandstones deposited in semi-humid to arid conditions, overlain by lacustrine or marine mudstones and limestones (e.g. Lower Madumabisa Formation).
4. **Sequence 4:** Comprises lacustrine and fluvial deposits deposited under warm humid to semi-arid conditions (e.g. Upper Madumabisa Formation). A regional unconformity marks the Permian-Triassic extinction event at the boundary between sequences 4 and 5.
5. **Sequence 5:** Comprises fluvial sandstones deposited under warm, hyper-humid conditions capped by lacustrine or more fine-grained fluvial strata deposited under hot, semi-humid conditions (e.g. Escarpment Grit and Interbedded Sandstone and Mudstone formations). The different “Braided Facies” and overlying “Meandering Facies” observed within the Escarpment Grit marks a change in fluvial style from braided streams to meandering rivers where material was deposited at point-bars or flood plains; this likely reflects the re-establishment of riverbank stabilizing vegetation, following the Permian-Triassic extinction event, as suggested by (Ward et al., 2000). The Interbedded Sandstone and Mudstone Formation has also been interpreted as deposition from a meandering river but the thickness and lateral continuity of the mudstone together with a lack of evidence for scouring and an absence of burrows or rootlet traces suggests that the mudstones may be shallow lake or lacustrine pro-delta deposits, rather than flood-plain deposits (Yeo et al., 2010). The sandstones have characteristics of point-bars; hence they may be delta distributary channel deposits.
6. **Sequence 6:** Comprises more fine-grained fluvial sandstones capped by Jurassic basalts (for example, Forest Sandstone and Batoka Basalt). Each sequence is punctuated by an episode of crustal extension and subsidence.

7.2. Regional geological structures

The mineralised zones are offset and impacted by various faults and fractures, but the mineralisation itself does not appear to have any significant structural controls.

Regionally, the Muntanga uranium deposit and other uranium occurrences in southern Zambia, lie near the northwest margin of the Mid-Zambezi Graben. This structure is essentially a half-graben, with its faulted footwall against the Precambrian crystalline rocks on the northwestern Zambian side, and passive onlap on crystalline basement rocks on the southeastern Zimbabwean side. The Mid-Zambezi Graben is subdivided into two major sub-basins by the northeast-trending Kamativi - Chizarira - Matusadona basement block. The north sub-basin is fault-bounded on both its margins and is, hence, a true graben. Cyclic upward fining of Karoo strata (Catuneanu et al., 2005) reflects episodic, fault-controlled subsidence in the graben.

7.2.1. Muntanga, Dibbwi and Dibbwi East

Northeast-trending faults likely controlled deposition of the Escarpment Grit “Braided Facies” and fault-related folds may control blind mineralisation in the Dibbwi and Dibbwi East area ((Yeo, 2011); (Ullmer, 2010); Figure 7-3). The Muntanga area of the Mid-Zambezi Valley is characterised by a series of northeast-trending, fault-bounded cuestas or fault blocks, uplifted to the northwest and dipping to the southeast. Three major northeast-trending anastomosing fault systems can be distinguished in the Muntanga area: the Lusitu, Dibbwi and Bungua Mountain fault zones. There are numerous minor faults of limited extent trending northwest to north.

7.2.2. Lusitu Fault Zone

This fault zone roughly follows the valley along the base of the escarpment, where it is obscured by Quaternary and alluvial deposits of the Lusitu and Lusengesi rivers and their tributaries. Along the northwest side of this fault zone, the down-throw is clearly to the southeast, with Karoo strata at the base of the basement rocks exposed on the escarpment. Madumabisa rocks appear to onlap basements in the Chalala stream area, suggesting that fault offset locally post-date deposition of the Madumabisa (late Karoo or younger).

Along the east side of the Lusengesi – Kayubila segment of the fault zone, downthrown is also interpreted to be to the southeast of the major fault trace. Younger rocks are exposed to the southeast of the older. In the axial part of the Lusengesi – Kayubila segment, the major fault trace is interpreted to be downthrown to the northwest. The relative age of rocks across the fault is uncertain, but moderately to steeply dipping, north- to northwest-trending bedding on the downthrown side is truncated by moderately dipping, northeast-trending. A gentle syncline on the downthrown side is a drag fold.

7.2.3. Dibbwi Fault Zone

The Dibbwi Fault Zone extends through the area of Dibbwi village north. It is a relatively straight, northeast-trending structure, comprising two anastomosing strands along much of its length. Southwest of Dibbwi, both strands are interpreted to be downthrown to the northwest. On the northwest and southeast strands, younger strata are downthrown relative to older ones. A gentle syncline in the hanging wall of the northeast fault strand and parallel to it lie strikes south-southeast sub-parallel to the Lusengesi River. A dome-like feature interpreted to be a diatreme dome lies near Dibbwi village. A prominent linear magnetic high coincides with the westernmost strand of the fault. This may represent a concealed dyke of Batoka basalt intruded along the fault, as interpreted by (Symons and Siegfried (2006)).

A single fault strand to the north of Muntanga splits into two farther to the northeast. Along these, Madumabisa mudstone is uplifted against Escarpment Grit strata. Although northeast-trending fractures parallel to the cliff edge at Muntanga suggest a fault at the base of the cliff, up-dip projection of the Madumabisa – the Muntanga cliffs have likely eroded back from the Dibbwi Fault through undercutting of the mudstone below the sandstone.

North of Muntanga, the southeast fault strand is interpreted to be downthrown to the northwest (e.g. "Meandering Facies" and "Braided Facies" downthrown against Madumabisa mudstone). A gentle anticline lies immediately northwest of this fault strand with its axis parallel to it. A gentle syncline lies parallel and to the northwest of the anticline.

7.2.4. The Bungua Mountain Fault Zone

The Bungua Mountain Fault System comprises two northeast-trending anastomosing fault traces with numerous splays. The two main fault traces pass on either side of Bungua Mountain, and join into a narrow zone northeast of Bungua Mountain, where the Lutele stream crosses the trace and splits again into two traces which extend on either side of another basement ridge north of Mbendele stream.

Southwest of Bungua Mountain, the east fault trace is interpreted to be downthrown to the northwest, consistent with the presence of younger strata to the northwest and older strata to the southeast. Gentle anticlines lie northwest of both the east and west fault traces with their axes sub-parallel to the faults. Along the northwest flank of Bungua Mountain, the west fault trace is interpreted to be downthrown to the northwest, with younger strata to the northwest and older basement rocks to the southeast. A gentle anticline with its axis subparallel to this fault trace lies just west of Bungua Mountain. Along the southeast side of Bungua Mountain, the east fault trace is interpreted to be downthrown to the southeast, with younger strata to the southeast and basement rocks to the northwest. Note that this sense of offset is opposite to the apparent displacement sense on the same fault trace southeast of Bungua Mountain.

Where the fault traces converge in the valley drained by the Lutele stream, downthrow is interpreted to be to the northwest, but exposures are poor and lithologies are indicated to be uncertain. Gentle folds, with axes subparallel to the fault trace, lie northwest of it. The west fault trace which extends along the west side of the basement outlier north of Mbendele stream is downthrown to the northwest.

Prominent linear magnetic highs, comparable to that on the east fault trace of the Dibbwi Fault Zone in the Dibbwi village area, coincide with the main fault trace along the western base of Bungua Mountain and to the southwest, as well as the fault segment about 10 km northeast of Bungua Mountain that extends along the northwestern base of another crystalline basement block. These too, likely represent concealed Batoka basalt dykes intruded along the fault zone.

7.2.5. Minor Faults

North- to northwest-trending faults, with extents of less than four kilometres, crosscut the major fault systems. In contrast with the major faults, they appear to be normal faults. These minor faults likely formed in response to differential uplift on the major faults. One of these extends southerly into the Dibbwi East mineral deposit.

A striking feature of all three fault zones is the development of gentle folds on their hanging-wall side, whose fold axes lie subparallel to the faults. The close spatial association of folds with faults and their orientation indicates that the folding is related to fault movement. Hanging wall folds are commonly associated with normal faults. Depending on the shape of the fault plane, either rollover anticlines or synclinal drag folds (Khalil and McClay, 2002) may be developed. Synclinal drag folds may be formed on the fault-side of rollover anticlines (Yamada and McClay, 2004; Withjack et al., .

As noted above, the extensive linear magnetic highs associated with the Dibbwi and Bungua Mountain fault zones are interpreted to result from Batoka basalt dykes, which are not exposed to the surface. This suggests that these faults were initiated as extensional features following deposition of the Karoo strata, in a final phase of rifting.

Regional seismic studies indicate present-day northwest-southeast crustal extension in the Mid-Zambezi Basin (Dumisani, 2001). Hence, northeast-trending faults are likely to have been reactivated as normal faults. This is consistent with the apparent post-depositional normal offsets of the faults. Although there is no direct evidence for when fault reactivation began or what caused it, it seems likely that it is related to the propagation of a little-studied southwest branch of the East African Rift System along the Karoo-aged Luangwa and mid-Zambezi rifts and further southwest along the Deka fault zone (Chorowicz, 2005; Dumisani, 2001).

7.3. Structural Geology – Dibbwi East (Yeo, 2011)

Historic AGIP geology maps of the Dibbwi East Zone 1 area show it to be cut by a series of east-northeast- to northeast-trending faults 1 km to 6 km long. These faults are subparallel to the major regional fault systems, such as the Dibbwi and Bungua Mountain faults. This contrasts with the minor faults at Muntanga and Dibbwi East, which have predominantly northerly trends.

A series of cross-sections constructed roughly perpendicular to the northeast-trending faults show that most of the minor faults in the Dibbwi East area are normal faults dipping steeply and mainly downthrown to the northwest. The southeastern faults, however, dip and are downthrown to the southeast. Hence the fault block between the northwest- and southeast-dipping faults is a small horst.

All of the faults in the Muntanga deposit region are interpreted to be normal faults (Money and Prasad, 1977; Staley et al., 2009; Titley, 2009; Ullmer, 2009). The continuity of stratigraphic units and offset of stratigraphic boundaries across the faults indicate that most of the observed fault offsets post-date deposition. Thickness changes, occurrence of hanging wall folds and widespread occurrence of soft-sediment deformation features all suggest, however, that some fault displacement was syndepositional. Hence, two distinct structural events have affected the area. Extensional faulting, associated with subsidence of the Mid-Zambezi rift in Upper Karoo time was followed much later by renewed extensional faulting, associated with the southwest branch of the East African Rift System. Most of the mapped faults are related to the later event.

The change in thickness of the Escarpment Grit “Meandering Facies” across the Dibbwi Fault, from about 180 m west of the fault to about 195 m east of it and thinning of the “Meandering Facies” southeast of Dibbwi, to about 70 m at Bungua Hill, suggests syndepositional subsidence, controlled by extensional faults. The faults likely propagated upwards as growth faults, since the two distinctive facies units of the Escarpment Grit are continuous across the faults without major thickness changes, except as noted above. The strong southwesterly orientation of Escarpment Grit “Braided Facies” paleocurrents, suggests deposition in stream systems draining southwest parallel to the axes of one or more half-graben, as noted by Money and Prasad (1977). The presence of numerous circular or elliptical structures, also commonly in the hanging walls of faults and interpreted by Ullmer (2009) as diatremes, and the widespread occurrence of soft-sediment deformation structures in the Escarpment Grit sandstones, are also consistent with syndepositional seismic activity and faulting.

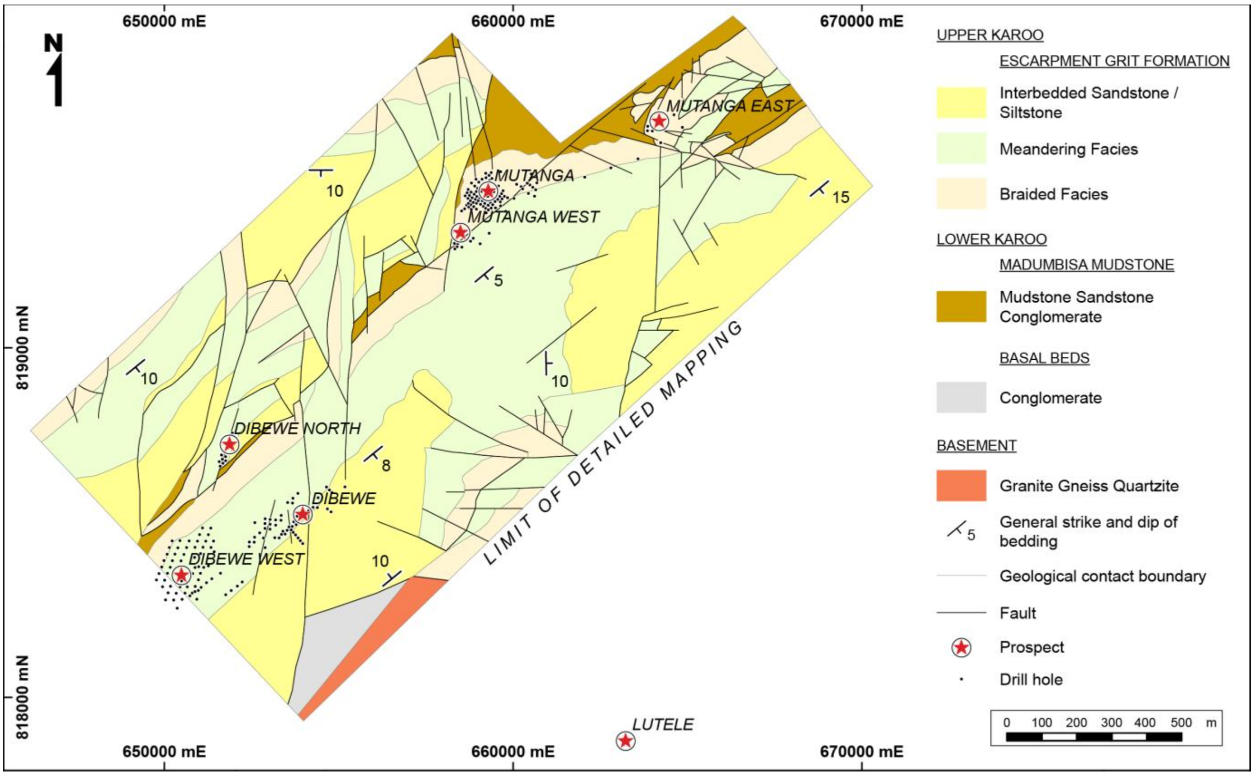

Figure 7-2: Geological map of the Dibbwi-Muntanga area (Source: CSA, 2013)
Note: This map contains historical spelling. “Dibewe” = Dibbwi, “Mutanga” = Muntanga

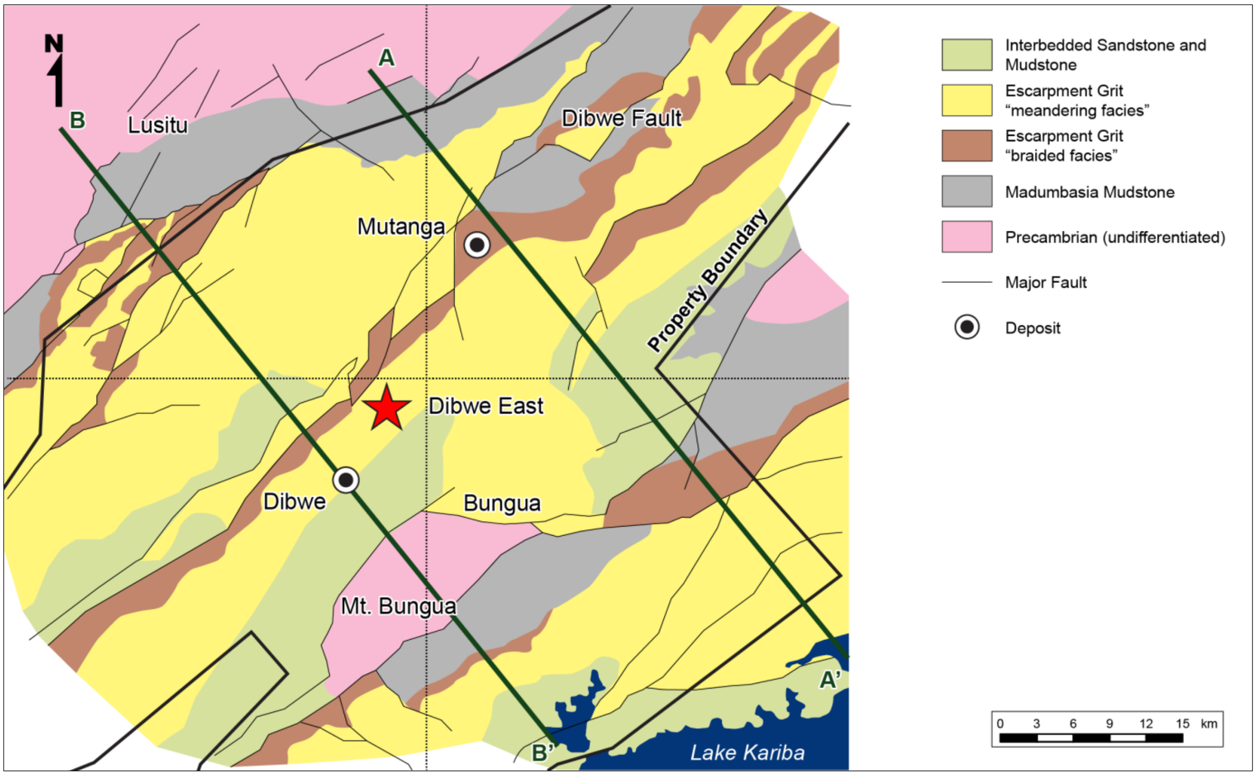

Figure 7-3: Geological map of the Dibbwi-Muntanga area (Source: simplified from Ullmer, 2010 in CSA, 2013)
Note: This map contains historical spelling. “Dibwe” = Dibbwi, “Mutanga” = Muntanga

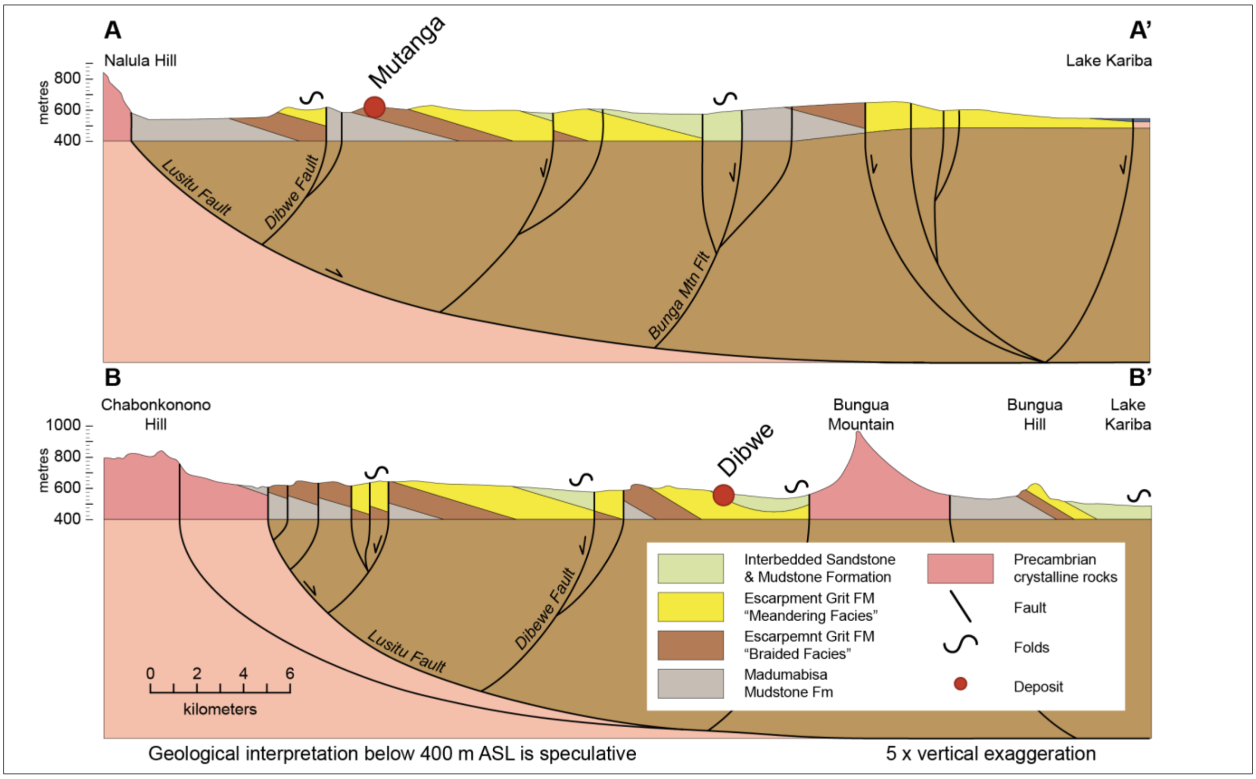

Figure 7-4: Geological cross-section of the Dibbwi-Muntanga area (Area of cross-section shown on Figure 7-3) (Source: Simplified from Ullmer, 2010 in CSA, 2013)
Note: This map contains historical spelling. "Dibewe" = Dibbwi, "Dibwe" = Dibbwi, "Mutanga" = Muntanga

7.3.1. Njame and Gwabi

The Njame uranium deposit consists of Escarpment Grit exposed on a gentle dip slope which faces to the southeast (Figure 7-5). In the northwest, the slope is a much steeper scarp controlled by the position of a northwest dipping normal fault. This fault is downthrown several hundred metres to the northwest, representing one of a number of faults that have caused imbrication in the Kariba Rift. The sequence is also cut by several smaller strike-parallel normal faults, which have caused northwest block-down displacements of up to 25 m. Similarly, the eastern limit of the Njame mineralisation is a major southeast trending wrench fault that truncates the slope and the stratigraphy. The sequence is cut by several smaller strike-parallel normal faults, which have caused down displacements of the northwest block.

Gwabi uranium mineralisation forms a broadly tabular body that dips very gently to the southeast and occurs at very shallow depths of between 3 m and 29 m below surface. In the northwest, the slope is a much steeper scarp controlled by the position of a northwest dipping normal fault. Minor post-mineralisation faulting has locally caused metre-scale offsets to the mineralisation and may have truncated the mineralisation along its southern boundary.

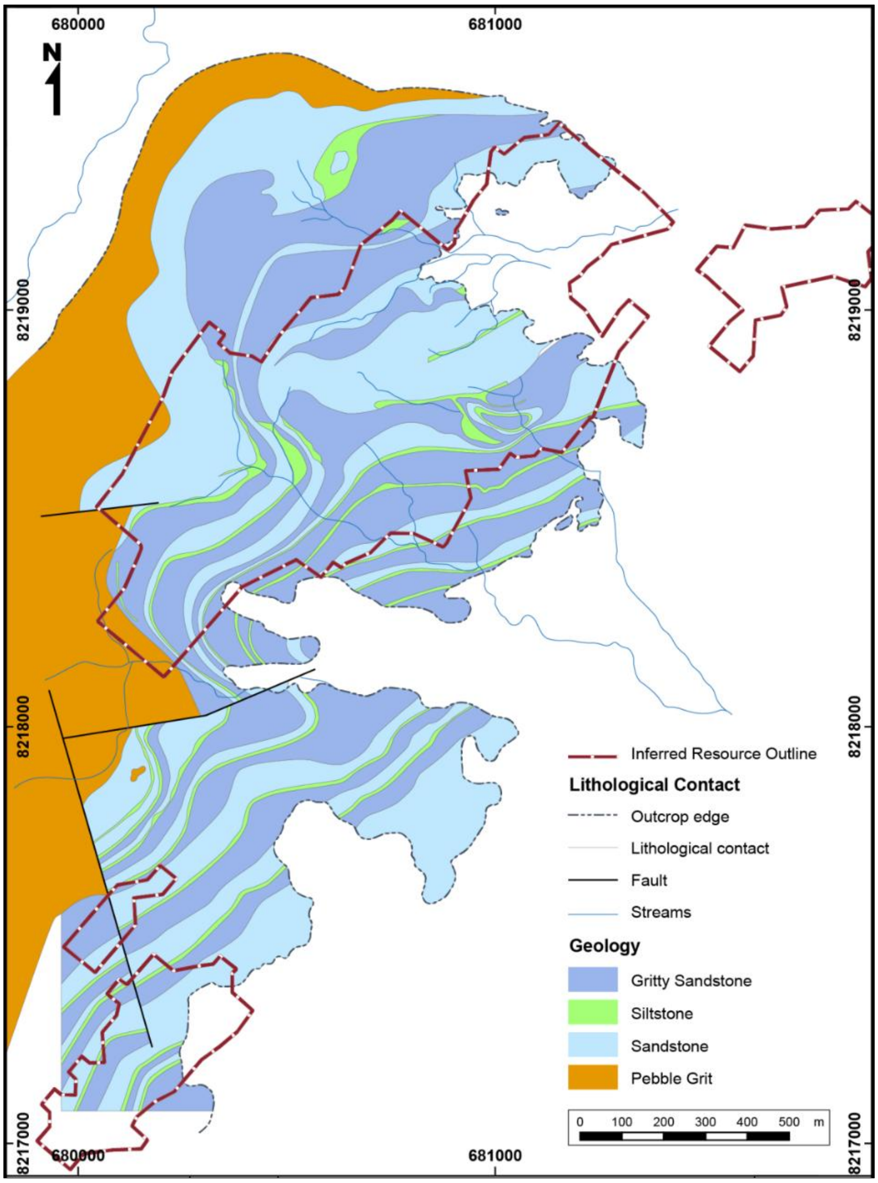

Figure 7-5: Geological map of the Njame deposit (Source: AFR, March 2009 and June 2013)

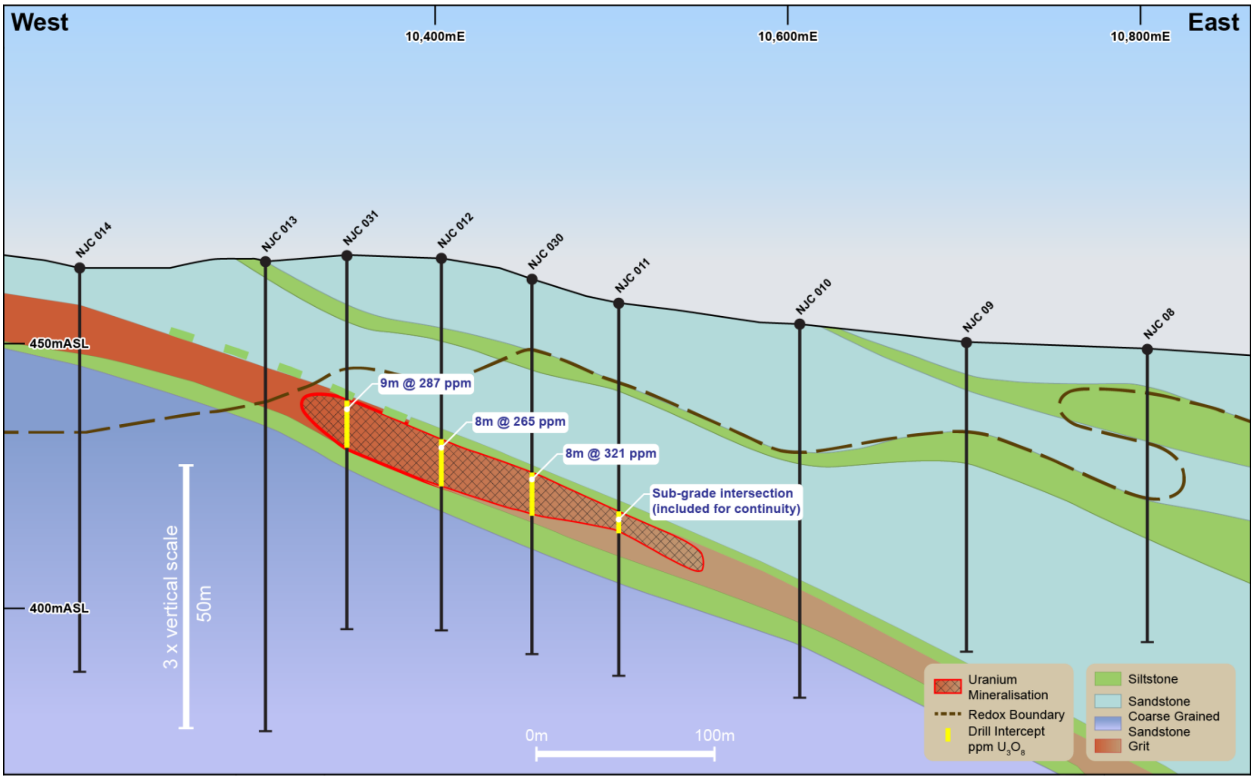

Figure 7-6: Geological cross-section for Njame (Source: AFR, March 2009 and June 2013)

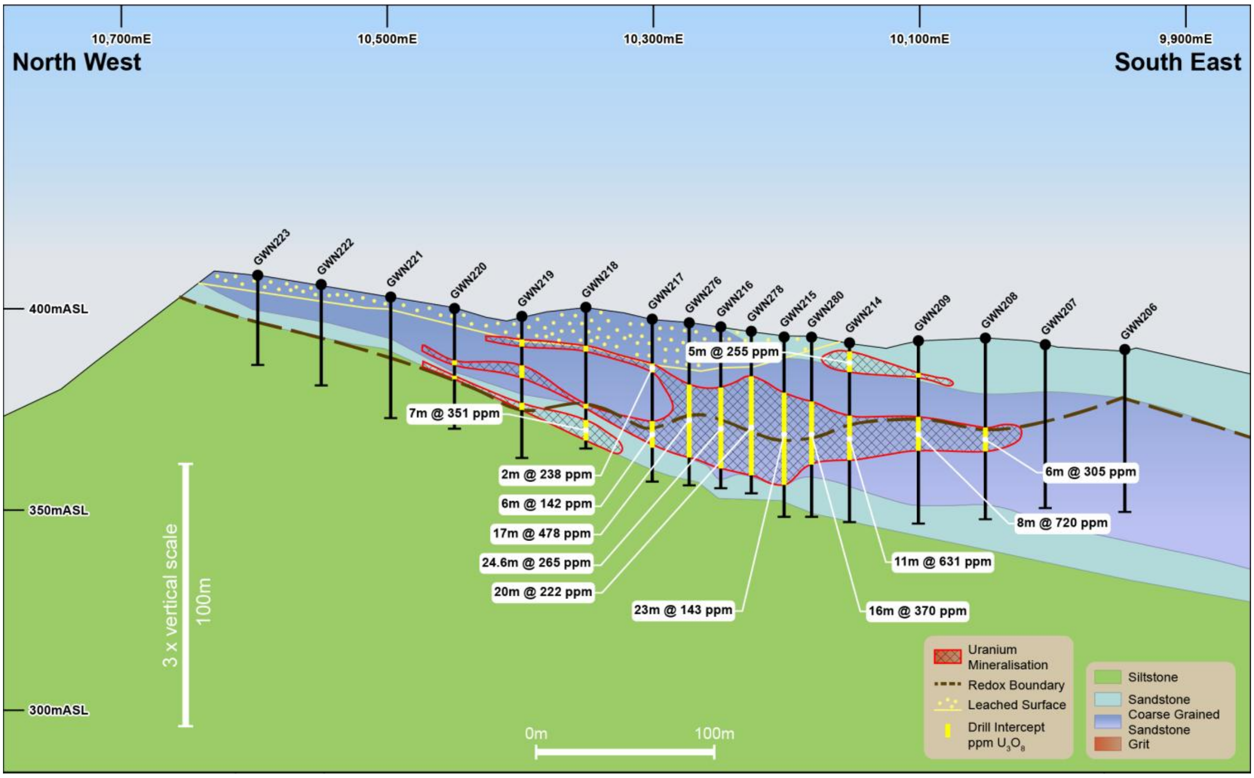

Figure 7-7: Geological cross-section for Gwabi (Source: AFR, March 2009 and June 2013)

7.4. Local geology

7.4.1. Muntanga, Dibbwi and Dibbwi East

The Escarpment Grit Formation sequence at the Muntanga uranium deposit comprises at least 120 m of sandstone and conglomerates with occasional mudstones and silts. The Escarpment Grit Formation overlies the Madumabisa Mudstone Formation which comprises a grey to dark grey silty mudstone, with a dark red hematised layer representing either oxidising groundwater or a sub-aerial surface. The mudstone forms an impermeable unit and is thought to have prevented uranium mineralisation from moving further down through the stratigraphy. The contact between the Madumabisa Mudstone Formation and the overlying Escarpment Grit Formation is between two and three metres above the dark red hematised layer.

7.4.2. Muntanga geology

The Muntanga uranium deposit is located 31 km northwest of Siavonga. Three stratigraphic zones (“Packages”) were historically identified from core logging and utilised as geological boundaries during the resource evaluation phase at Muntanga. The stratigraphic sequence for these packages commences with Package A as the Basal Zone, overlain by Package B, and Package C at the top. The three packages are detailed as follows:

7.4.2.1. Package A

‘Package A’ is approximately 24 m thick. Overlying the Madumabisa Mudstone Formation, it is a thick, dark grey mudstone coarsening upwards into pyritic, coarse-grained sandstones. Small-scale slump structures and occasional possible dewatering features are observed. Occasional iron oxides are noted. ‘Package A’ is capped by an approximately 5 m thick, coarse matrix- supported conglomerate. This conglomerate marks a sudden, high-energy event, possibly a channel. The sequence is thought to be representative of a prograding, possibly deltaic system.

7.4.2.2. Package B

‘Package B’ is approximately 70 m thick. Overlying ‘Package A’ is a sequence of repeated fining-up cycles that coarsen upwards. Each fining-up unit starts with a very coarse-grained sandstone or conglomerate and fines up to a mudstone or siltstone. The units contain a variety of sedimentary structures including trough and tabular crossbedding and laminations.

The fining-up cycles are thought to be representative of a fluvial, possibly meandering system, in which mudstones were laid down in calm lacustrine, bow lake or over bank deposits. The deposits laid down in such hiatal periods could give a series of laterally continuous deposits that could be used as marker bands. Their role in mineralisation is discussed below.

Sulfides are observed to be within an approximate depth of 50 m from the surface. Above this depth, oxidisation and weathering are evidenced by reddish brown and orange iron oxides and the breakdown of micaceous and feldspathic minerals. For drill hole logging purposes, the top of the Escarpment Grit Formation ‘Package B’ is taken as being the first down hole presence of mudstone.

7.4.2.3. Package C

‘Package C’ is approximately 25 m thick. Overlying ‘Package B’ is interpreted from drilling as the uppermost unit within the Escarpment Grit Formation in the area. ‘Package C’, although possibly related to ‘Package B’, is distinguished by grain size and structural variances. ‘Package C’ comprises bedded, generally very coarse-grained sandstones with occasional conglomerates. Both sandstones and conglomerates contain less sedimentary structures than ‘Package B’ and display smaller variations in grain size with little or no cyclic variation (although individual beds can display sedimentary structures). Mudstones are generally absent, although conglomerates often contain mud balls. ‘Package C’ may represent a less ordered environment than Package ‘B’, possibly a braided channel system.

7.4.3. Dibbwi and Dibbwi East geology

The Dibbwi uranium deposit is located approximately 10 km to 15 km west of the Muntanga area. Mineralisation in the Dibbwi area appears to be hosted by relatively un-faulted "Meandering Facies” units of the Escarpment Grit Formation.

The Dibbwi East mineral deposit is predominantly composed of Escarpment Grit Formation. The surface geology is characterised by a few scattered sandstone outcrops. Two major units can be distinguished in core, the “Braided Facies” member of the lower Escarpment Grit Formation and the “Meandering Facies” member of the upper Escarpment Grit Formation which appear to be transitional from one another. Most of the Dibbwi East mineralisation occurs in the “Meandering Facies”. At Dibbwi East a clear interface can be observed between surface oxidation to a depth of approximately 40 m, where the sedimentary sequence is bleached with red iron oxide horizons, usually at the interface between mudstones and sandstones. Underlying this oxidised sequence, the sedimentary pile could be considered fresh, where the colour of the sandstone, mudstone and siltstones is dominantly grey to dark grey to green, with sulfides present in areas.

Strata dip at about 8° to 15° in the south-easterly direction and strike in the northeast-south-westerly direction. The sandstones are predominantly massive looking with cross beddings indicating that they are channel deposits. Cross-bed foreset orientations are variable suggesting high sinuosity (meandering) river deposition. Sandstone layers 10 m to 50 m thick tend to alternate with 2 m to 5 m thick mudstones and siltstones. Mudstones can be laterally continuous for hundreds of metres.

Manganese nodules are common at the surface. These manganese nodules are composed of pyrolusite and hollandite and usually contain uranium mineralisation. The uranium is homogeneously distributed within the host manganese and phosphatic minerals. The manganese nodules are believed to have formed by compaction of wet sediments which led to the remobilisation and formation of manganese nodules at the aerated sediment-water interface, and uranium-enriched phosphorite lenses below the interface in reducing conditions. Epigenesis occurred through the passage of solution fronts which recrystallised the manganese and phosphatic minerals and remobilised the uranium which was leached away. The mechanism of uranium uptake in manganese phases most probably involves the adsorption of $((UO_2)_3.(OH)_5)^+$ complexes on precipitating minerals.

Mudballs are present in the drill core. These are rounded clasts of clay which bind sediments and minerals to their surfaces. Most are pyritic and sticky, which presumably facilitated preservation during transport over hundreds of metres in a river, with eventual disintegration.

7.4.4. Njame and Gwabi

7.4.4.1. Njame geology

The geology of the Njame uranium deposit is relatively simple, consisting entirely of Escarpment Grit exposed on a gentle dip slope which faces to the southeast. In the northwest, the slope is a much steeper scarp controlled by the position of a northwest dipping normal fault. This fault is downthrown several hundred metres to the northwest, representing one of a number of faults which has caused imbrication in the Kariba Rift.

The sequence is also cut by a number of smaller strike-parallel normal faults which have caused northwest block down displacements of up to 25 m. A south-east trending fault appears to have caused a rotational offset between the northern and eastern parts of the Njame deposit. Furthermore, a second series of faults with displacements less than the strike-parallel faults have offset the stratigraphy and the mineralised horizons.

A variety of clastic sediments are developed at Njame, ranging from coarse conglomerate beds several tens of metres thick to thinly bedded or cross-bedded fine to medium-grained sandstones. Thin bands of shale and mudstone are intercalated in the sequence. AFR historically identified five facies packages (AFR, March 2008), numbered F1 to F5 from base to top, which showed a general fining upwards trend, often with a thin mudstone or shale horizon defining the top of the sequence and marking the base of the next cycle. Individual sequences also trend towards finer sediments down-dip, reflecting changes from proximal to distal environments. This interpretation is consistent with paleo-current indicators suggesting transport from between the northwest and northeast. The mudstone horizons, which represent quiescent phases in the sedimentation, comprise the most laterally continuous lithologies and are thus useful marker horizons.

7.4.4.2. Gwabi geology

Similarly to Njame, the geology of the Gwabi uranium deposit also consists entirely of Upper Karoo Escarpment Grits exposed on a gentle dipping southeast-facing slope. A variety of clastic sediments are developed at Gwabi, ranging from coarse conglomerate beds several tens of metres thick to thinly bedded or cross-bedded fine to medium-grained sandstones. Thin bands of shale and siltstone are intercalated in the sequence. Below the grits are well-developed calcareous shale and siltstone layers, possibly representing the upper part of the underlying Madumabisa Mudstone.

7.5. Mineralisation

7.5.1. Muntanga, Dibbwi and Dibbwi East

Uranium mineralisation appears to be later than at least some of the normal faults which cut the Escarpment Grit Formation. This is evident from the good correlation of the radiometric logging data between adjacent holes within the Muntanga deposit separated by interpreted faulting (Lusambo, 2011).

The source of the uranium is believed to be the surrounding proterozoic gneisses and plutonic basement rocks. Having been weathered from these rocks, the uranium was dissolved, transported in solution and precipitated under reducing conditions in siltstones and sandstones. Post-lithification fluctuations in the groundwater table caused dissolution, mobilisation and redeposition of uranium in reducing, often clay-rich zones and along fractures.

Mineralisation is not strictly associated with a particular unit in the stratigraphic section. It is observed to occur in both the fine-grained and coarser material and in mudstones, especially where fractures and mud balls occur. Some mineralisation occurs in association with manganese oxide or disseminated with pyrite. Mineralisation in some bore holes is seen to occur where there was a grey alteration, limonite and feldspar alteration and in dark grey mudstones (Sakuwaha, 2011). The strata dip in the south-easterly direction and mineralisation seems to occur along dip.

Uranium mineralisation occurs in a number of different associations:

- Disseminated uranium mineralisation
  - Occurs in sandstones, conglomerates, and within mud layers, mud balls and mud flakes. Uranium is present as interstitial fine-grained crystals or small amorphous masses constituting less than 1 % by volume. Grades vary considerably between zones of dissemination, from approximately 20 ppm U_3O_8 to 2 000 ppm U_3O_8 in mineralisation thought to be solely of a disseminated nature. The presence of sulfides alongside uranium oxides may indicate a transitional zone and/or preferential replacement/ reduction of uranium compounds by one chemical route over another (such as decaying organic matter over oxidation of sulfides) as uraniferous groundwaters moved through the lithologies.
- Uranium mineralisation associated with mudstones and siltstones
  - Muddy lithologies include mud balls (within sandstones), flakes and interbeds. In some cases, mud balls may be completely replaced by uranium mineralisation. The degree of replacement varies from fully replaced mud balls to those with a thin selvage of mineralisation, whilst others are unmineralised. This is attributed to different ground water chemistry, varying volumes of reducing matter within the mud (fully replaced material may have been a peat-like material), and porosity of the muddy lithology during the influx of uraniferous ground water.
- Fracture-hosted uranium mineralisation
  - Uranium mineralisation is seen as crystal coatings on surfaces and as concentrations close to surfaces. Most notably at the Dibbwi-Muntanga-Dibbwi corridor, these fractures are coated with black iron (“Fe”)/ manganese (“Mn”) oxides which in turn may be coated with secondary uranium phosphate mineralisation (autunite, meta-autunite and selenite).
- Primary uranium mineralisation
  - Outside of the overlying oxidised zone, the mineralisation is associated with redox fronts within sandstone layers, where the interface can be seen by a change in colour from pale grey white to darker grey and the presence of pyrite. It is interpreted that mineralised fluids move along the layers as opposed to the Oxidised zone, where fluid movement is vertical. Other controls on mineralisation appear to be the permeability differences where finer-grained sediments and “dirty” sandstone are better hosts to uranium due to the presence of reductants such as organic matter or sulfides but also reduce the flow rate of groundwater such that reduction reaction can happen. The mineralisation is considered primary and consists mostly of pitchblende, uraninite or coffinite.

7.5.2. Njame and Gwabi

At Njame, the uranium mineralisation occurs at the interface between siltstones and sandstones at redox boundaries. Approximately 25 % of the Njame mineralisation is siltstone hosted, with the balance in coarser-grained sandstones and grits.

Drilling conducted by AFR (AFR, March 2008; April 2012) identified two main mineralised horizons; the thickest, most consistent and highest grade is the lower horizon within the second sequence from the base. Drilling was carried out along the entire length of the 5 km long system, with uranium mineralisation encountered along the entire length. Unlike the high-energy sandstone and grit horizons, which show very rapid changes over several tens of metres, the siltstone horizons are generally laterally continuous for hundreds of metres, except where younger grit/sandstone channels have cut through them. There is a clear stratigraphic control on mineralisation at the deposit scale, although structural control may be present on a larger scale.

Similarly to Njame, the uranium mineralisation at Gwabi is related to the redox front; there is one main mineralised horizon which appears to be controlled by both lithology and the redox boundary. It is hosted by the coarse-grained sediments that are interpreted to be the along-strike continuation of the Escarpment Grits which host the Njame uranium mineralisation. Uranium mineralisation at the Gwabi deposit occurs in red, oxidised, coarse-grained sandstones, grits and pebble conglomerates which overlie a green, non-mineralised, reduced silty-shale horizon. This is interpreted to represent a major redox boundary and maybe the regional unconformity between the upper and lower Karoo.

8. Deposit types

8.1. Summary of sandstone uranium deposits

The primary uranium mineralisation in the Karoo rocks of the Project conforms to a sandstone-hosted fluvial channel-type deposit ((Nash et al., 1981); (Turner, 1988)). Sandstone uranium deposits are generally of three types:

- Roll-front type uranium deposits – arcuate bodies of mineralisation that crosscut sandstone bedding, such as those that occur at the boundary between the up-dip and oxidised part of a sandstone body and the deeper down-dip reduced part of a sandstone body
- Peneconcordant or tabular sandstone uranium deposits – irregular, elongate lenticular bodies parallel to the depositional trend, also called Colorado Plateau-type deposits, most often occur within generally oxidised sandstone bodies, often in localised reduced zones, such as in association with carbonised wood in sandstone paleochannels incised into underlying basement rocks
- Tectonic/ Lithologic uranium deposits – occur in sandstones adjacent to a permeable fault zone; mineralisation forms tongue-shaped mineralised zones along the permeable sandstone layers adjacent to the fault. Often there are several mineralised zones 'stacked' vertically on top of each other within sandstone units adjacent to the fault zone (McKay and Miezitis, 2001).

Sandstone uranium deposits are contained within medium to coarse-grained sandstones deposited in a continental fluvial or marginal marine sedimentary environment. Impermeable shale or mudstone units are interbedded in the sedimentary sequence and often occur immediately above and below the mineralised horizon (Dalhkamp, 1993). Uranium is mobile under oxidizing conditions and precipitates under reducing conditions, and thus the presence of a reducing environment is essential for the formation of uranium deposits in sandstones (Nash et al., 1981).

The Karoo basins of sub-Sahara Africa comprise what may be the world's largest sandstone-hosted uranium province (Figure 8-1). Compared to the well-known uranium-bearing sandstone basins of the western US, the area of the Karoo basins is about 30 % greater, but their known uranium content as of 2003 was only about 7 % of that in the US basins. Whereas both areas contain broadly similar, little deformed, predominantly non-marine strata, mainly of Mesozoic age, the order of magnitude lower apparent uranium content of the Karoo basins indicates that they are relatively underexplored (Roux, 1998; Bowell et al., 2009).

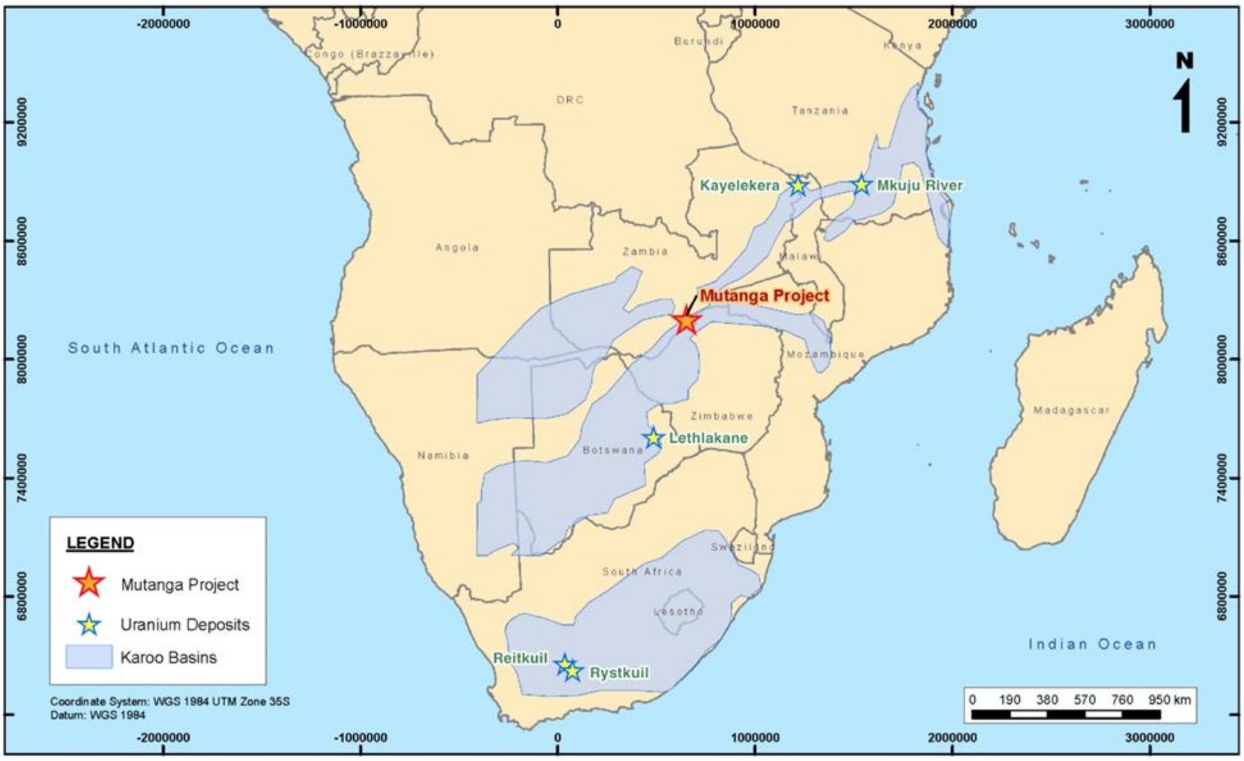

Figure 8-1: Surface extent of Karoo Basins in Sub-Sahara Africa and proximity of known uranium deposits

Only one Karoo uranium deposit, Lotus Energy's Kayelekera deposit in Malawi, has been developed. Kayelekera is on care and maintenance, but in October 2024 Lotus Energy released an accelerated restart plan with an eight-to-ten month timeline to first uranium production. Other deposits have economic potential (Yeo, 2010).

These deposits have some key features in common:

- All are hosted in fluvial arkosic sandstones that have undergone post-depositional faulting and uplift (tectonic inversion)
- All lie at or near the surface and hence, typically have strong surface radiometric expression
- All appear to have tabular geometry; no classic roll-front deposits have been convincingly demonstrated
- Most feature a range of mineralisation styles, including primary uranium oxides and silicates in relatively reduced sandstones, secondary uranyl phosphates or vanadates in more strongly iseoxidised sandstones and secondary mineralisation remobilised into surficial calcretes
- Mineralisation is commonly associated with stratigraphic contacts indicative of a marked drop in stream energy.

9. Exploration

9.1. Introduction

In addition to the drilling described in Section 10, extensive exploration work has been conducted on all of the deposits of the Project by the former owners of the Project. More recent exploration activity has been carried out by GoviEx, focusing specifically on the Nabbanda exploration license that is located between the Muntanga and Chirundu Mining licenses. In 2024, GoviEx drilled five holes on the Nabbanda exploration license and three of these drillholes indicated the potential presence of uranium, hosted within the escarpment grit formation. This recent data highlights the potential of Nabbanda as a promising prospect for further uranium exploration within the overall Project.

9.2. Muntanga, Dibbwi, and Dibbwi East

The earliest phase of exploration for uranium in the area covering the Muntanga and Dibbwi deposit areas was conducted by AGIP in the late 1970s to the mid-1980s.

AGIP carried out systematic exploration, comprising outcrop mapping, ground radiometric surveys, air-borne photographic and geophysical surveys, trenching and pitting. Regional exploration drilling was carried out in the broad Muntanga-Dibbwi area. A summary of the regional mapping completed is shown in Figure 9-1.

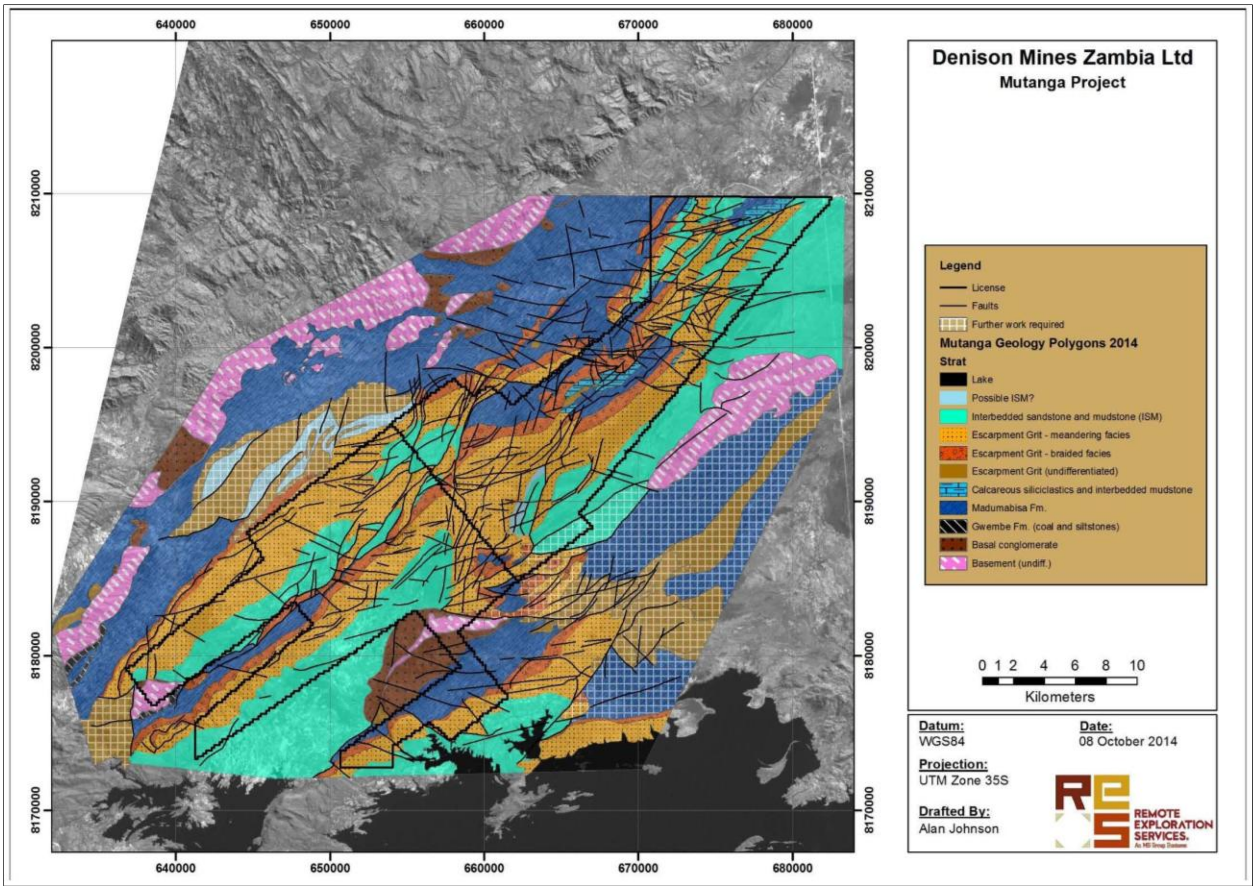

Figure 9-1: Dibbwi – Muntanga geological map (Source: RES, 2013)

During 2006, a detailed aeromagnetic and radiometric survey was carried out by OmegaCorp which confirmed the position and tenor of the existing uranium prospects and identified additional targets, based on interpreted radiometric signatures. Conclusions of the 2006 airborne survey noted the following:

- The Escarpment Grit Formation appears to have two clear radiometric signatures as shown in Figure 9-2
  - A reddish-brown ternary radiometric signature indicates the presence of potassium ("K") in the Formation, consistent with the description of the Escarpment Grit Formation as feldspathic sandstone. This part of the Escarpment Grit Formation was mapped and designated as D1 (Figure 9-3)

  - The areas marked as D2 appear to have a similar K response but with additional uranium producing a white ternary radiometric signature
- The structures identified indicate an extensional half-graben regime with normal faults trending in a generally northeast direction. The movement on these faults appears to down-throw blocks to the northwest. Later faulting in a northwest, west-northwest and north-northeast direction crosscutting the Karoo stratigraphy is also noted.

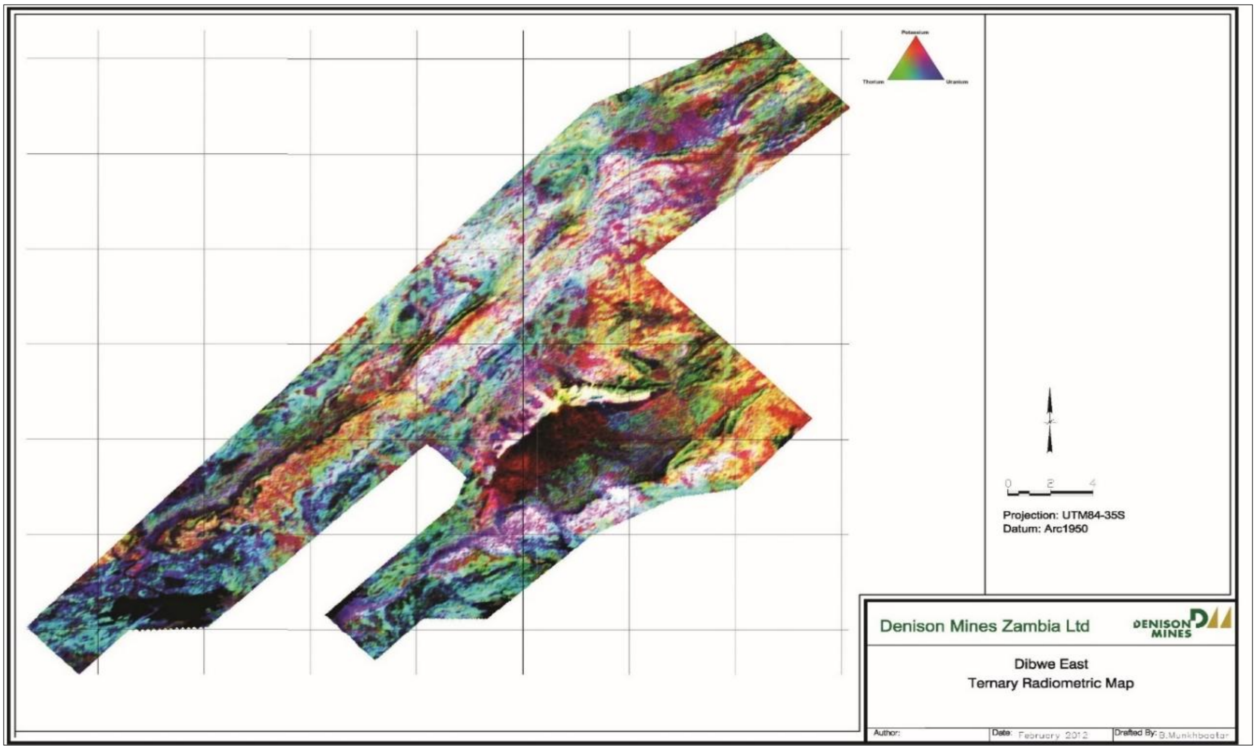

Figure 9-2: Ternary radiometric plot (Source: Denison-RPA, 2012)

In 2011, a Denison geophysicist noted some obvious errors in the magnetic data quality and derived products and subsequently had an external processor look at the 2006 data, who confirmed that the gridded data within this region was representative of their processing sequences. Assumptions were made that since the radiometric signal from the equivalent potassium was mapping the near-surface expression of the Escarpment Grit Formation; this implied that the high-frequency content from the magnetic signature (2nd vertical derivative grid) was also representative of geological variations within the Escarpment Grit Formation.

Furthermore, by closely examining the potassium/magnetic datasets on larger formational trends an inverse relationship occurs between mudstones and sandstones. The units are distinguishable with mudstones having a high mag/low potassium signature and sandstones as a low mag/high potassium signature (Denison-RPA, 2012). Resolution of the magnetic dataset is much better at defining faulting, lineaments and/or edges of magnetic domains as evidence in a provisional interpretation of lineaments and offsets in the area (Figure 9-4).

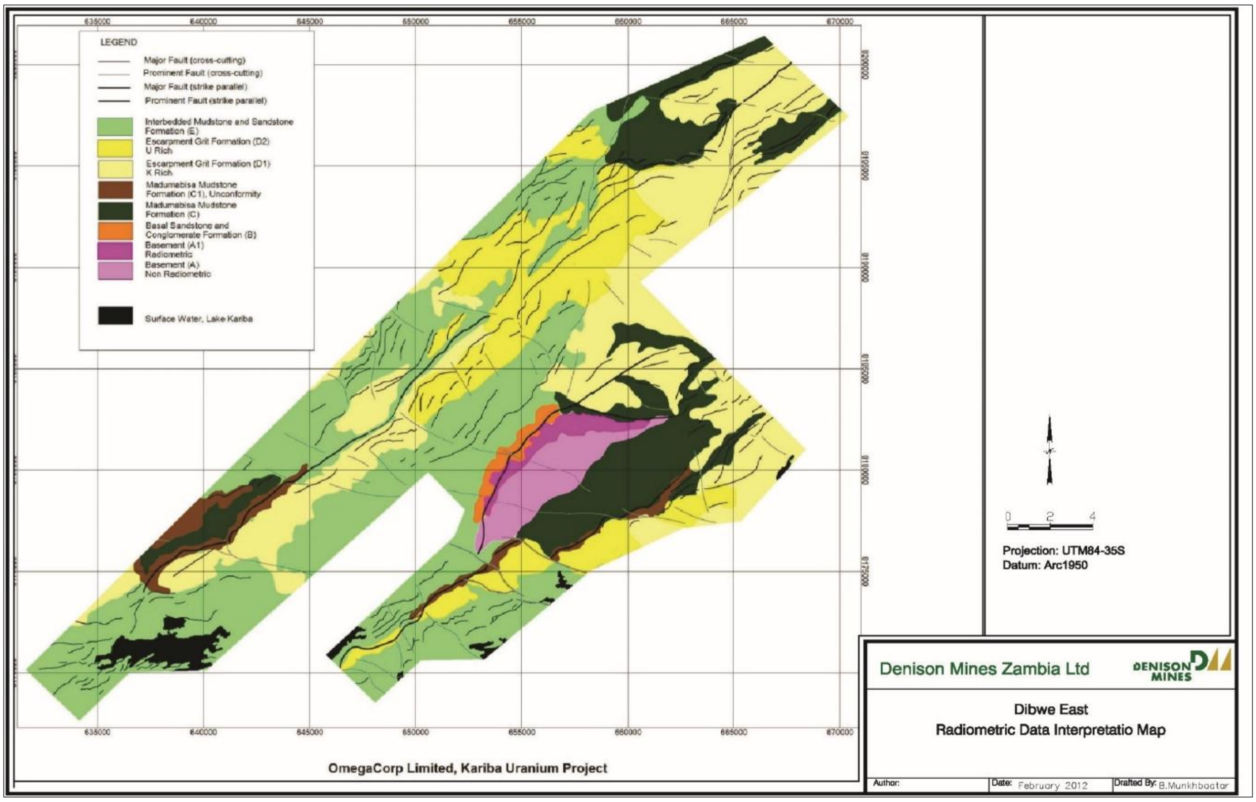

Figure 9-3: Interpretative map, based on radiometric data shown in Figure 9-2

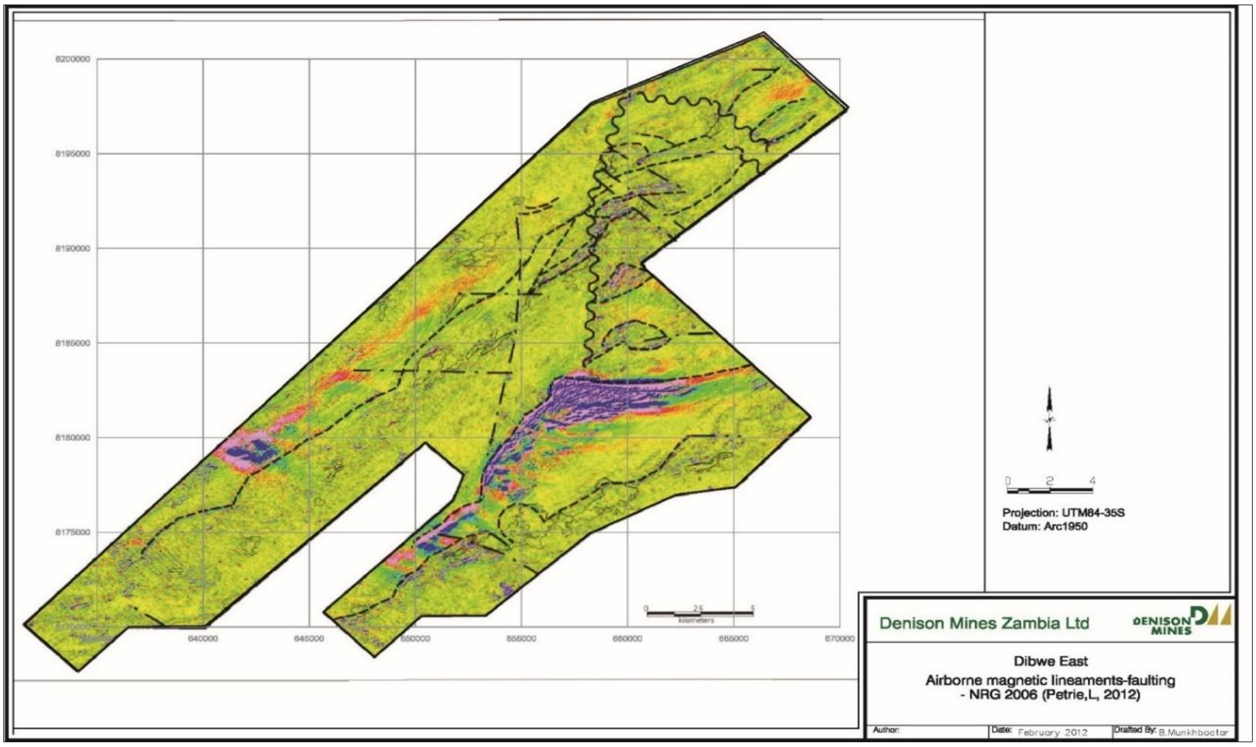

Figure 9-4: Airborne magnetic lineaments-faulting (Denison-RPA, 2012)

During August and September 2013, Geotech Ltd. carried out a helicopter-borne geophysical survey of the Project. Principal geophysical sensors included a versatile time domain electromagnetic ("VTEMplus") system, and horizontal magnetic gradiometer. Ancillary equipment included a GPS navigation system and a radar altimeter. A total of 1 903-line kilometres of geophysical data were acquired during the survey. In-field data quality assurance and preliminary processing were carried out daily during the acquisition phase. Preliminary and final data processing, including the generation of final digital data and map products, was undertaken from the office of Geotech Ltd. in Aurora, Ontario. The processed survey results are available as the following maps:

- Electromagnetic stacked profiles of the B-field Z Component
- Electromagnetic stacked profiles of dB/dt Z Components
- B-Field Z Component Channel grid
- Total Magnetic Intensity (“TMI”)
- Fraser Filtered dB/dt X Component Channel grid
- Magnetic Total Horizontal Gradient
- Magnetic Tilt-Angle Derivative
- Calculated Time Constant (Tau) with contours of anomaly areas of the Calculated
- Vertical Derivative of TMI and
- RDI sections are presented.

Digital data includes all electromagnetic and magnetic products, plus ancillary data including the waveform. The survey report describes the procedures for data acquisition, processing, final image presentation and the specifications for the digital data set.

Geological mapping of the Muntanga property was undertaken during August and September 2014 by Remote Exploration Services (“RES”) of Cape Town, South Africa. A total of 324-line kilometres of mapping traverses were completed including 1 815 mapping stations. Field mapping data were integrated with airborne geophysical data, satellite imagery and previous geological maps and interpretations to produce a revised geological map for the Muntanga property (Figure 9-1).

The Project area was covered with soil geochemical and radon surveys from 2013 to 2015. The objective of the surveys was to delineate any significant exploration targets outside of the drill-defined uranium deposits. Previous drilling had largely focused on testing airborne radiometric anomalies and the soil geochemical and radon approach allowed for possible detection of blind or buried mineralisation, particularly in areas of thick or transported regolith. Surveys were carried out in the dry months between May and November. Coincident soil and radon stations were 100 m apart on 800 m spaced northwest-southeast survey lines. Survey data and results have been stored in an Access database. A summary of the soil and radon samples collected from 2013 to 2015 is provided in Table 9-1 and shown in Figure 9-5. Prior to the implementation of the surveys, calibration exercises were conducted over known mineralisation to establish optimal methodologies.

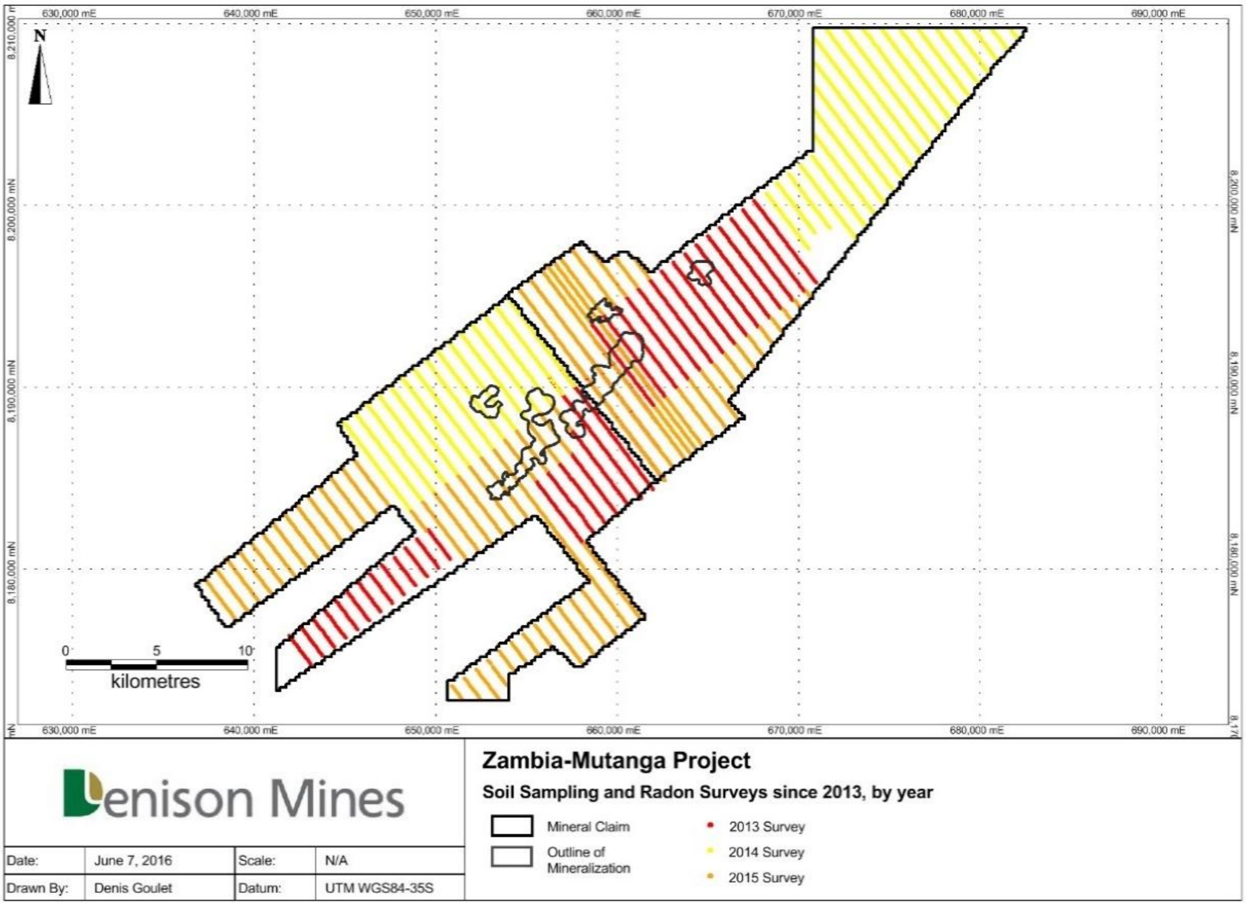

Figure 9-5: Soil geochemical and radon maps, 2013 to 2015 (Denison-RPA, 2012)

Table 9-1: Summary of the soil and radon samples collected from 2013 to 2015

Year Date	Soil samples	Soil field duplicates	AlphaTrack	RadonX
2013	1 780	93	1 680	0
2014	2 029	105	0	2 028
2015	2 248	93	0	2 247
Total	**6 057**	**291**	**1 680**	**4 275**

At each sample site, a 300-gram unscreened sample was collected from the A-horizon. Sample site information and coordinates were recorded in field notebooks. Samples were sent to ACME Laboratories in Vancouver, Canada for analysis using Group 1F, aqua regia digestion ultra trace ICP-MS method. Quality control was monitored with field duplicate samples that were collected at a frequency of one duplicate in every 20 samples.

In 2013 the AlphaTrack method was used, following successful orientation work conducted in 2011. AlphaTrack cups are 1-litre plastic cups with a small piece of special plastic film taped to the inside. The cups are buried in an inverted position so that any radon gas percolating upward will be trapped in the cup. The cups are typically left in place for about 4 weeks. Radon gives off alpha particles which leave microscopic trackways on the film. The trackways can be counted in the lab to give a quantitative measurement of the amount of radon trapped in the cup. This, in turn, indicates the location and grade of subsurface uranium mineralisation.

In 2014 and 2015 the RadonX^{TM} method was utilised, following successful orientation work in 2012. RadonX is provided by RES of Cape Town, South Africa. RadonX is based on the radon-on-activated-charcoal technique initially developed by the SA Atomic Energy Board but refined and enhanced by RES. Unlike other radon emanometry methods that rely on alpha-particle detection, RadonX measures the gamma emission from radon's daughter products, bismuth (214Bi) and lead (214Pb), following adsorption of the radon onto activated charcoal. This method of detection excludes the detection of thoron (220Rn) arising from thorium that may be contained in the bedrock, representing a significant advantage of the RadonX method. Radon gas is adsorbed onto activated charcoal contained within a cartridge fitted into the base of an inverted cup that is buried in the ground. Gamma radiation from the daughter products of the adsorbed radon is then measured using a field scintillometer. Background effects are reduced and corrected by using a lead castle. During the ten-day cup burial period, weather is to be monitored. Rainfall and temperature are known to affect the ability of charcoal to adsorb radon. RadonX cartridges are subjected to stringent quality control measures from the time of initial loading of activated carbon through field deployment up to the time of taking scintillometer readings.

The soil geochemical and radon surveys produced numerous anomalies across the Project area and new exploration targets were defined for follow-up. The soil geochemical and radon methods utilised adequately detected the drill-defined mineralisation and showed a reasonable correlation with radiometric anomalies, thereby confirming this exploration approach. The new exploration targets were defined based on combinations of anomalous soil uranium, soil uranium pathfinders, radon and soil radioactivity. In some cases, the targets corresponded with surficial cover (thicker soils) alluding to a buried source. Targets located over prospective geology and structure were prioritised for follow-up. Figure 9-6 and Figure 9-7 shows the gridded soil uranium and gridded radon results respectively.

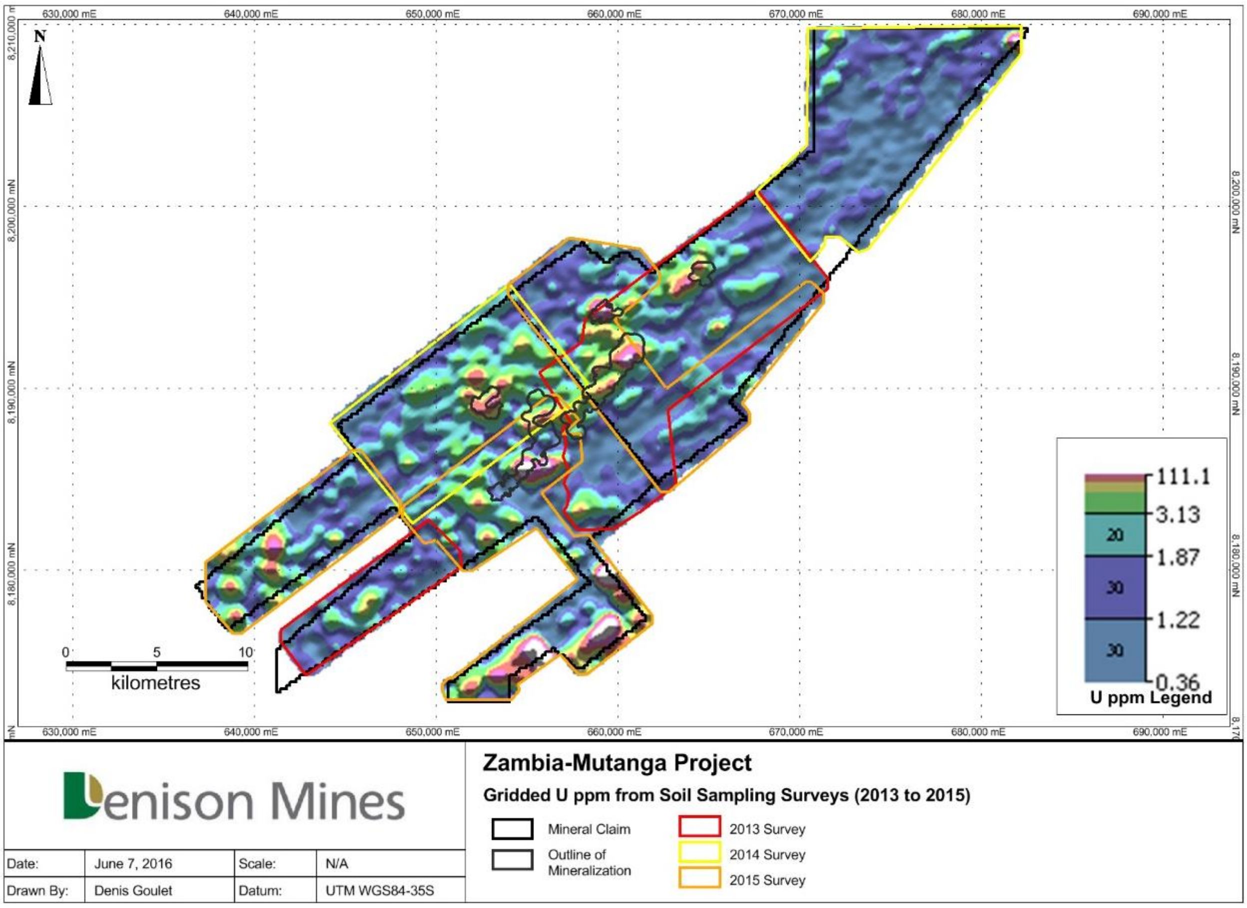

Figure 9-6: Gridded soil uranium results

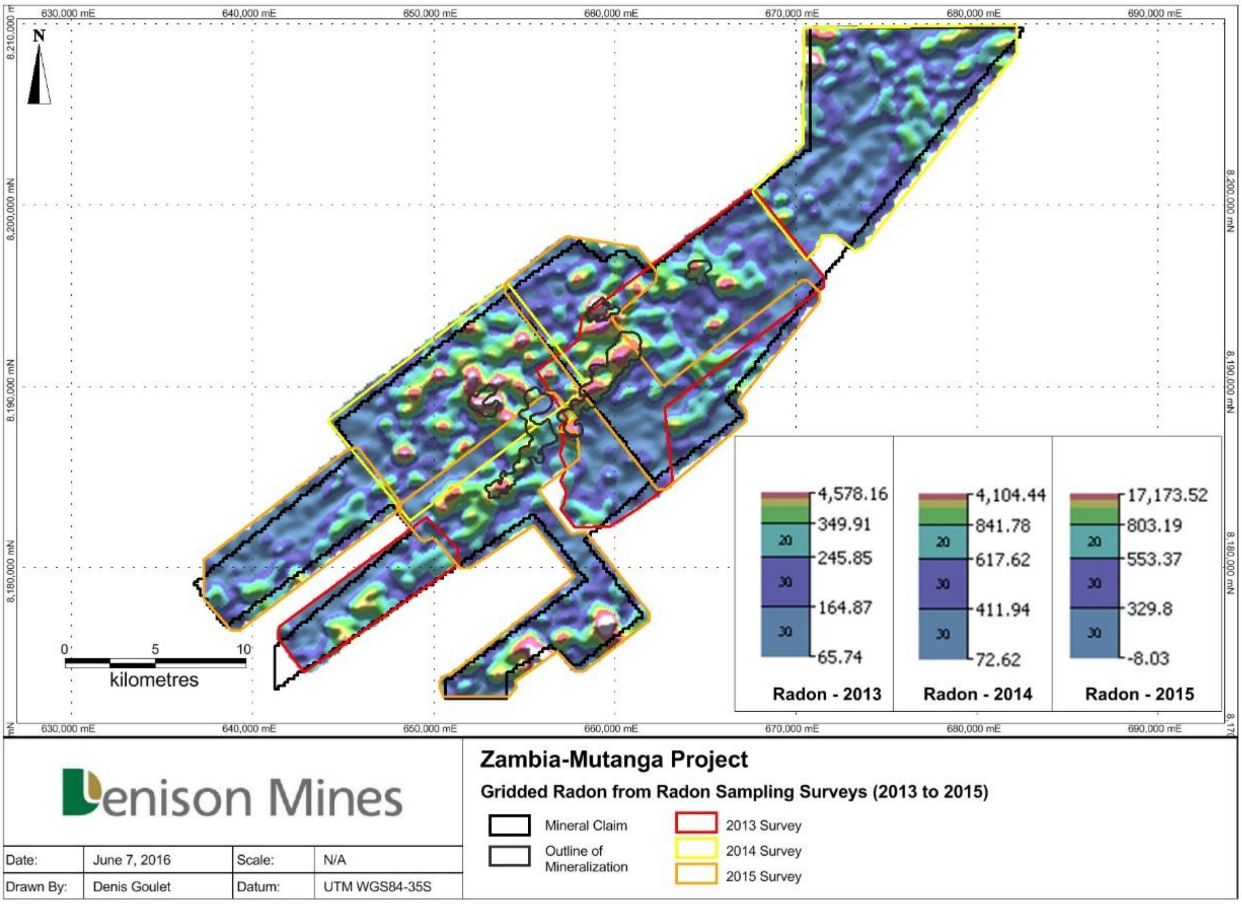

Figure 9-7: Gridded radon results

Trenching was undertaken to test for additional mineralised horizons outside of the drill-defined uranium deposits. The trenching provided a cost-effective follow-up methodology, before any drilling, to test targets generated from the soil geochemistry and radon surveying. Trenches provided a means of accessing the fresh bedrock, or otherwise saprock, for the in-situ determination of geology and mineralisation.

Trenches were located over priority targets based on the interpretation of the soil geochemical and radon results from 2013. 2014 and 2015. Targets considered a combination of airborne or ground radiometric anomalies and 2014 geological mapping. Trenches were typically located along, and parallel to, the soil and radon survey lines which were roughly perpendicular to stratigraphic strike and known mineralisation. The soil and radon anomalies tended to follow stratigraphic strike parallel trends. Trenches were designed to cover the entire anomaly and to extend into the background by $^{1}/_{3}$ to $^{1}/_{2}$ of the anomaly width in each direction. A summary of the trenches excavated in 2014 and 2015 is provided in Table 9-2. Trench locations are provided in Figure 9-8.

Table 9-2: Summary of the trenches excavated in 2014 and 2015

Trench number	Target area	Year	Length [m]	Average depth [m]
MCT1	Manchavwa	2014	900	1.3
MCT2	Manchavwa	2014	966	1.6
MCT3	Manchavwa	2014	853	2
MET4	Muntanga East	2014	708	1.5
MET5	Muntanga East	2014	707	1.2
MET6	Muntanga East	2014	698	2
A-1	Kanyanga	2015	242	1.5
A-2	Kanyanga	2015	200	1
C&D-1	Muntanga East	2015	274	1
C&D-2	Muntanga East	2015	202	1.5
E-1	Dibbwi Muntanga Corridor	2015	420	2
E-2	Dibbwi Muntanga Corridor	2015	146	2
F-1	Dibbwi North	2015	623	3
G-1	Dibbwi West	2015	182	2
G-2	Dibbwi West	2015	332	2.5
H-1	Dibbwi West	2015	900	3.5
H-2	Dibbwi West	2015	210	3
H-2a	Dibbwi West	2015	86	1
H-3	Dibbwi West	2015	216	2
I-1	Kanyanga	2015	192	1.5
I-2	Kanyanga	2015	74	1
Total			**9 131**	

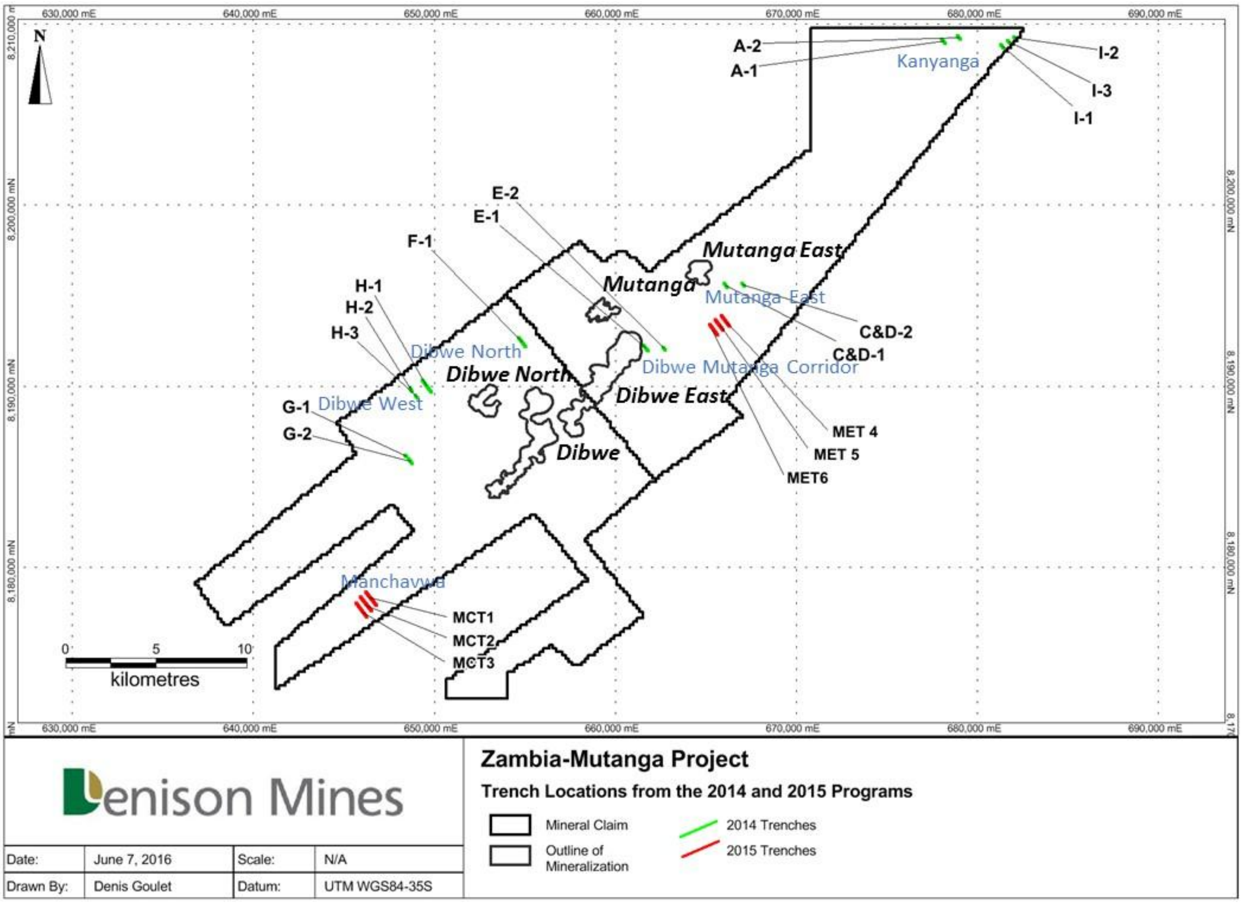

Figure 9-8: Trench locations

Note: This map contains historical spelling. “Dibwe” = Dibbwi, “Mutanga” = Muntanga

Trenching was undertaken using an excavator and allowed sufficient width (approximately 1m) to allow the geologist to work within the trench for mapping and sampling. Trenches were excavated into relatively fresh bedrock and roughly parallel to the regolith-bedrock contact. Where possible, bedrock in the sidewall of the trench was exposed to allow for structural geology measurements.

Trenches were viewed as ‘horizontal drill holes’ in terms of the information collected along them. A 100 m tape was laid out along the base of the trench as a reference. Before commencing trench mapping and sampling the trenches were cleaned from excessive soil or rubble. Trench mapping utilised the same logging codes as used previously for Muntanga drilling in terms of lithology, structure, alteration and mineralisation.

Continuous total gamma scintillometer readings were taken along the base of the trenches. The readings were visually averaged and recorded for every 2 m interval. The maximum total gamma reading and its location for the interval were recorded.
Trench sampling was undertaken over intervals where elevated gamma readings were encountered. For each trench, an elevated gamma threshold was established using log probability plots. Continuous-chip sampling was undertaken from the base or sidewall of the trench where bedrock was exposed. The sample intervals ranged from a maximum of 2 m to a minimum of 50 cm and were adjusted for geological contacts. At least two samples of 2 m each were collected on either side of elevated gamma zones as ‘shoulder samples’. Samples were approximately 1 kg in weight. A scintillometer reading was taken of the bagged sample away from other samples and in an area of low background. A field duplicate sample was collected every 20th sample (5 % field duplicates) and a coarse crush blank was inserted every 25th sample (4 % blanks). Trenching data and results have been stored in an Access database.

Table 9-3: Summary statistics of trench total Gamma and uranium

Trench number	Average Gamma [cps]	Maximum Gamma [cps]	Count of assay samples	Average U ppm	Minimum U ppm	Maximum U ppm	Standard deviation U ppm
A-1	337	1 750	1	30	30	30	
A-2	288	620	0				
C&D-1	254	500	0				
C&D-2	297	620	0				
E-1	527	1 330	49	20	2	55	13
E-2	306	380	0				
F-1	474	2 300	61	17	2	68	13
G-1	695	2 630	45	27	2	124	23
G-2	428	1 500	38	5	2	10	2
H-1	447	1 850	73	13	2	49	9
H-2	371	850	2	6	6	7	0
H-2a	537	1 030	13	19	6	32	9
H-3	378	1 130	7	13	2	27	10
I-1	406	1 200	4	11	8	17	4
I-2	418	1 050	4	16	13	19	3
MCT1	336	1 134	88	10	1	33	7
MCT2	348	2 129	86	11	1	30	6
MCT3	367	1 519	119	11	1	69	9
MET4	373	1 334	112	7	1	39	5
MET5	435	2 098	74	23	1	65	18
MET6	354	1 549	66	13	1	52	11

Figure 9-9: Average and maximum total Gamma readings for 2014 trenches (0 m represents the southern end of the trench)

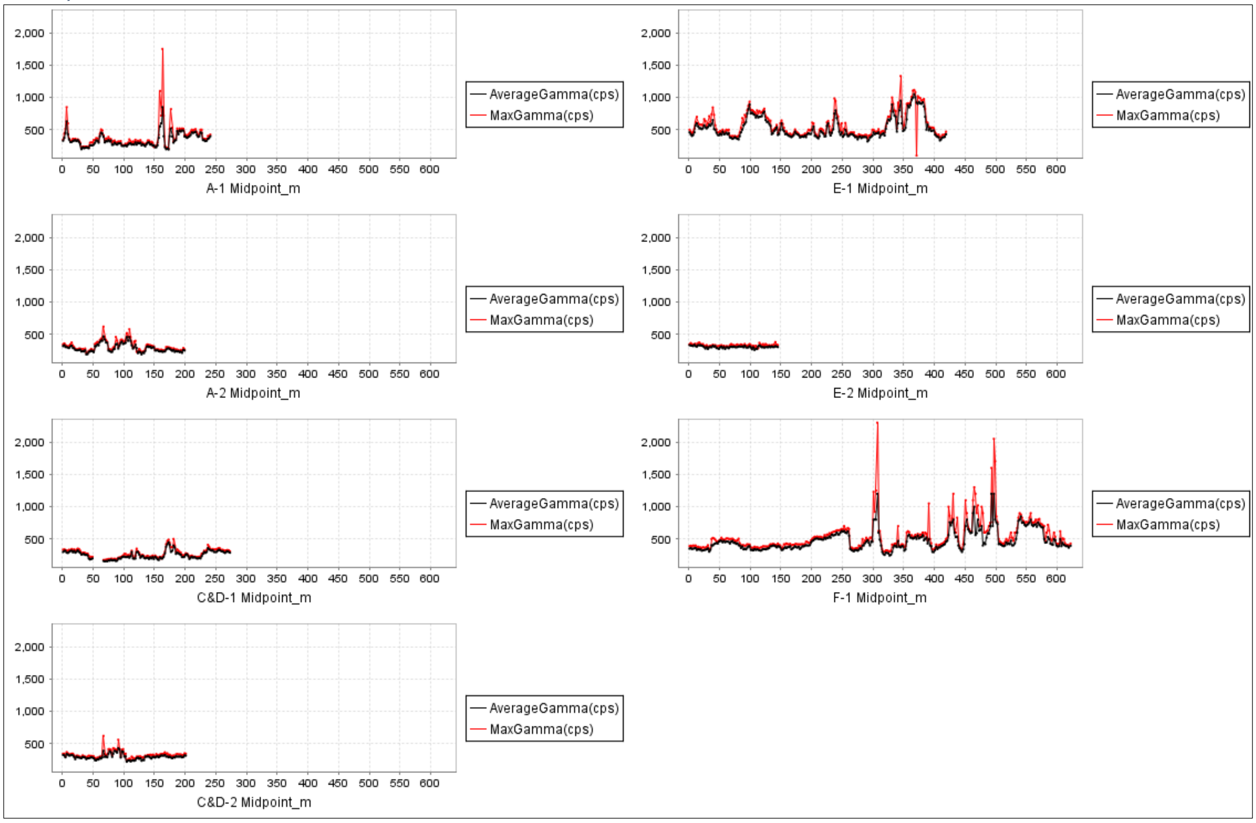

Figure 9-10: Average and maximum total Gamma readings for 2015 trenches (A, C&D, E & F target areas; 0 m represents the southern end of the trench)

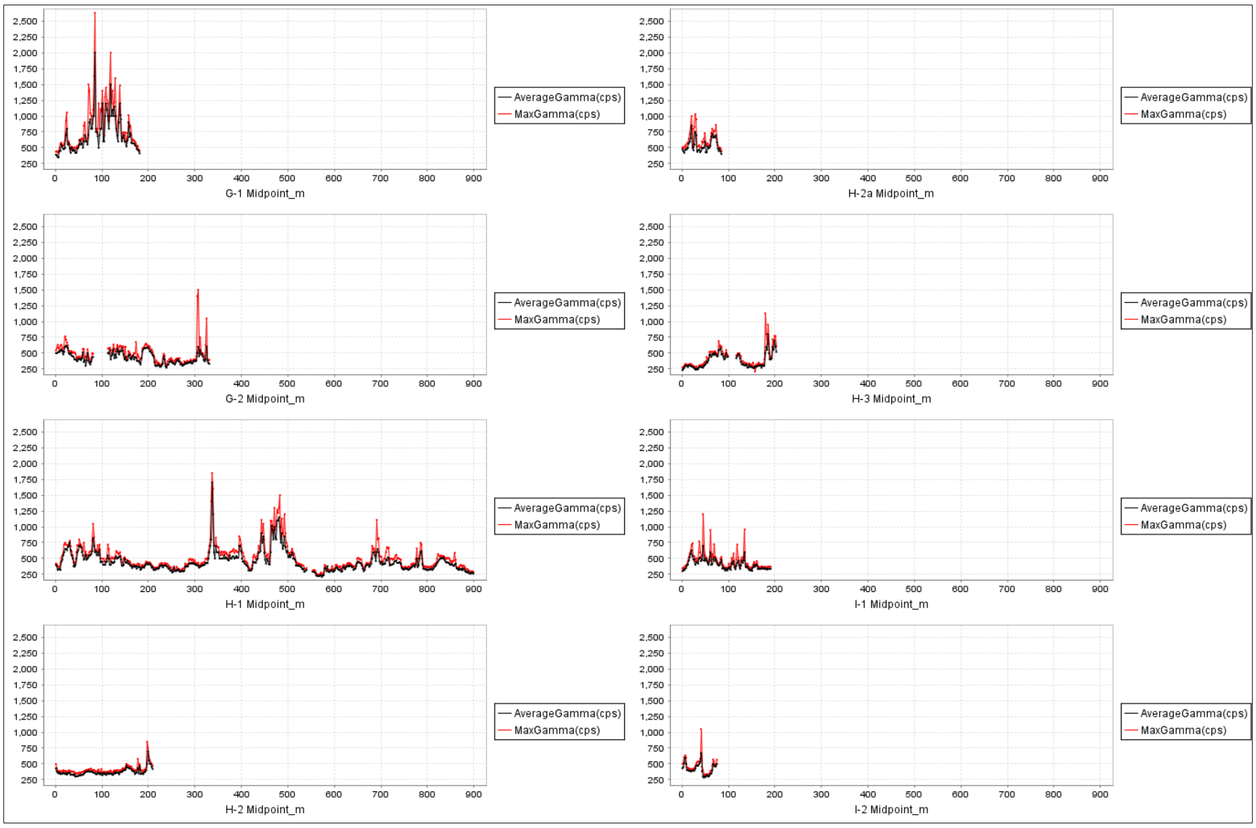

Figure 9-11: Average and maximum total Gamma readings for 2015 trenches (G, H & I target areas; 0 m represents the southern end of the trench)

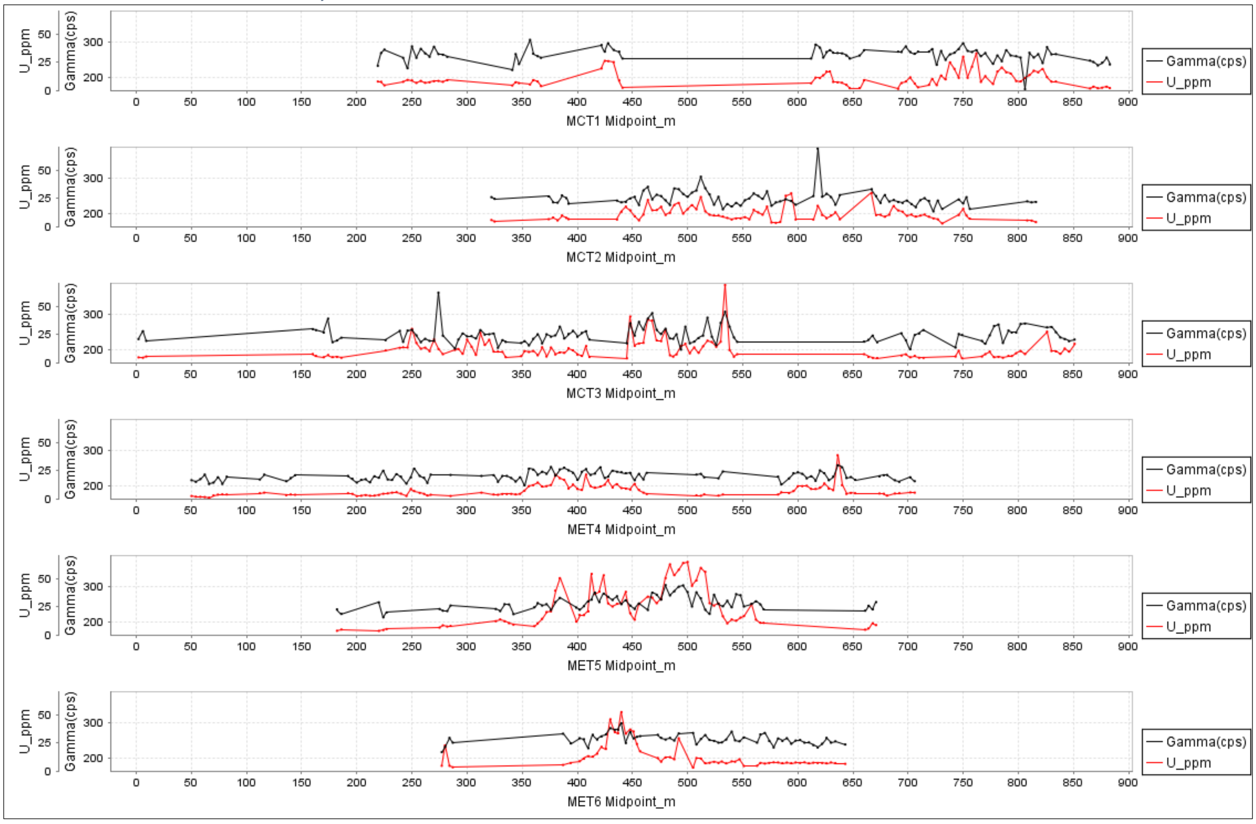

Figure 9-12: Uranium assay and sample total Gamma readings for 2014 trenches (0 m represents the southern end of the trench)

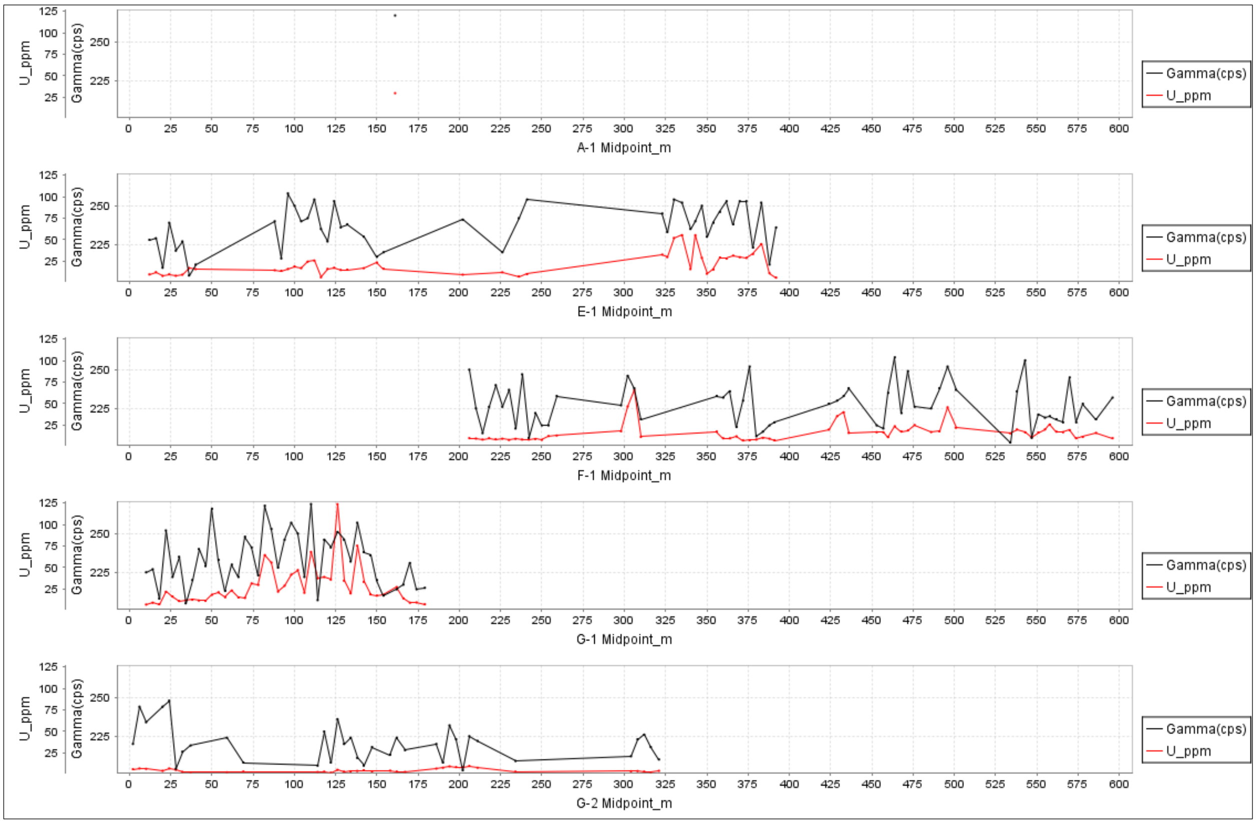

Figure 9-13: Uranium assay and sample total Gamma readings for 2015 trenches (A, E, F & G targets; 0 m represents the southern end of the trench)

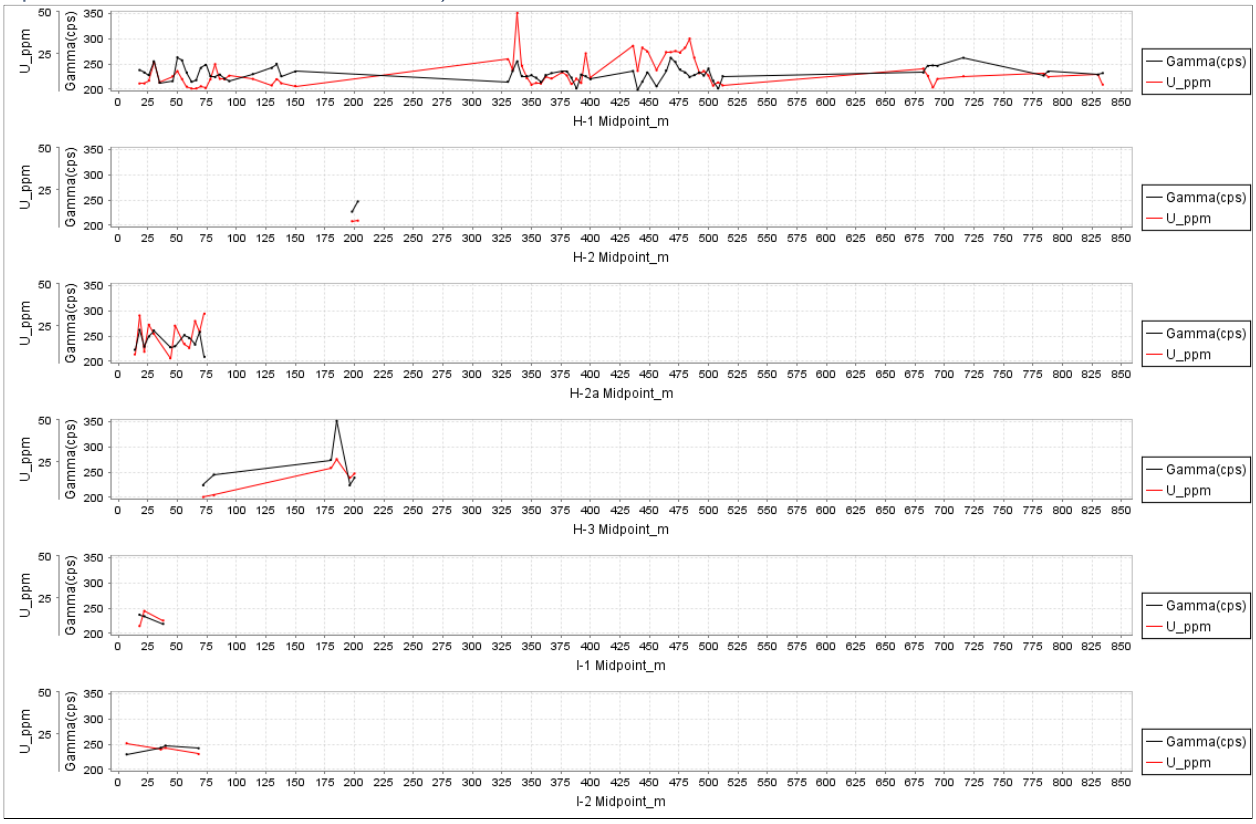

Figure 9-14: Uranium assay and sample total Gamma readings for 2015 trenches (H & I targets; 0 m represents the southern end of the trench)

Weak mineralisation was encountered in the majority of the trenches and a few distinct mineralised horizons were discovered (Table 9-3 and Figure 9-9 to Figure 9-14). Leaching at the regolith-bedrock interface where trench samples were collected may be the reason higher grades were not encountered.

In 2021, GoviEx drilled 12 vertical down-the-hole (“DTH”) hammer holes to a depth of 120 m each over the trenches at Muntanga East (MTD 4,5 and 6), as they are along strike from the Dibbwi East deposit. Unfortunately, the results were disappointing, and no uranium was encountered at depth.

The soil and radon anomalies generated from 2015 surveys warrant follow-up, either through additional trenching or percussion drilling. Geological mapping and ground-truthing are recommended before trenching or drilling.

In 2024, GoviEx had an extensive sterilisation and exploration drilling programme over areas where infrastructure is planned to be built at Muntanga, Dibbwi East, Dibbwi, Njame and Gwabi, and in areas designated for the relocation of villages affected by the potential development of the proposed Muntanga Mine. A total of 102 drillholes, amounting to 9 110 m, were drilled, and all results returned negative for significant mineralisation.

Details of all drilling activities are described in Section 10.

9.3. Gwabi and Njame

The earliest known exploration for uranium in the area covering the Gwabi and Njame deposits was conducted by AGIP in the late 1970s to the mid-1980s. AGIP completed a major regional programme of ground radiometric surveying which identified numerous radiometric anomalies in the area along the northern shores of Lake Kariba. A number of these anomalies were evaluated with more detailed ground radiometric surveying and a small number were subsequently tested with rotary percussion drilling, wagon drilling (“WD”) and in some cases diamond drilling.

AGIP ceased their work in Zambia in 1985, and no further uranium exploration was undertaken in the vicinity of the Gwabi and Njame deposit area until AFR commenced work in 2005.

Albidon (Zambia) Limited acquired the Mugoto PLLS.250 tenement in June 2005 as part of their Munali nickel project tenement holding. The tenement was subsequently transferred to Albidon Exploration Limited in 2006 with Ministerial approval. In October 2005, Albidon Exploration Limited signed a joint venture agreement with AFR under which the latter would explore the eastern part of the Mugoto PLLS for uranium, coal and coal bed methane. This is the area in which both the Gwabi and Njame deposits are located.

AFR undertook a major exploration programme from 2006 to 2007, which included:

- Drilling at the Njame deposit which identified additional uranium mineralisation to that defined by AGIP
- An airborne radiometric survey identified a significant uranium anomaly at Gwabi; this was tested with surface radiometric surveying, soil sampling and
- Subsequent drilling at Gwabi which outlined uranium mineralisation.

Through 2008 and 2009, AFR then completed a series of infill drilling programs, comprising reverse circulation (“RC”) and diamond drilling (“DDH”) to define the extent of both the Njame and Gwabi deposits, as well as tighten the drilling patterns to improve confidence in the geological and Mineral Resource models.

In 2022, Rocketmine from South Africa were contracted to carry out a photogrammetry and LIDAR survey using a drone platform. The areas selected for surveying covered each of the deposit areas at Dibbwi, Dibbwi East-Muntanga, Njame and Gwabi. The LIDAR data have been used in the current MRE to define the ground surface.

9.4. Nabbanda

In 2020, GoviEx geologists carried out a mapping, hand-held radiometric survey and soil sampling programme along traverses over the Nabbanda licence. A total of 21.1 km of traverses were covered. Samples were collected at 50m intervals, along with geological information. Geological contacts and structures were mapped as they were encountered. The purpose was to define any anomalies that could link up the adjoining mining licences. Soil and radiometric results showed a number of anomalies that were eventually followed up by drilling in 2024.

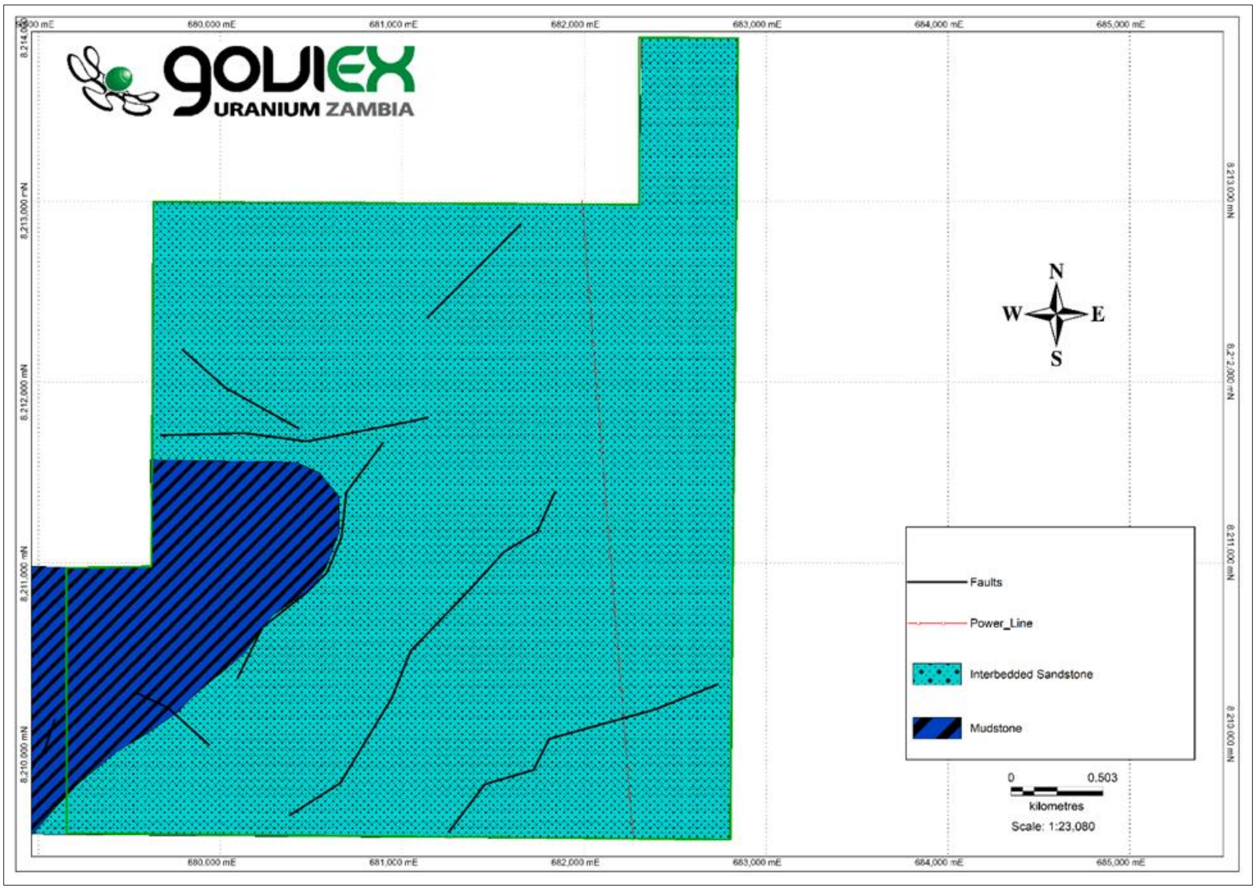

Figure 9-15: Map of Nabbanda geology

In 2024, five drillholes were completed to a maximum depth of 100 m on the Nabbanda exploration licence (22803-HQ-LEL) targeting radiometric and soil anomalies identified in 2020 by GoviEx geologists. One drillhole returned an anomalous uranium value of 600 ppm over 3 m at a depth of approximately 30 m.

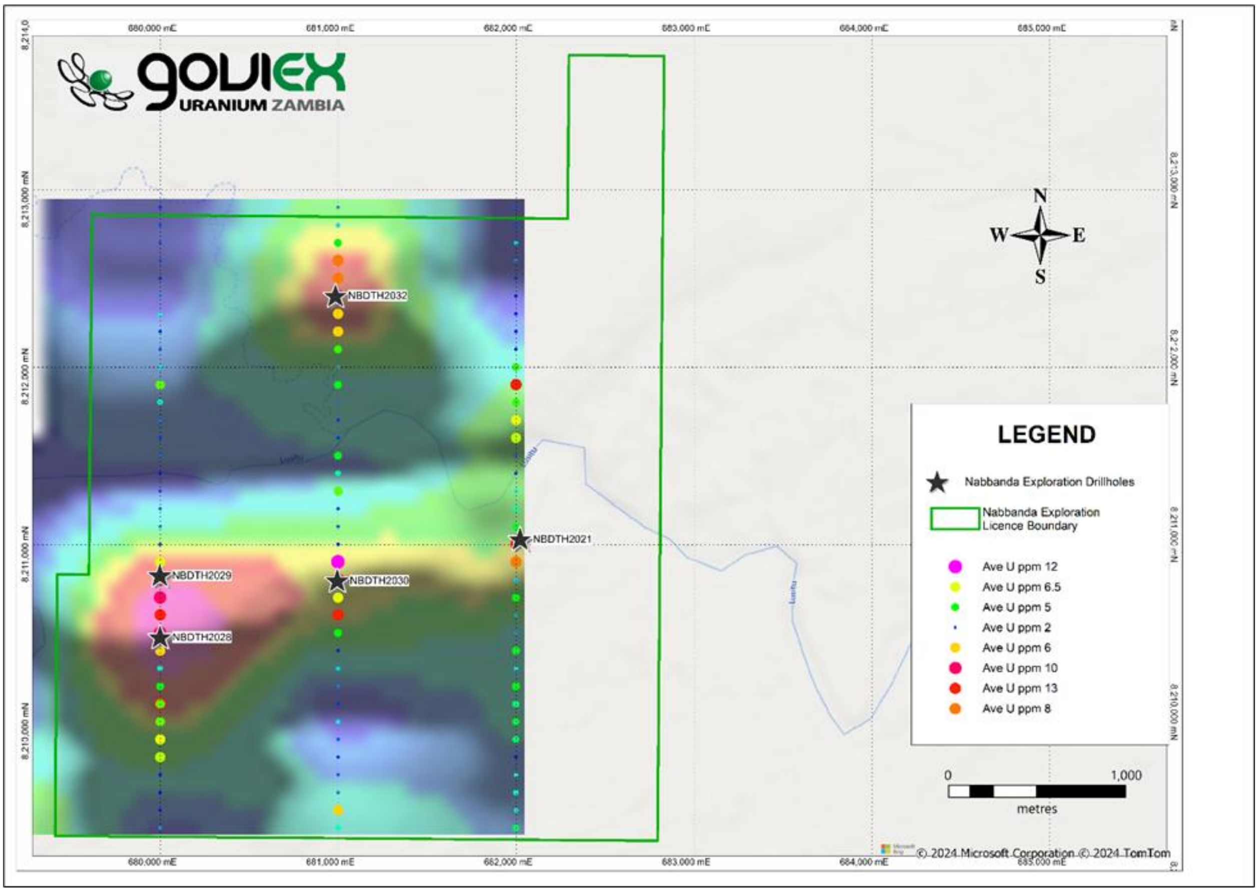

Figure 9-16: Map of Nabbanda drill locations and soil sample locations over gridded uranium in soils

10. Drilling

10.1. Introduction

Drilling at the Dibbwi East, Dibbwi, and Muntanga deposits was completed in three major phases. Historically, drilling was conducted by AGIP and the Zambian Geological Survey (1973 to 1984), followed later by OmegaCorp and Denison (2006 to 2012), and most recently by GoviEx between 2021 and 2024, which was predominately comprised of infill drilling at Dibbwi East and limited confirmation drilling at the Muntanga and Dibbwi deposits. In 2024, the drilling consisted mostly of sterilisation drilling around the proposed infrastructure and relocation sites.

Drilling at the Gwabi and Njame deposits was managed by AFR and completed between 2006 and 2009. GoviEx conducted limited drilling at Njame and Gwabi from 2022 to 2024.

Summaries of annual drilling completed on the main deposit areas are provided in Table 10-1 to Table 10-5. A summary of drilling completed on areas adjacent to the main deposits is provided in Table 10-6. Types of drilling techniques used on the Project include diamond core drilling (“DD/DDH”) and percussion style drilling which includes reverse circulation (“RC”), DTH hammer, air core (“AC”), and percussive wagon drill (“WD”).

Table 10-1: Dibbwi East deposit drilling summary

Year	DDH	DDH metres	Percussion holes	Percussion metres
1980	14	3 575	0	0
2008	49	3 602	27	2 009
2011	34	3 842	98	10 438
2012	29	4 151	29	3 792
2021	0	0	49	5 980
2022	35	4 699	158	21 725
2023	13	1 700	115	13 254
2024	0	0	32	2 780
Total	**174**	**21 569**	**508**	**59 978**

Table 10-2: Dibbwi deposit drilling summary

Year	DDH	DDH metres	Percussion holes	Percussion metres
1980	33	3 300	40	5 266
2006	0	0	25	1 362
2007	27	1 682	1	110
2008	140	12 914	114	7 343
2010	9	495	0	0
2012	6	1 101	14	1 681
2022	3	300	0	0
2023	4	401	3	360
2024	0	0	7	640
Total	**222**	**20 193**	**204**	**16 762**

Table 10-3: Muntanga deposit drilling summary

Year	DDH	DDH metres	Percussion holes	Percussion metres
1980	47	4 406	180	6 621
2005	7	332	0	0
2006	32	1 788	70	2 052
2007	32	1 897	9	540
2008	207	11 391	263	14 168
2010	6	313	0	0
2012	1	293	2	300
2022	11	610	0	0
2023	7	454	44	2 830
2024	0	0	44	4 200
Total	**350**	**21 484**	**612**	**30 711**

Table 10-4: Njame deposit drilling summary

Year	DDH	DDH metres	Percussion holes	Percussion metres
2006	0	0	63	2 794
2007	28	1 412	255	14 617
2008	126	6 113	258	14 822
2009	0	0	80	3 540
2022	3	150	0	0
2023	5	440	4	326
2024	0	0	11	800
Total	**162**	**8 115**	**671**	**36 899**

Table 10-5: Gwabi deposit drilling summary

Year	DDH	DDH metres	Percussion holes	Percussion metres
2007	5	200	226	10 905
2008	34	1 168	54	1 628
2022	3	150	0	0
2023	4	330	4	225
2024	0	0	5	220
Total	**46**	**1 848**	**289**	**13 008**

Table 10-6: Summary of Annual Exploration Drilling Campaigns Conducted in Areas Adjacent to the Main Deposits

Year	DDH	DDH metres	Percussion holes	Percussion metres
1980	56	5 495	214	14 276
2006	0	0	60	3 679
2008	18	1 352	330	19 924
2009	0	0	59	2 980
2010	0	0	18	775
2011	3	242	11	775
2012	24	2 936	36	4 245
2023	37	925	24	2 095
2024	0	0	99	8 670
Total	**138**	**10 950**	**851**	**57 419**

10.2. Muntanga, Dibbwi and Dibbwi East deposits

10.2.1. Historical drilling

Prior to 2006, AGIP and the Zambian Geological Survey undertook drilling across the Muntanga and Dibbwi licence areas (circa 1980). Several hundred drill holes were completed, and the main known deposits were identified, along with a number of prospects. However, due to insufficient historical records being available to verify the reliability of these data, all drill hole information from the time frame has been excluded from the MRE process.

During the OmegaCorp/ Denison tenure (2006 to 2012), RC and DD were the principal methods of exploration and delineation drilling after initial geophysical surveys. Drilling was generally conducted during the dry season. Well-established drilling industry practices were used in the drilling programs. Drill holes were numbered with a prefix of the Project (DM), followed by type (C-rotary, D-diamond), followed by the hole number, with almost all drill holes being drilled vertically or at 70° from the surface to the target at depth.

In 2006, OmegaCorp drilled DDH to twin previous drilling at the Muntanga deposit. Results confirmed the broad tenor of the earlier mineralised intercepts.

From 2007 to 2008, Denison completed work on the Muntanga deposits, focussing on the Muntanga and Dibbwi areas. The work included an appraisal of all available data (maps, plans, sections, limited geological interpretations, radiometric, and AGIP historical MREs). From this information, Denison produced several databases covering Muntanga along with other prospects.

Denison commenced drilling operations on July 16, 2008. The purpose of the drilling programme was to:

- Provide first-pass exploration data for the radiometric anomalies identified by the 2006 and 2008 airborne geophysics programmes, and
- Provide bulk sample material for metallurgical test work.

After a two-year delay due to the suspension of exploration activities, a two-phase drilling campaign resumed in April 2011. Phase 1 drilling on Dibbwi East and Muntanga targets commenced in April and ended in July 2011. The results for Phase 1 confirmed the continuity of uranium mineralisation identified in the 2008 drilling programme at Dibbwi East, with a northeast-southwest strike length greater than 2.5 km.

Based on the encouraging results obtained with the Phase 1 drilling over the Dibbwi East area, a Phase 2 drilling programme was completed between August to October 2011. This drilling programme discovered primary mineralisation at depth and increased the strike length to 4.0 km.

In 2012, the primary targets for drilling were the Dibbwi East, Dibbwi and Muntanga deposit areas, to further delineate and infill within the deposit footprints.

The locations of historical drill holes completed between 1980 and 2012 across the Muntanga and Dibbwi licence areas are shown in Figure 10-1.

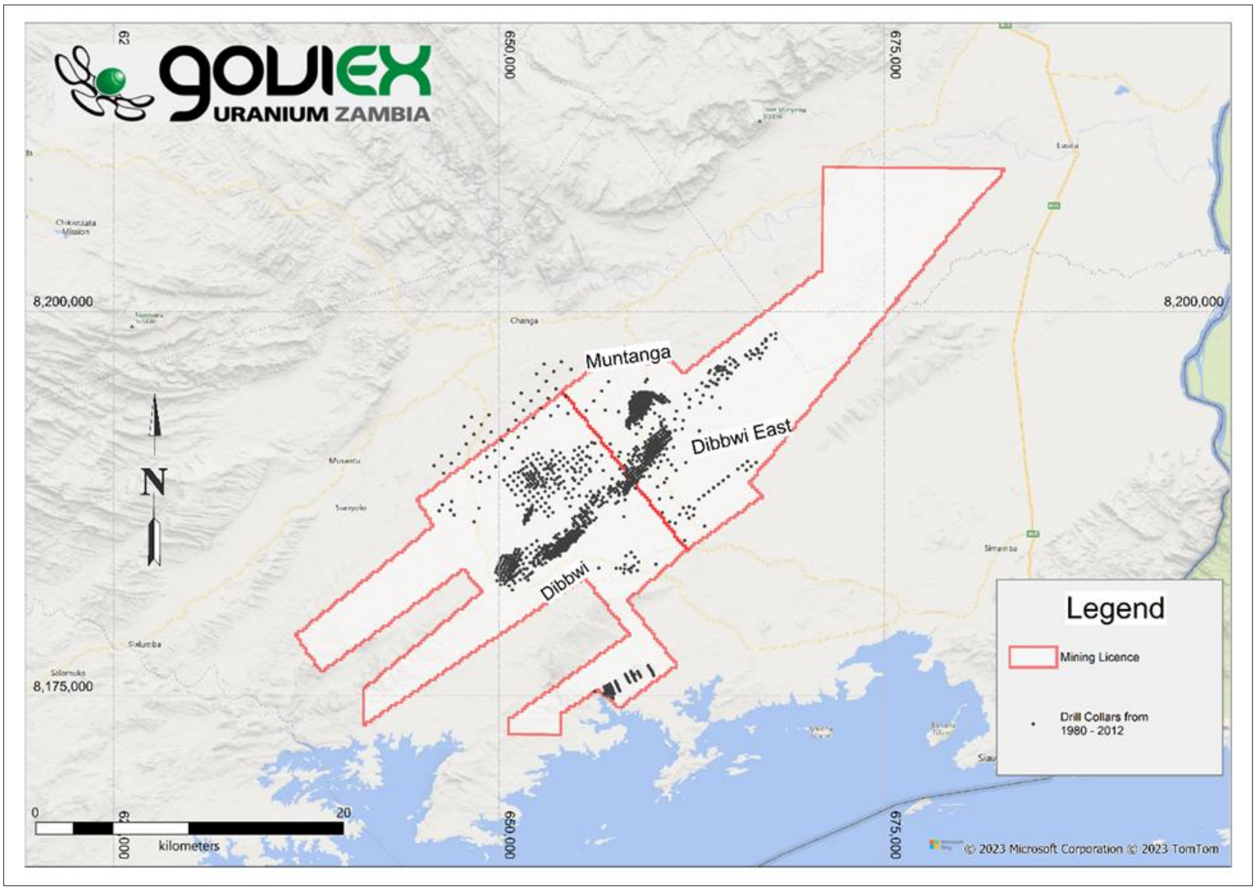

Figure 10-1: Historical drill hole location map

10.2.2.GoviEx drilling

During the 2021 to 2023 drilling campaigns, GoviEx carried out drilling mostly on the Dibbwi East deposit to infill the existing drill pattern to a 100 m line spacing with drill holes at 50 m between holes. Selected areas were drilled at a closer spacing of 25 x 25 m to assess the continuity of mineralisation for MRE purposes. Most of these drill holes were drilled using an open hole DTH method as it is a cost-effective and quick drilling technique. All uranium grade data for DTH holes were determined using a downhole gamma probe. DDH made up approximately 10 % of the total drilling meterage, with a number of holes drilled to collect metallurgical samples, and others drilled to twin historical holes for data validation purposes. DDH were drilled on all deposits by GoviEx during the 2021 and 2022 drilling campaigns.

The 2023 drilling programme was driven by the success of the 2021 to 2022 exploration efforts and the updated MRE reported on July 17, 2023. During the second half of the year, a total of 15 835 m of infill drilling was conducted across 160 drill holes, primarily at the Dibbwi East and Muntanga deposits. The focus was on upgrading Inferred Mineral Resources into the indicated category by improving drill hole spacing and expanding the Dibbwi East open pit Mineral Resource. DD was undertaken to validate gamma and radon corrections in downhole logs, with twin holes drilled to match earlier percussion drilling. Additionally, 14 geotechnical holes were completed across all deposits to optimise pit wall geometry, with samples sent to Rocklab in South Africa for analysis. Hydrogeological work included 29 holes to assess dewatering needs, along with nine water bores and pump testing to support future water supply planning. Geotechnical investigations involved the digging of 119 shallow test pits to evaluate soil characteristics for infrastructure design, including leach pads and waste dumps.

Building on the progress made in 2023, the 2024 drilling programme focused on sterilisation drilling around proposed pit areas, mine plant locations, and resettlement zones. The drilling was done to ensure the identification of non-mineralised zones suitable for infrastructure placement and community relocation. The company did hydrogeological drilling to secure water supply for the processing plant, supported by groundwater modelling that has indicated sufficient aquifer resources near the deposits. These activities further refine project planning and align with the broader pre-construction objectives.

Drill holes completed by GoviEx during the 2021 to 2024 campaigns are shown in Figure 10-2 to Figure 10-4.

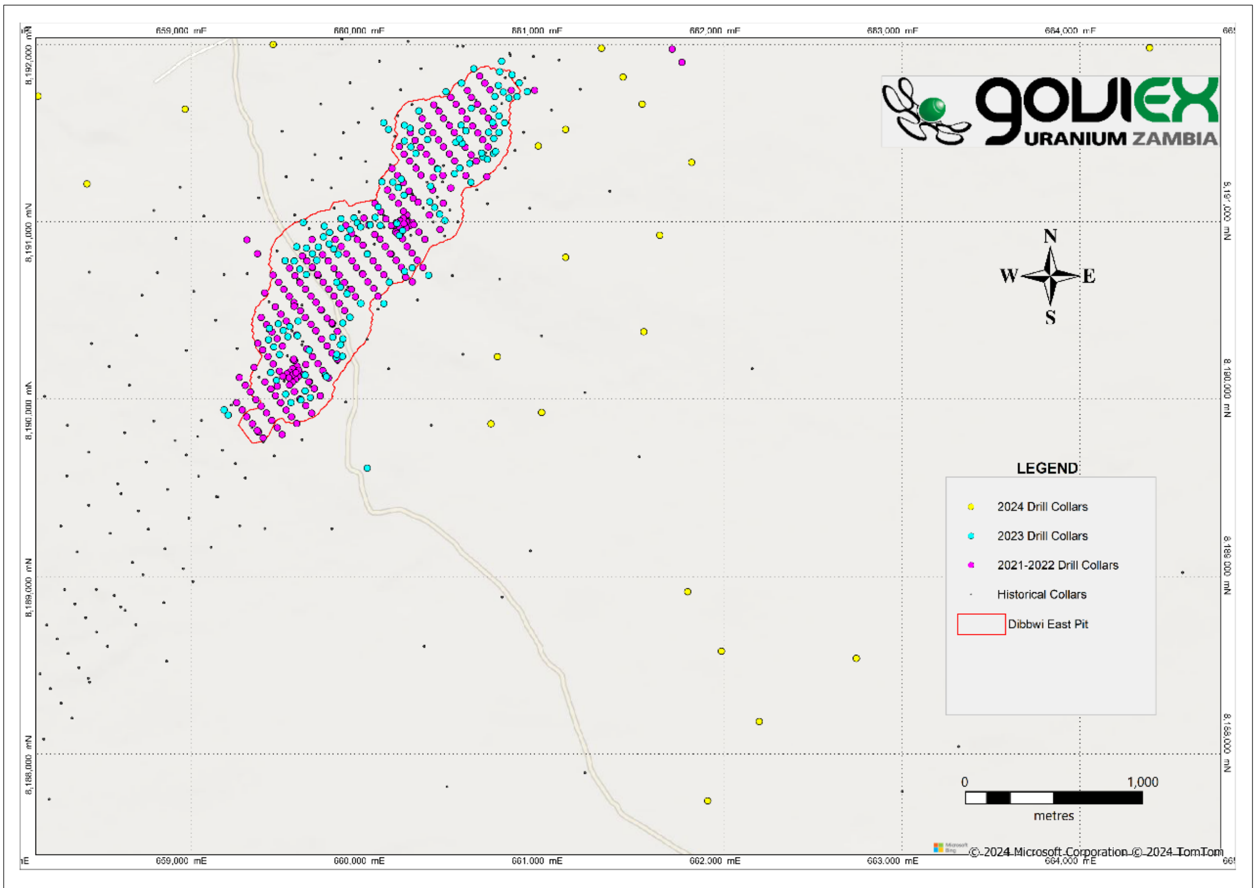

Figure 10-2: GoviEx drill hole location map for Dibbwi East

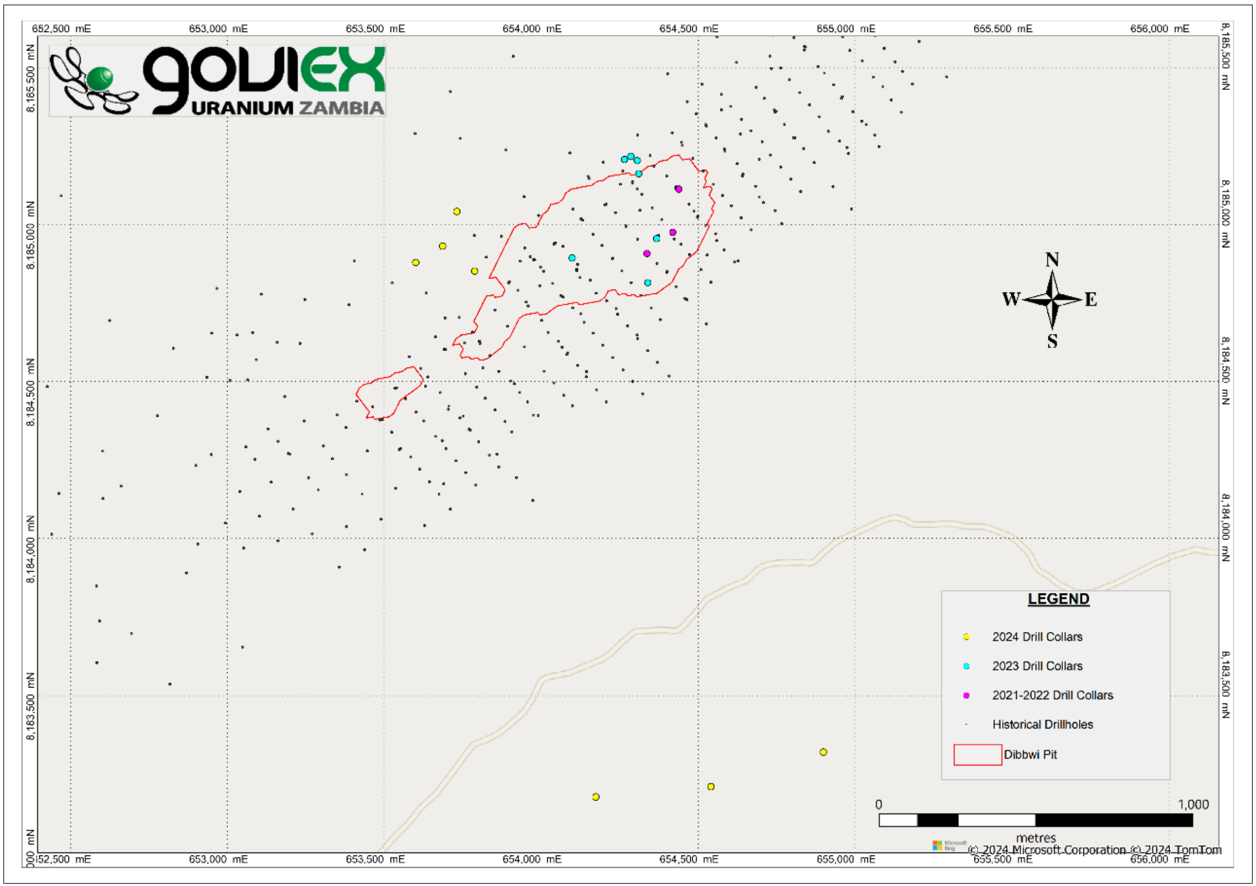

Figure 10-3: GoviEx drill hole location map for Dibbwi

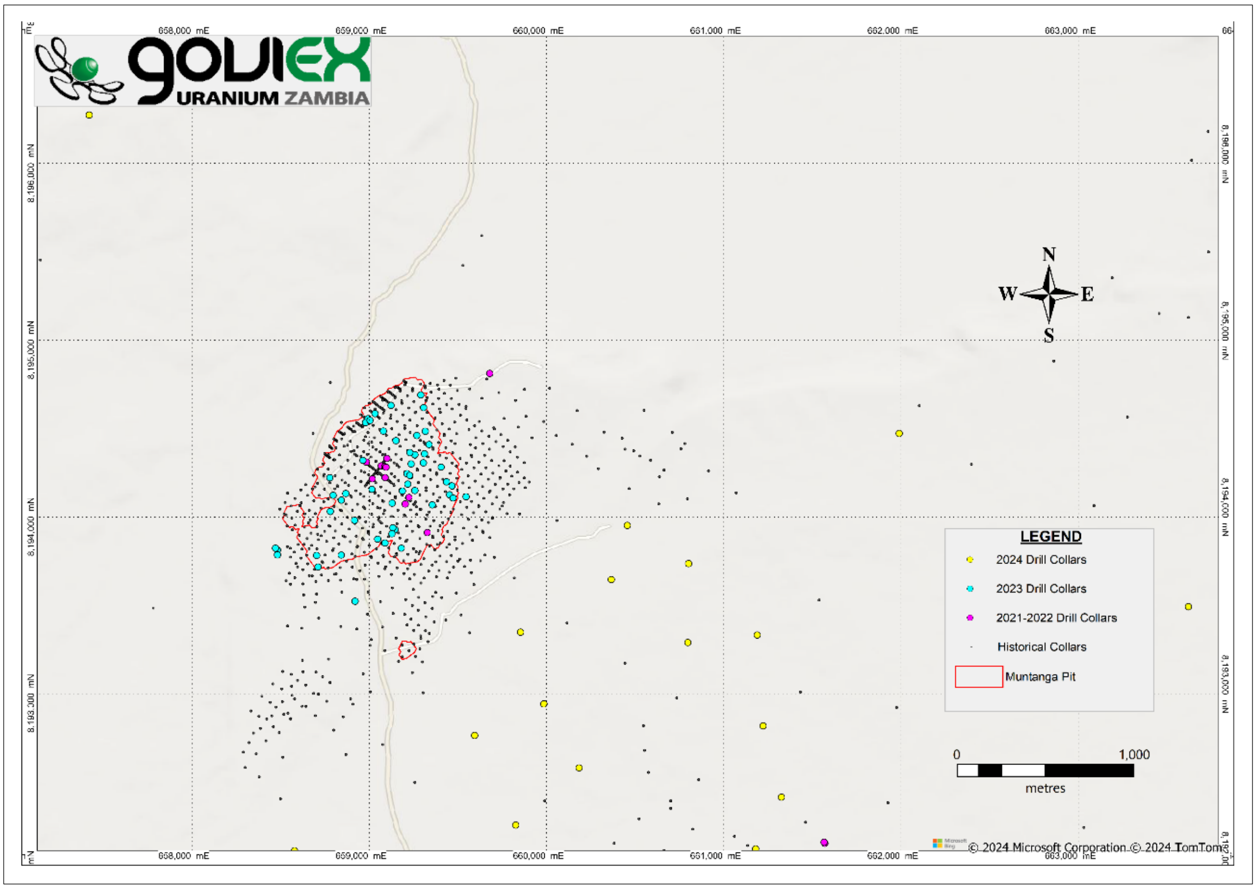

Figure 10-4: GoviEx drill hole location map for Muntanga

10.2.3.Down-hole deviation surveys

Historically, all holes were drilled vertically, and no down-hole survey data were available for historic drilling prior to the 2006 OmegaCorp drilling campaigns. However, the amount of deviation is negligible as holes were relatively shallow, with depths averaging 40 m and ranging from 10 m to 110 m; and stratigraphic bedding is relatively flat and rock competency low.

OmegaCorp's 2006 and Denison's 2007 to 2012 drilling campaigns consisted of DDH and RC drilling, predominately drilled vertically, along with some inclined holes. Limited checks on hole deviation demonstrated deviations of less than 2°. All DDH were drilled at angles ranging from 55° to 80°, and at a number of azimuths although dominantly towards 135° or 315°. Down-hole survey measurements were taken using a single-shot camera at 15 m down-hole intervals.

During the 2021 and 2022 GoviEx drilling campaigns, down-hole deviation surveys were conducted using a Boart Longyear Trushot digital survey tool. Deviation survey measurements were done at 5 m to 10 m interval spacing depending on the total depth of the hole.

Core orientation was conducted using a Boart Longyear Trucore UPIC orientation tool and down-hole spear. Orientation of the drill core was completed on every drill run for the DDH.

10.2.4.Logging and sampling

In general, the core logging and sampling methodologies used by GoviEx closely follow the practices used by Denison, with only minor changes to how data are collected and stored.

10.2.4.1. Scintillometer logging

All drill cores and chips were systematically logged with a Terraplus RS-125 Gamma-Ray Spectrometer/ Scintillometer. The general concept behind the scintillometer is similar to the gamma probe except the radiometric pulses are displayed on a scale and the respective count rates are recorded manually by the technician logging the core or chips. The hand-held scintillometer provides a qualitative measurement of uranium mineralisation only and cannot be used to calculate equivalent uranium grades. However, it does allow the geologist to identify uranium mineralisation in the core and to select intervals for geochemical sampling. The scintillometer readings are used by the geologists to depth match the core depth with the geophysical depths, to ensure alignment between assay grades and geophysical-derived equivalent grades.

10.2.4.2. Reverse circulation - Down-the-hole logging and sampling

Drill chip samples from RC and DTH drilling were laid out in piles next to the rigs for geological logging. They were logged for lithology, grain size, alteration, and colour. Representative samples were collected in chip trays for eventual relogging if required and storage at the Muntanga Camp core yard.

During Denison's tenure, all percussion chips were collected via a cyclone and split on-site at the time of drilling. The cuttings for each metre were put through a riffle splitter to give an approximate 1.5 kg primary sample, an approximate 1.5 kg field duplicate and, depending on the hammer size, a residual bulk sample of approximately 15 kg to 20 kg. Approximately 10 % of anomalous intercepts (more than twice the background level of counts per second ("cps") as determined by a handheld scintillometer) in RC holes were selected for assay in 2012.

During the 2005 to 2007 drilling, approximately 1.5 kg primary samples representing anomalous intervals of RC holes that collapsed before they could be probed were also sent for pressed powder x-ray fluorescence ("XRF") analysis.

In 2021 and 2022, no samples were collected from the DTH drilling as this drilling technique is an open-hole technique and therefore does not provide appropriate representative sample material for assaying.

During the 2021 and 2022 campaigns, GoviEx used a similar logging format to that used by Denison, however, Seequent's MX Deposit logging application was used for data entry in the field using tablets. This application stores the data in the cloud such that it is readily accessible anywhere in the world. The data are regularly backed up onto the company's cloud server.

10.2.4.3. Core logging and sampling

All DDH were logged for lithology, structure, alteration, mineralisation and geotechnical characteristics. In 2009, data were entered into DHLogger software on laptops in the field and then transferred into a Fusion database. Hard copies of drill logs are stored at the site.

Prior to core logging, down-hole geophysical probe information is reviewed, with the major lithological contacts, structures and mineralised horizons being inferred from the Gamma and conductivity readings. These inferences are then reviewed alongside the core.

The core is then measured and metre marked, and the core yard technician records core recovery, longest piece and scintillometer readings.

Once the core is marked up, a geologist records the following information directly into DHLogger:

- Lithology (major and minor)
  - Escarpment Grit Formation Package C - B boundary
  - Escarpment Grit Formation Package B - A boundary
  - Escarpment Grit Formation Package A - Madumabisa Mudstone boundary
  - Correlation in the mudstone boundaries (in accordance with cross-section information)
  - Other significant, unusual or potential correlation lithologies
- Alteration
  - Identify zones of limonite, hematite and goethite by colour
- Structure
  - Alpha angles: Most cores are too broken to permit orientation marks and lines so the collection of beta angles (i.e. angle of rotation against a line running down the bottom of the hole) is difficult – or not possible. Thus, the alpha angles (the angle to the long core axis) were recorded. Where possible, at least one bedding plane per tray was recorded, as well as every measurable contact between the key lithologies.
- Faults (other significant, unusual or potential correlation in structures)
  - Mineralisation (in conjunction with WellCAD and Gamlog data)
  - Confirm/ refute high-grade zones (i.e. +700 cps) as indicated by the scint data
  - Attempt to identify uranium mineral species and habit
  - Any other information or comments
  - The core is then photographed wet and dry before being stacked in the core storage area.

At GoviEx in 2021 and 2022, the DDH core data were collected using tablets and the Seequent MX Deposit Application, with data stored directly in the cloud. Local backup and backup to the company's cloud server were carried out regularly. Most of the core mark-ups and photography are done on the drill pad so that the quality of the core is not lost during transport to the core farm. The core is then logged geologically using the descriptions outlined above. The core is then marked up for sampling.

10.2.5. Down-hole geophysical logging

Exploration for uranium deposits in Zambia typically involves the identification and testing of sandstones within reduced sedimentary sequences. The primary method of collecting information is through extensive drilling (both RC and DDH) and the use of down-hole geophysical probes. The down-hole geophysical probes measure the electrical properties of the rock from which lithologic information can be derived and natural gamma radiation, from which an indirect estimate of uranium content can be made. The down-hole geophysical probes measure the following parameters:

10.2.5.1. Conductivity

Conductivity logs measure the electrical conductivity of the soil or rock surrounding the borehole. They provide a detailed measure of changes in conductivity with depth and are also termed electromagnetic (“EM”) induction logs. The electrical conductivity of soil or rock (and its reciprocal, electrical resistivity) depends on the porosity, groundwater conductivity, degree of saturation, clay content, and other bulk soil properties. Hence it is a useful tool in determining the changes with depth of any of these properties. These logs can be very useful in identifying zones of increased groundwater conductivity, often indicative of contaminant concentrations.

10.2.5.2. Resistivity

Resistivity logging is a method of characterising the rock or sediment in a borehole by measuring its electrical resistivity. Resistivity is a fundamental material property which represents how strongly a material opposes the flow of electric current.

10.2.5.3. Self-potential

The self-potential (“SP”) log is a measurement taken to characterise rock formation properties and is particularly useful in mapping sand/shale contacts. The log works by measuring small electric potentials (measured in millivolts) between depths in the borehole and a grounded voltage at the surface resulting from the flow of electrical current in the earth. The change in voltage through the well bore is caused by a buildup of charge on the wellbore walls. Clays and shales (which are composed predominantly of clays) will generate one charge and permeable formations such as sandstone will generate an opposite one. There are many possible sources of these currents. The major source is the different salinity interfaces, such as the borehole fluid (drilling mud) and the formation water (connate water). Whether the mud contains relatively more or less salt compared to the connate water will determine which way the SP curve will go. SP cannot be used for quantitative interpretation.

10.2.5.4. Single point resistance

Single point resistance (“SPR”) measures the electrical resistance (ohms) between a surface electrode and electrode in the down-hole probe. Single-point-resistance logs record the electrical resistance between the borehole and an electrical ground at the land surface. In general, resistance increases with grain size and decreases with borehole diameter, density of water-bearing fractures, and increasing dissolved-solids concentration of borehole fluid. A fluid-filled borehole is required for single-point-resistance logs. SPR logs cannot be used for quantitative interpretation but are an excellent source of lithologic information.

10.2.5.5. Deviation

The deviation is a measurement made to determine the angle from which a hole drilled deviated vertically during drilling. There are two basic deviation survey or drift survey instruments: one reveals the angle of deviation only, and the other indicates both the angle and direction of deviation.

10.2.5.6. Natural gamma

The radiometric (gamma) probe measures gamma radiation which is emitted during the natural radioactive decay of uranium (“U”) and variations in the natural radioactivity originating from changes in concentrations of the trace element of thorium (“Th”) and changes in concentration of the major rock-forming element potassium (“K”).

Potassium decays into two stable isotopes (argon and calcium) which are no longer radioactive and emits gamma rays with energies of 1.46 MeV. Uranium and thorium, however, decay into daughter-products which are unstable (i.e. radioactive). The decay of uranium forms a series of about a dozen radioactive elements in nature which finally decay to a stable isotope of lead. The decay of thorium forms a similar series of radioelements. As each radioelement in the series decays, it is accompanied by emissions of alpha or beta particles or gamma rays. The gamma rays have specific energies associated with the decaying radionuclide. The most prominent of the gamma rays in the uranium series originates from the decay of 214 bismuth (“Bi”), and in the thorium (“Th”) series from the decay of 208 thallium (“Tl”).

The gamma radiation is detected by a sodium iodide crystal, which when struck by a gamma ray emits a pulse of light. This pulse of light is amplified by a photomultiplier tube, which outputs a current pulse which is known as cps. The gamma probe is lowered to the bottom of a drill hole and data are recorded as the tool is withdrawn up the hole. The current pulse is carried up a conductive cable and processed by a logging system computer which stores the raw gamma cps data.

Since the concentrations of these naturally occurring radioelements vary between different rock types, natural gamma-ray logging provides an important tool for lithologic mapping and stratigraphic correlation. For example, in sedimentary rocks, sandstones can be easily distinguished from shales due to the low potassium content of the sandstones compared to the shales. However, the greatest value of the gamma-ray log in uranium exploration is determining equivalent uranium grade.

Because there should be an equilibrium relationship between the daughter product and parent, it is possible to compute the quantity (concentration) of parent 238U and 232Th in the decay series by counting gamma rays from 214Bi and 208Tl respectively. If the gamma radiation emitted by the daughter products of uranium is in balance with the actual uranium content of the measured interval, then uranium grade can be calculated solely from the gamma intensity measurement.

Down-hole gamma data (measured in cps) is subjected to a complex set of mathematical equations, considering the specific parameters of the probe used, speed of logging, size of the borehole, drilling fluids and presence or absence of any type of drill hole casing. The result is an indirect measurement of uranium content within the sphere of measurement of the gamma detector.

The basis of the indirect uranium grade calculation (referred to as "eU_3O_8" for "equivalent U_3O_8") is the sensitivity of the detector used in the probe which is the ratio of cps to known uranium grade and is referred to as the probe calibration factor. Each detector’s sensitivity is measured when it is first manufactured and is periodically checked throughout the operating life of each probe against a known set of standards "test pits" with various known grades of uranium mineralisation or through empirical calculations. In addition, certain boreholes (MTC51600-04) near the Dibbwi East deposit are cased and the probes are periodically checked for any instrument drift. Application of the calibration factor, along with other probe correction factors, allows for immediate grade estimation in the field as each drill hole is logged.

10.2.5.7. Denison gamma grade determination (counts per second to equivalent U_3O_8 grade conversion)

Denison used an in-house developed computer programme known as GAMLOG to convert the measured cps of the gamma rays into an equivalent per cent U_3O_8 (eU_3O_8%). GAMLOG was based on other “standard” grade calculation programs that were developed within the uranium industry using Scott’s Algorithm developed in 1962.

10.2.5.8. GoviEx gamma grade determination (counts per second to equivalent U_3O_8 grade conversion)

Down-hole gamma data collected by GoviEx were converted into eU_3O_8 using the ALT Wellcad software supplied by an external geophysical contractor, Terratec Geophysical Services. The final data were transferred to GoviEx as .csv format files for input into the master drill hole database maintained by GoviEx.

10.2.6. Drill collar survey

All historical data collected prior to 2006 were collected using the UTM Coordinate: Arc 1950 Map Datum, Zone 35S. Drill collar surveys were completed by Datum Surveying Consultants, from Lusaka, Zambia, using a high-precision GPS.

Post 2006, drill collar locations were spotted on a grid and surveyed by differential base station GPS using the WGS84 UTM zone 35S reference datum. Drilling was conducted on a nominal drill hole grid spacing of 200m northeast-southwest by 100m northwest-southeast. Drill collar elevations were estimated by the Denison DGPS system, which was on average approximately 8m lower than the previously used elevation datum for historical holes drilled in the 1980s. As a result, all historical data had been adjusted in elevation to fit the Denison elevation datum at that time.

The base station control points established by Denison and used for drill collar surveys are provided in Table 10-7.

Table 10-7: Differential GPS base station control points

Base station ID	Easting	Northing	Height	Location
DM1	659 694.46	8 194 890.19	613.38	Muntanga Camp
DM2	659 634.44	8 194 801.19	606.79	Muntanga Camp
DM3	653 849.15	8 185 116.71	601.69	Dibbwi Camp
DM4	653 850.01	8 185 238.42	611.42	Dibbwi Deposit

For the 2021 to 2023 drilling campaigns completed by GoviEx, all drill collar locations were initially spotted using a handheld GPS and final collar surveys were performed by professional surveyors (Benchmark Geospatial Engineering Consultants) using DGPS systems using the WGS84 UTM Zone 35S reference datum. Base stations listed in Table 10-7 were used as control points for the 2021 and 2022 final surveys. Check surveys of historical collar locations were also performed during the 2021 and 2022 final surveys on all deposits.

10.3. Njame and Gwabi deposits

10.3.1. Drilling

Drilling was carried out by a combination of DDH, RC and AC techniques. The AC method was only used at the early-stage exploration at Njame in 2006, and all subsequent drilling at the Njame and Gwabi deposits was completed by RC and DDH techniques. Figure 10-5 provides a drill hole location map for the Njame and Gwabi deposit areas.

The RC drilling technique was the primary method for obtaining suitable samples for MRE at these deposits and was carried out along drill lines spaced between 25 m and 50 m apart along prospective anomalies. All RC drilling at Njame and Gwabi was completed by Capital Drilling (Zambia) Limited using rig types typically similar to Schramm 450, medium-sized truck-mounted rigs with air capability of 1 100 cfm/350 psi. All RC drilling was completed with a 5” face hammer.

The majority of the DDH drilling was completed in 2008 and was carried out by Capital Drilling (Zambia) Limited. A truck-mounted LF-90 (Rig31) and a truck-mounted LF-90 (Rig26) rig were used. All DDHs were completed using PQ and NQ wireline tools.

Collar positions for all holes were initially established using handheld GPS. Drill sites and access were cleared using a bulldozer when required and the drill position was re-marked using handheld GPS. Upon hole completion, each drill hole was left with a polyvinyl chloride (“PVC”) collar tube cut at ground level. The collar coordinates were re-checked using handheld GPS. Subsequently, most drillhole collars were surveyed with a differential global positioning system (“DGPS”) by a professional surveyor (Chris Kirchhoff) and Lusaka-based Rankin Engineering.

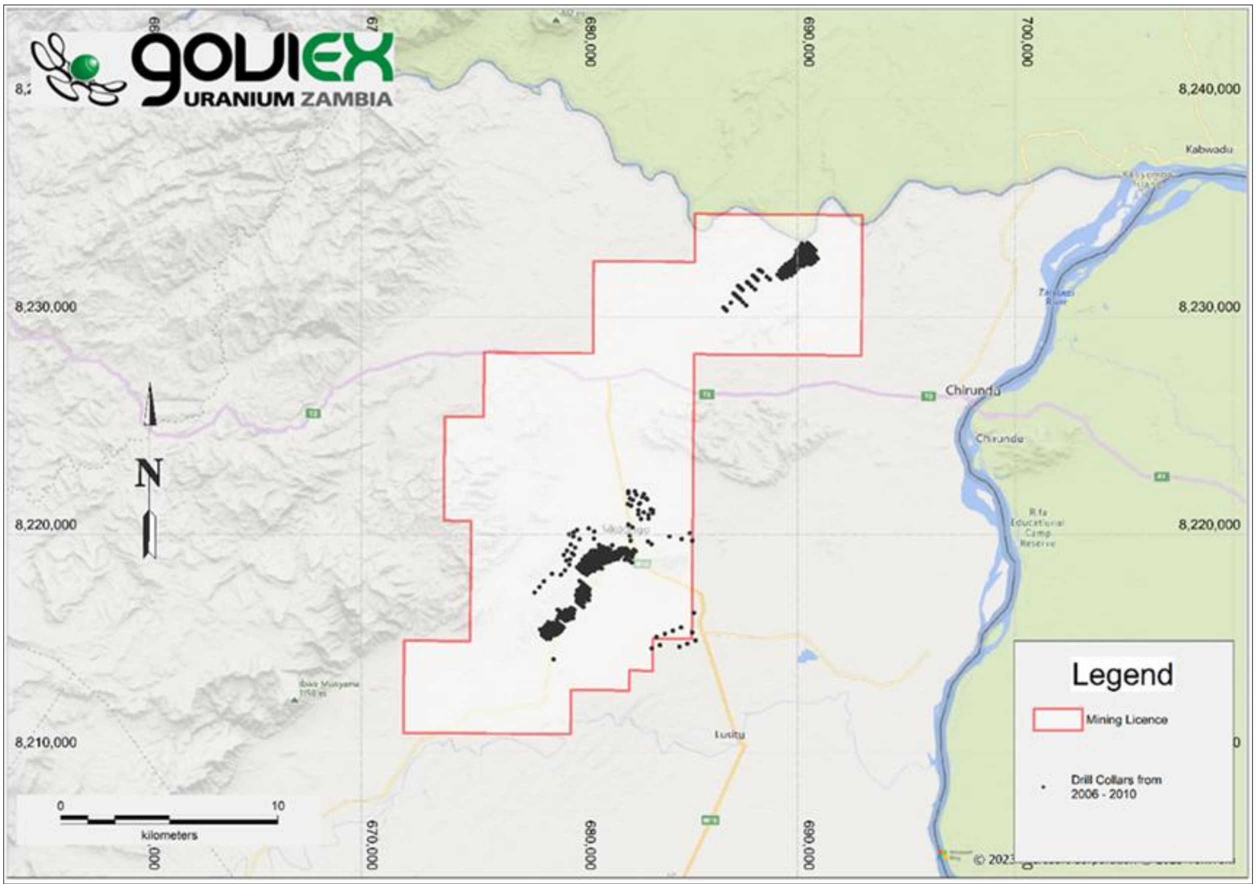

Figure 10-5: Drill hole location map for the Njame and Gwabi deposits

10.3.2.Logging and sampling

AFR used well-documented procedures for RC and DDH sample logging. In general, RC chips were logged immediately after drilling whereas the core was logged after being carefully joined up and marked on a V-trough. The information recorded included lithological, structural, geotechnical, weathering/ oxidation and mineralogical logs. For cored holes, the mineralised zones of each were selected at the discretion of the logging geologist.

The RC samples were collected as follows:

- RC drill chips were collected at 1 m intervals down-hole using a cyclone into PVC bags prior to splitting
- The collected samples were riffle split using multiple passes through a single-stage riffle splitter; a final sample of approximately 2 kg was collected for submission to the laboratory for analysis
- In wet holes, the samples were left to dry as best possible and then homogenised and quartered by hand.

RC chip trays were systematically logged by collecting the sieved RC chips and storing them in a tray, with each labelled compartment of the tray containing the chips from 1 m.

The DDH sampling methodology was as follows:

- Sampling was preceded by radiometric scanning of the core whilst on the V-frame. Scanning was carried out using either a RS-125 spectrometer or an Exploranium GR-110G handheld scintillometer. Care was taken to ensure minimum influence from any possible source of ionising radiation, thus scanning of the core on the V-trough was carried out at a minimum distance from any suspected ionising radiation source
- The maximum sample length was 1 m and the minimum sample length was 0.25 m
- The total width of the sampled zone extended 2 m above and below the mineralised zone as determined by the scintillometer readings
- The other guiding factor to sampling besides the scintillometer readings was the lithology. Sampling across lithologies was avoided where possible
- NQ core was sampled using half-core samples, while the PQ core was sampled using a core saw taking a 25 mm wide 'fillet' from the core width

- Trained and supervised technicians sampled the drill core. Each sample was taken from the left-hand half of each piece of core for that metre (leaving the half with the orientation line and/or metre marks in the tray) and placed into an appropriate sample bag
- Calico sample bags with drawstrings were used for core sampling. Sample tickets were used in the sampling process with one half (identical halves) of each ticket, which had a printed sequence of sample numbers (six figures), placed in the calico sampling bag
- The sample tickets were annotated with the drill hole number and the sample interval. As part of the quality control protocols, the technician verified that the metered interval marked on the core matched the metred interval written on the sample ticket and matched the metered interval on the sample form. The technician verified that the corresponding sample number on the sample form, for that interval, matched the sample number of the sample ticket, and matched the sample number written on the sample bag.

10.3.3.GoviEx drilling

GoviEx completed three drill holes on each of the Njame and Gwabi deposits in 2022 for data confirmation and geometallurgical sampling. The locations of these holes are provided in Figure 10-6 and Figure 10-7. Logging and sampling procedures used for these holes are consistent with the procedures used for drilling completed on the Muntanga, Dibbwi and Dibbwi East deposit drilling campaigns.

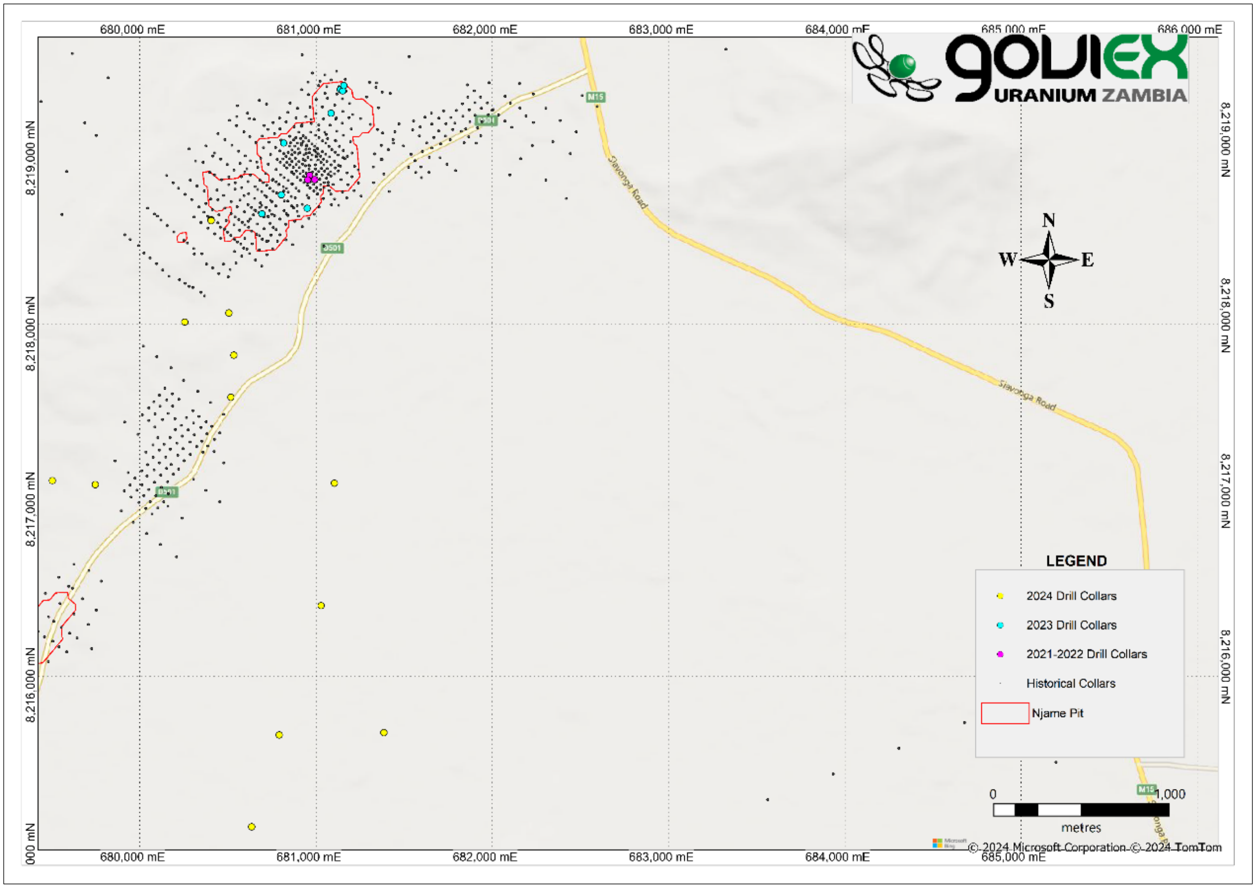

Figure 10-6: GoviEx drill hole location map for Njame

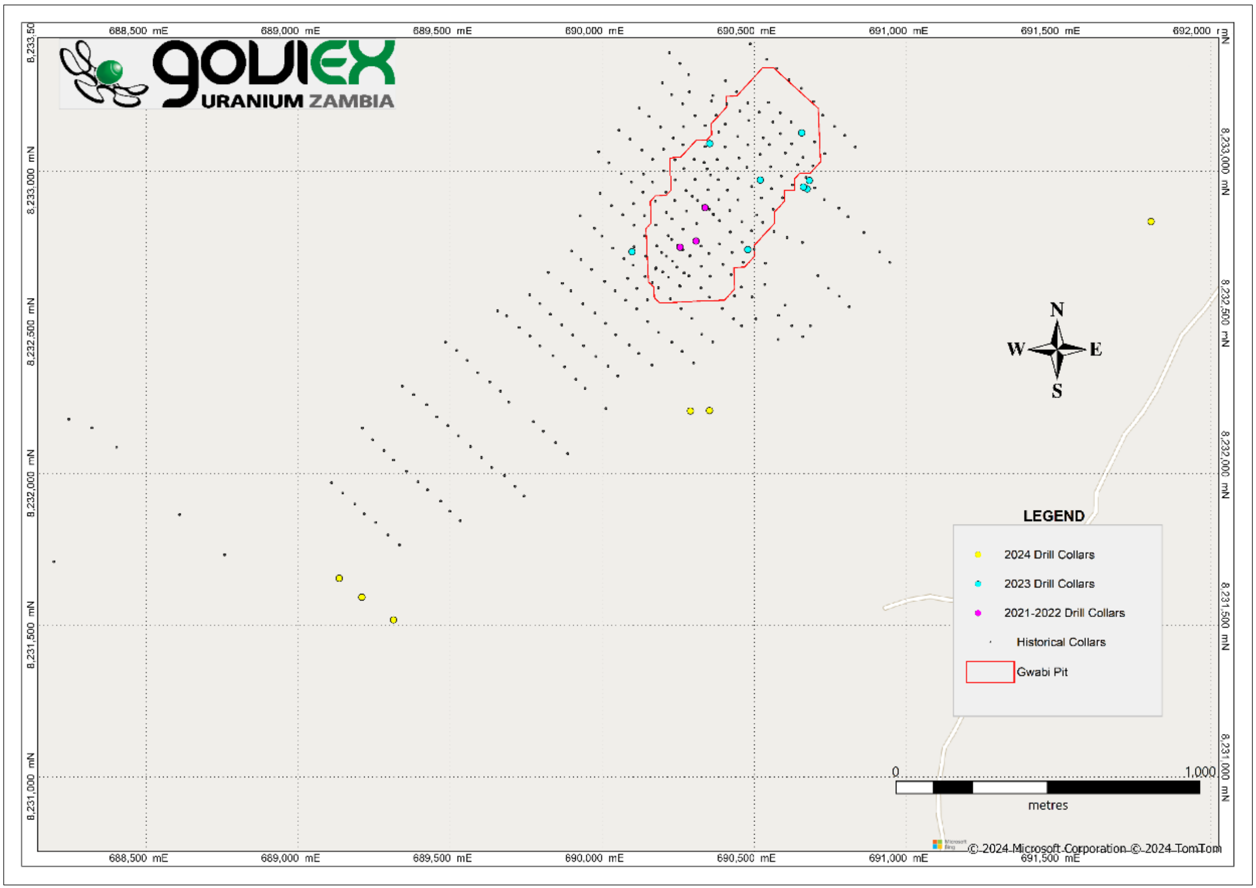

Figure 10-7: GoviEx drill hole location map for Gwabi

11. Sample preparation, analyses and security

11.1. Historical sample preparation, analysis and security for the Muntanga, Dibbwi and Dibbwi East deposits

Records and details for drilling conducted on the Muntanga, Dibbwi and Dibbwi East deposits prior to 2006 (circa 1980) are not available to allow sufficient verification of data collected during this timeframe. Therefore, all drilling prior to 2006 has been excluded from the MRE process. The description of sample preparation, analysis and security for programs completed between 2006 and 2012 is taken from the Project September 12, 2013, NI 43-101 technical report (CSA, 2013).

11.1.1. Sample preparation, dispatch and security

Drilling conducted by OmegaCorp (2006) and Denison (2007 to 2012) included both percussion and diamond drilling. Drill core and/or chips were photographed, logged, marked for sampling, split, bagged, and sealed for shipment at their field logging facility.

From 2006 to 2008, the samples were transported in a dedicated truck from Zambia to Johannesburg, South Africa where Genalysis Laboratory Services ("Genalysis") operates a dedicated sample preparation facility. Sample preparation was carried out via a process of drying, crushing and milling of RC and diamond core samples. Crushers were cleaned with a silica rock (waste rock) after every sample. Milling was done in a ring and puck pulveriser and contamination was avoided by cleaning with compressed air and silica rock (waste rock) after every sample. With every batch of 40 samples one waste rock blank was assayed, to monitor contamination. Following sample preparation, the assay pulps were forwarded by Genalysis to its Perth, Australia assay laboratory where the samples were held in secure, quarantined storage.

From 2009 to 2012, sample preparation was undertaken at ALS Chemex in Johannesburg. Received sample information was verified by ALS personnel and logged in the ALS tracking system; a sample receipt and sample list were generated and sent to the appropriate authorised Denison personnel. Sample preparation consisted of weighing and drying of each sample, followed by fine crushing of the entire sample to 70 % passing -2 mm. A 250 g split was collected from each sample and pulverised to 85 % passing 75 microns for analysis.

11.1.2. Laboratory analysis procedures

From 2006 to 2008, assay pulps were sent to Perth, Australia for analysis at Genalysis's laboratory by pressed powder XRF methods. Genalysis is an accredited National Association of Testing Authorities, Australia ("NATA") laboratory (Number 3244). Genalysis has been approved by Australian Quarantine and Inspection Service ("AQIS") for the receipt and treatment of samples from interstate and overseas. Genalysis is an Associate Member of the Association of Mining and Exploration Companies Inc. and a Member of the Standards Association of Australia.

Between 2009 and 2012, sample analysis was undertaken at ALS Minerals in Johannesburg, South Africa, using a combination of pressed powder XRF methods including ME-XRF05 and ME-XRF10.

Access to the assay laboratory premises was restricted by an electronic security system and sample results were stored using encryption and password protection.

11.1.3. Assay quality assurance/ quality control 2006 to 2008

From 2006 to 2008, a total of 91 samples underwent assaying at SGS for QAQC analysis. These were submitted as two sample batches for analysis in May 2008 from the 2007 to 2008 drilling campaign. They included field duplicates, field standards, field blanks and laboratory standards.

Table 11-1 summarises the numbers of samples submitted and their proportion as percentages and ratios of the total number of assays submitted.

Table 11-1: QAQC sample summary

QAQC sample/Assay type	Number of samples*	% of Total samples	Ratio
SGS Standard Samples	7	0.53	1:190
Omega Standard Samples	19	1.43	1:88
Omega Blank Samples	38	2.86	1:35
Omega Field Duplicate Samples	27	2.03	1:50

*QAQC conducted on holes drilled from 2007 to 2008. Total number of samples from 2007 to 2008 drill holes was 1 327.

11.1.3.1. Field duplicates

There is a reasonable correlation between primary samples and their duplicates submitted by Denison as shown in Figure 11-1. There is a general trend towards the underreporting of duplicates relative to their primary value as can be seen from where the points plot relative to the x=y line. However, 93 % of duplicate samples submitted were below 100 ppm U_3O_8 and therefore, moderate to higher grades are not well represented.

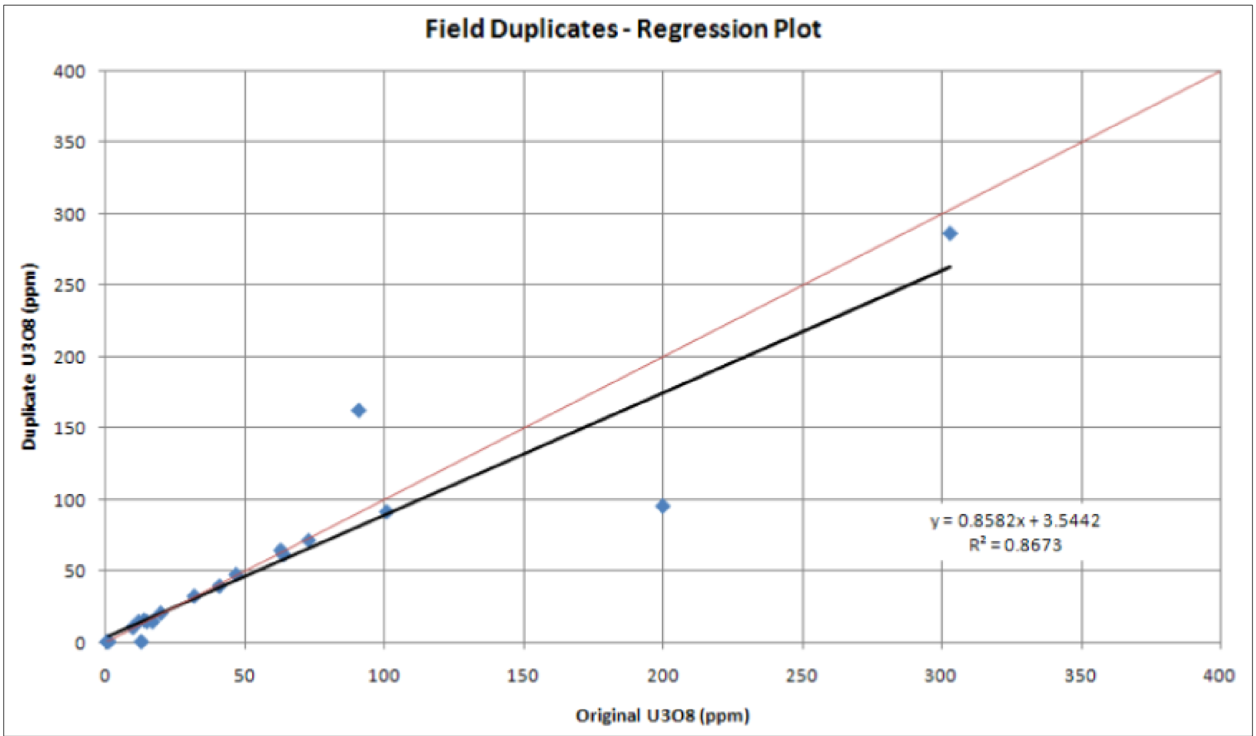

Figure 11-1: Field duplicate scatter plot

It should be noted that the duplicate dataset contains few samples and as such, conclusions from statistical comparison are somewhat limited. In general, there appear to be no significant issues with duplicate repeatability. However, it was highly recommended that in future drilling campaigns the assay QAQC database be significantly increased to a ratio of 1:20 rather than 1:50, and that QAQC samples are selected to be representative of the grade distribution at each mineral deposit and that sampled material is spatially representative.

11.1.3.2. Field standards

Four field standards (low grade, medium grade, high grade and very high grade) were submitted to SGS for analysis as part of sample batches submitted in May 2008 from the 2007 to 2008 drilling, to assess the level of confidence that could be applied to returned assay data from samples submitted. These were certified reference materials (CRM) of which expected values, and 95 % confidence limits (low, high) are listed in Table 11-2.

Table 11-2: List of field standards with expected values (U) and action limits

Name of standard	Number of samples	Expected value [ppm]	Upper action [ppm]	Lower action [ppm]	Data	
					Between action limits [%]	Beyond action limits [%]
UREM 3	5	439	455	423	40	60
UREM 4	4	100	115	85	100	0
UREM 5	5	775	792	756	0	100
UREM 6	5	1 887	1 925	1 867	0	100
Total	**19**				**32**	**68**

- UREM 3/SARM 23 is a moderate grade standard (expected value 439 ppm). The results of the analysis suggested a trend towards over-reporting of this standard. All five samples reported over the expected value, with three outside of the action limits.
- UREM 4/SARM 24 is a low-grade standard (expected value 100 ppm). Four samples were submitted and all performed well, returning values within the action limits close to the expected value. This is the grade range for which most duplicates were submitted.
- UREM 5/SARM 25 is a moderate to high-grade standard with an expected value of 775 ppm. Five samples were submitted, and all were above the 95 % upper action limit, returning values that were on average 10 % above the certified value.
- UREM 6/SARM 26 (expected value 1 887 ppm) performed poorly. Five samples of this standard were submitted four over-reported above the 95 % upper action limit and one under-reported significantly by over 10 %.

Control plots were plotted against Batch ID and over time. In cases where cyclical patterns of assay results against time can be seen in the control plots for standards, it can commonly be attributed to analytical drift, where assays report closer to their expected values when the analytical equipment is re-calibrated and drift further from their true values between calibrations. However, without direct consultation with the laboratory addressing the reasons for cyclicity, this cannot be confirmed.

Table 11-3: List of laboratory standards with expected values (U) and action limits

Name of Standard	Number of samples	Expected value [ppm]	Upper action [pp]	Lower action [ppm]	Data	
					Between action limits [%]	Beyond action limits [%]
UREM 3	2	439	455	423	0	100
UREM 4	2	100	115	85	50	50
UREM 5	2	775	792	756	0	100
UREM 6	1	1 887	1 925	1 867	0	100
Total	**7**				**10**	**90**

11.1.3.3. Quality assurance/ quality control conclusions and actions

Conclusions from the assay QAQC analysis of the 2007 to 2008 drilling campaign were:

- The limited number of blanks submitted by Denison all performed well with all samples reporting below detection. This suggests that field sampling methods and contamination-limiting procedures at SGS were adequate
- Results from the submission of external field standards were mixed. However, due to the limited number of samples submitted, future programs should aim to increase this number and to closely monitor the results
- Results from internal standards (UREM standards) were poor overall. Six out of seven standards reported within ±10 % of their certified values, but the average percentage error was 11 % outside the expected value
- Ongoing monitoring of internal laboratory control alongside external control was highly recommended as part of future drilling programmes and should be implemented as a matter of course. A set of pulp duplicates should be submitted to an umpire laboratory which can then be analysed alongside SGS samples, also testing laboratory precision
- The number of QAQC samples submitted overall was low and it was advised that in future drilling campaigns, this number should be increased to be more representative. It was also advised that, as a matter of course, QAQC data should be analysed concurrently with drilling. By doing this, if issues arise, it allows for the laboratory to be consulted, samples re-assayed and procedures reviewed if necessary, resulting in problems being resolved at the time and thus prevented for the rest of the campaign.

11.1.4. Assay quality assurance/ quality control 2009 to 2012

QC samples (reference materials, blanks and duplicates) were included with each analytical run, based on the rack size associated with the method. The rack size is the number of samples including QC samples within a batch. A blank was inserted at the beginning, standards were inserted at random intervals, and duplicates were analysed at the end of the batch.

Denison used standards provided by ALS Chemex for uranium assays. ALS Chemex standards were added to the sample groups by ALS Chemex personnel, using the standards appropriate for each group. In addition, for each assay group, an aliquot of Denison blank material was also included in the sample run. In a run of twenty samples, at least one ALS Chemex standard and one Denison blank were included. A list of standards used is provided in Table 11-4.

Table 11-4: ALS Chemex uranium standards

Standard ID	Element	Method	Expected value [ppm]
AMIS0029	U	XRF	890
AMIS0054	U	XRF	1 472
AMIS0096	U	XRF	137
AMIS0097	U	XRF	543
AMIS0098	U	XRF	848
AMIS0114	U	XRF	550
SARM-98	U	XRF	205
UREM3	U	XRF	439
UREM4	U	XRF	100

At the time of the drilling campaigns, CSA conducted checks on QAQC data and plotted returned standard assays against the certified values, as well as plotting duplicates against original samples for comparison. The precision for analyses was deemed acceptable, and for the most part, the accuracy of the analyses for the six reference standards and blank used was within industry acceptability as shown in Figure 11-2. For standard AMIS0098 as shown in Figure 11-3, the low point during November 2011 was due to a “blank” value being mislabelled as a “field standard”.

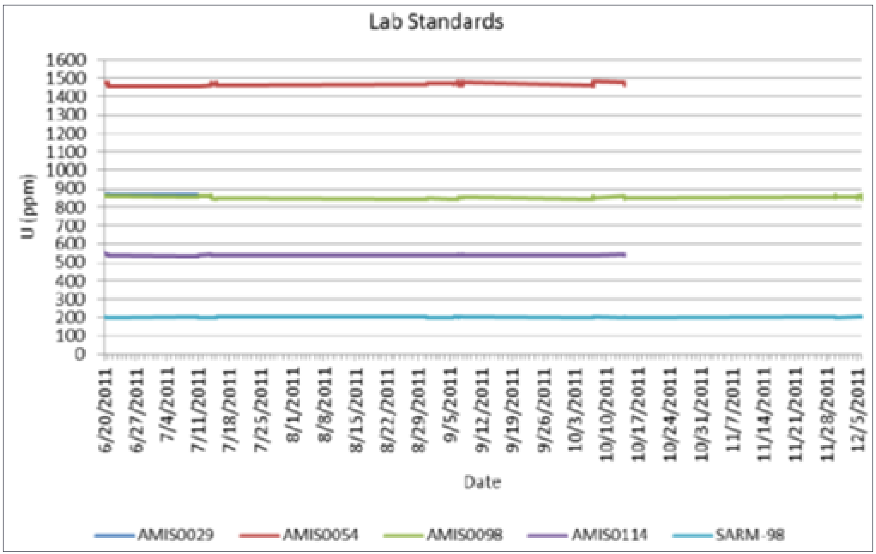

Figure 11-2: Control chart for ALS Chemex uranium standards

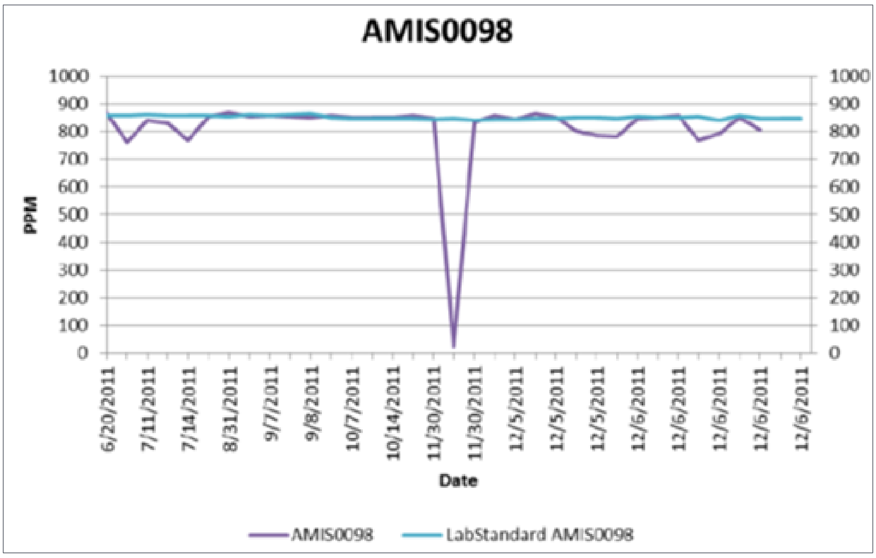

Figure 11-3: Control chart for ALS Chemex standard AMIS0098

11.1.5.Geophysical probe calibration, down-hole logging and quality assurance/ quality control

Prior to 2021, probe calibration was undertaken initially in the USA using the Grand Junction DOE pits prior to delivery to the site. Further periodic checks were undertaken using drill hole MTC51600-04 as a standard. If problems were detected in the probes during test hole logging, the equipment was sent back to the USA for repair and calibration.

Down-hole logging performed by Denison was conducted by trained and dedicated personnel devoted solely to this task. The tools, and a complete set of spares, were manufactured by Mount Sopris Instrument Company in Golden, Colorado and were shipped to Zambia in 2007. Drill hole logging data were stored on digital media in the logging truck at the exploration sites. The raw and converted logging data were periodically copied electronically to Denison's Lusaka, Toronto, Saskatoon and Denver offices, where all data were checked and reviewed.

Denison retained the services of a senior geophysical consultant to oversee training, implementation, and quality control protocols with the Zambian logging personnel. Denison's policy at the Project was for trained technicians to probe every drill hole immediately upon completion of drilling. Initially, all holes were probed 'open hole', but local bad ground conditions and water inflows necessitated probing to be completed inside the drill string and, depending upon ground conditions, also in the open hole. Representative chips or cores from the anomalous sections of holes that collapsed prior to down-hole probing were sent for XRF analyses.

At the end of the 2011 drilling campaign, 14 holes were chosen to re-probe at the end of the season due to concerns about radon contamination and the repeatability of probe results. Drill holes DMC1002, DMC1009, DMC1034, DMC1036, DMD1003, DMD1006, DMD1016, DMD1017, DMD1020, DMD1027, DMD1030, DMD1033, DMD1061, and DMD1077, were selected for re-probing and analysis. In some holes, it was not possible to re-probe the entire hole length because a portion of the hole had collapsed. Figure 11-4 provides a comparison of the original and repeat probe results from the selected 2011 holes, demonstrating acceptable repeatability of the probing results.

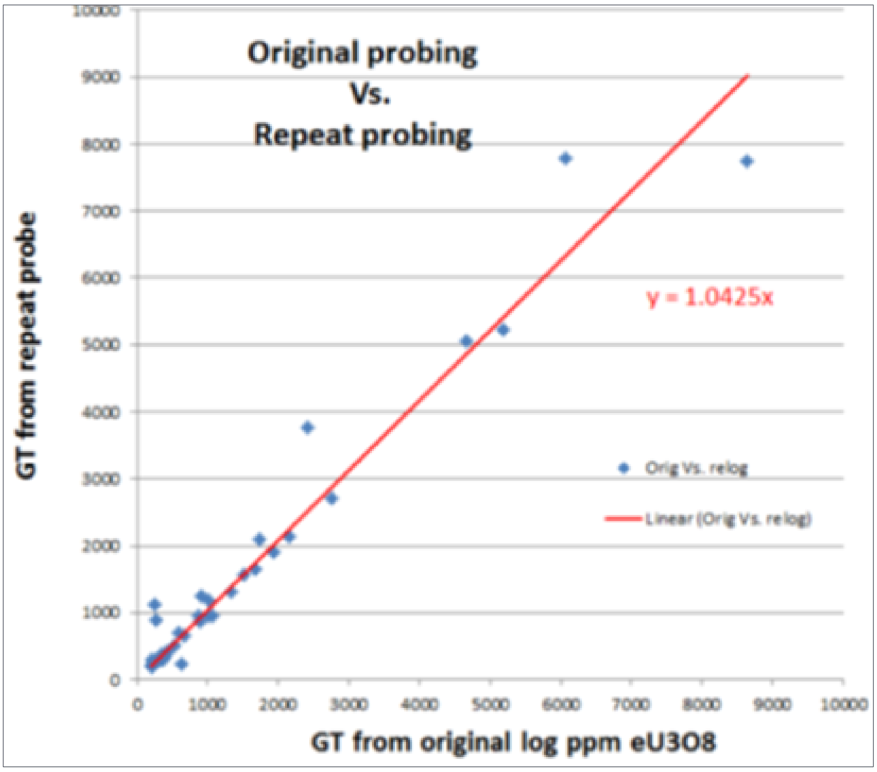

Figure 11-4: Repeat logging exercise

11.2. Historical sample preparation, analysis and security for the Njame and Gwabi deposits

11.2.1. Sample preparation, dispatch and security

Sample preparation on site was restricted to core logging and splitting. Once individual samples were placed in the calico bags, along with the sample ticket, the bags were closed and taped firmly. QC samples, including blanks and certified reference materials ("CRM"), were inserted at a rate of one blank and CRM per 50 samples.

Pool sand, obtained from an area north of Lusaka (Katuba), was put into sample bags and used as "blank" samples.

Three certified standards were regularly inserted into the sample sequence as part of the QC protocols. These samples were inserted on a rotating basis (Standard AMIS0004 or AMIS0045, alternating with Standard AMIS0029).

AFR drilling procedures required samples to be taped closed once taken from the RC sampling site or diamond core sampling facility. Samples were then transported directly to Lusaka, Zambia for air freight to ALS Chemex Johannesburg.

Reference material was retained and stored on-site, including quarter-core, fillet-core or RC chips and photographs generated by diamond and percussion drilling, and duplicate pulps and residues of all submitted samples. All pulps were stored at ALS Chemex Johannesburg storage facility for three months, after which they were returned to AFR in Lusaka.

11.2.2. Laboratory analysis procedures

ALS Chemex Ltd was used as the principal analytical laboratory company for U_3O_8 analysis. The sample preparation was completed at ALS Chemex Johannesburg, with analytical analysis (i.e. assaying) of the sample pulps completed at either the ALS Chemex analytical laboratories in Johannesburg or Vancouver, Canada. The ALS Chemex laboratories in Johannesburg and Vancouver are both ISO 9001:2000 accredited.

The analytical method used by ALS Chemex is ME-XRF 05. The method description for this is as follows:

"A pressed pellet is prepared and analysed by wavelength dispersive XRF for the selected elements. Uranium (DL–2.5 ppm), converted to U_3O_8 (by ALS Chemex) using conventional conversion factors."

11.2.3. Specific gravity determinations

Specific gravity ("SG") determinations were carried out by AFR. The method applied to density collection included sun drying, weighing the core in air, followed by plastic wrapping and weighing in water. The bulk density was then determined as a ratio of weight in air overweight in water. The weighing was completed using high-quality electronic scales which underwent regular calibration.

Samples were taken from the dominant rock types at both Njame and Gwabi. The average measured density per logged rock type for all samples weighing more than 1.0kg is presented in Table 11-5 and Table 11-6 for the Gwabi and Njame deposits, respectively.

Table 11-5: Specific gravity measurements for Gwabi by Logged Rock Type (samples greater than 1.0kg)

Rock type	Number of samples	Specific gravity		
		Minimum	Maximum	Mean
GRIT	20	1.94	2.42	2.06
GSSTN	44	1.86	2.36	2.02
PGRIT	39	1.85	2.62	2.12
PSSTN	33	1.40	2.46	2.13
SLTSTN	2	1.96	2.14	2.05
SSTN	53	1.71	2.44	2.03

Table 11-6: Specific gravity measurements for Njame by Logged Rock Type (samples greater than 1.0kg)

Rock type	Number of samples	Specific gravity		
		Minimum	Maximum	Mean
CNGLM	1	2.26	2.26	2.26
GRIT	29	1.82	2.16	1.97
GSSTN	63	1.77	2.16	1.98
PGRIT	52	1.89	2.26	2.06
PSSTN	24	1.88	2.30	2.13
SLTSTN	66	1.84	2.31	2.06
SSTN	263	1.72	2.68	1.98

11.3. GoviEx sample preparation, analysis and security

11.3.1. Sample preparation, dispatch and security

Since 2021, only diamond drill core has been sampled for assay by GoviEx. The core is marked for Geotech and photographed before being transferred to the core farm where it is logged, marked for sampling, split, bagged and sealed for transport to the Ndola, Zambia prep facility of ALS Global. Here the samples are crushed to >70 % passing through a 2 mm screen, and a 250 g subsample is collected and pulverised to >85 % passing through a 75 micron screen (Tyler 200 mesh). The pulverised sample is then bagged and dispatched to ALS Global's Johannesburg analytical laboratory.

11.3.2. Laboratory analysis procedures

Since 2021, sample analysis undertaken by ALS Global (ALS) has used their ME-MS61 technique which involves a four-acid digest followed by ICP-MS and ICP-AES. Results are sent via email to be authorised by GoviEx personnel for incorporation into the master sample database.

11.3.3. Assay quality assurance/ quality control

QC samples (reference materials, blanks and duplicates) were included with each analytical run. A total of 5 882 samples including quality control samples underwent assaying at ALS for the 2021 to 2023 drilling campaigns. These included field duplicates, field standards, field blanks and laboratory standards that were submitted at a rate of one duplicate, one standard and one blank within sample batches of 20 samples. Table 11-7 provides details of the CRM used during the 2021 to 2023 drilling campaigns. For the 2024 QAQC progress, these included field duplicates (22), CRMs (22), and blanks (22).

Table 11-7: CRM-certified concentrations and limits (2021-2023)

CRM ID	Element	Method	Expected Value [ppm]	2 x Standard Deviation [ppm]
AMIS0106	U	M/ICP	114	8
AMIS0514	U	4A_MICP	329	24
AMIS0186	U	M/ICP	2 686	257

11.3.3.1. Blanks

A typical QAQC programme includes the submission of blank sample material to confirm no sample contamination is occurring. A total of 293 blank samples were analysed for uranium for the 2021 to 2023 drilling campaigns. The blank performance plot is provided in Figure 11-5. The results for the blank samples show that there is scatter in the blank sample data set, with periodic elevated values, and a slight progressive increase over time. Further investigation is warranted to determine the cause of the occasional data spikes and gradual increase in values over time of the blank sample results.

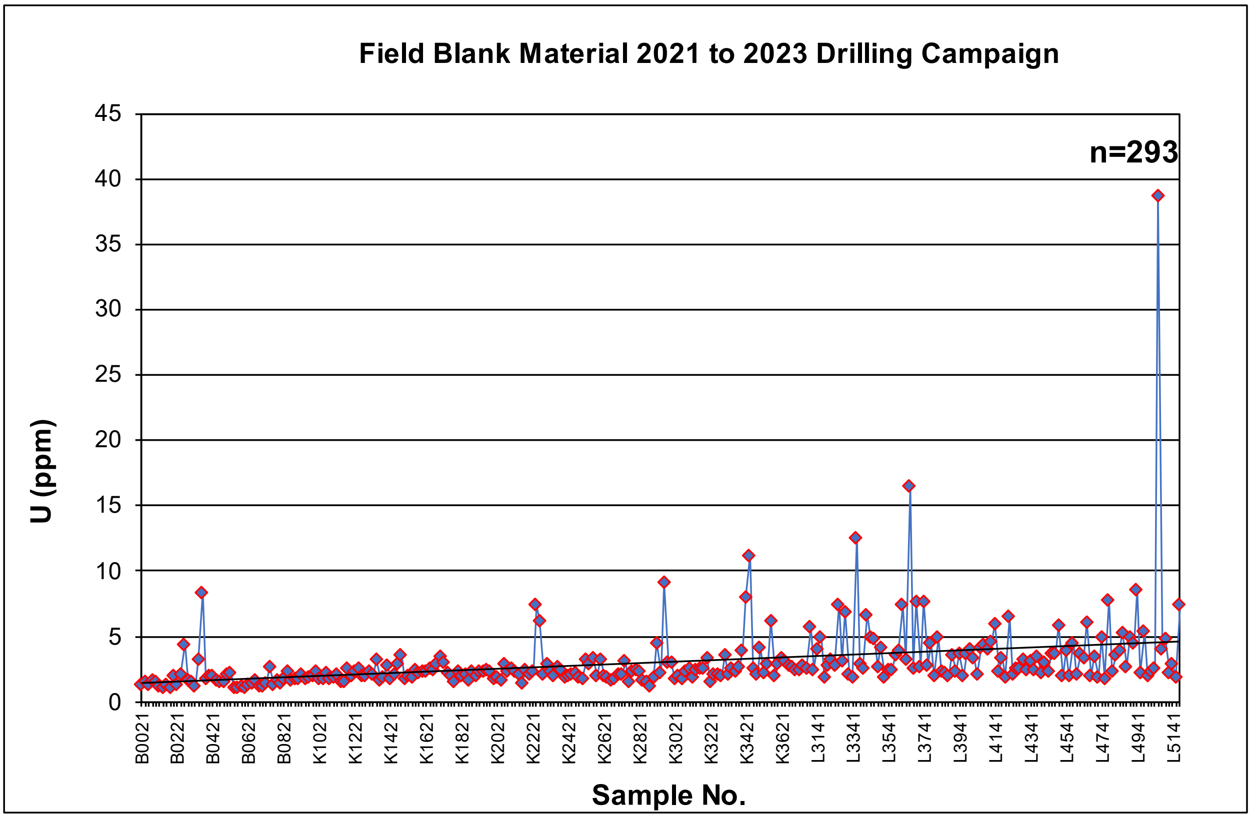

Figure 11-5: Blank sample performance chart

11.3.3.2. Certified Reference Materials

A total of 184 CRM samples were submitted during the 2021 to 2023 drilling campaigns, at a rate of one in every 20th assay sample. A total of 92 samples of each CRM AMIS0514/257 and AMIS0106/633 were submitted for analysis and the results are provided in Figure 11-6. The performance plots for both the CRMs demonstrate that the analytical results fall within an acceptable range of typically ±2 standard deviations of the expected value. However, the performance of CRMs AMIS0514/257 and AMIS0106 consistently falls below their expected U value of 329 ppm and 2 686 ppm.

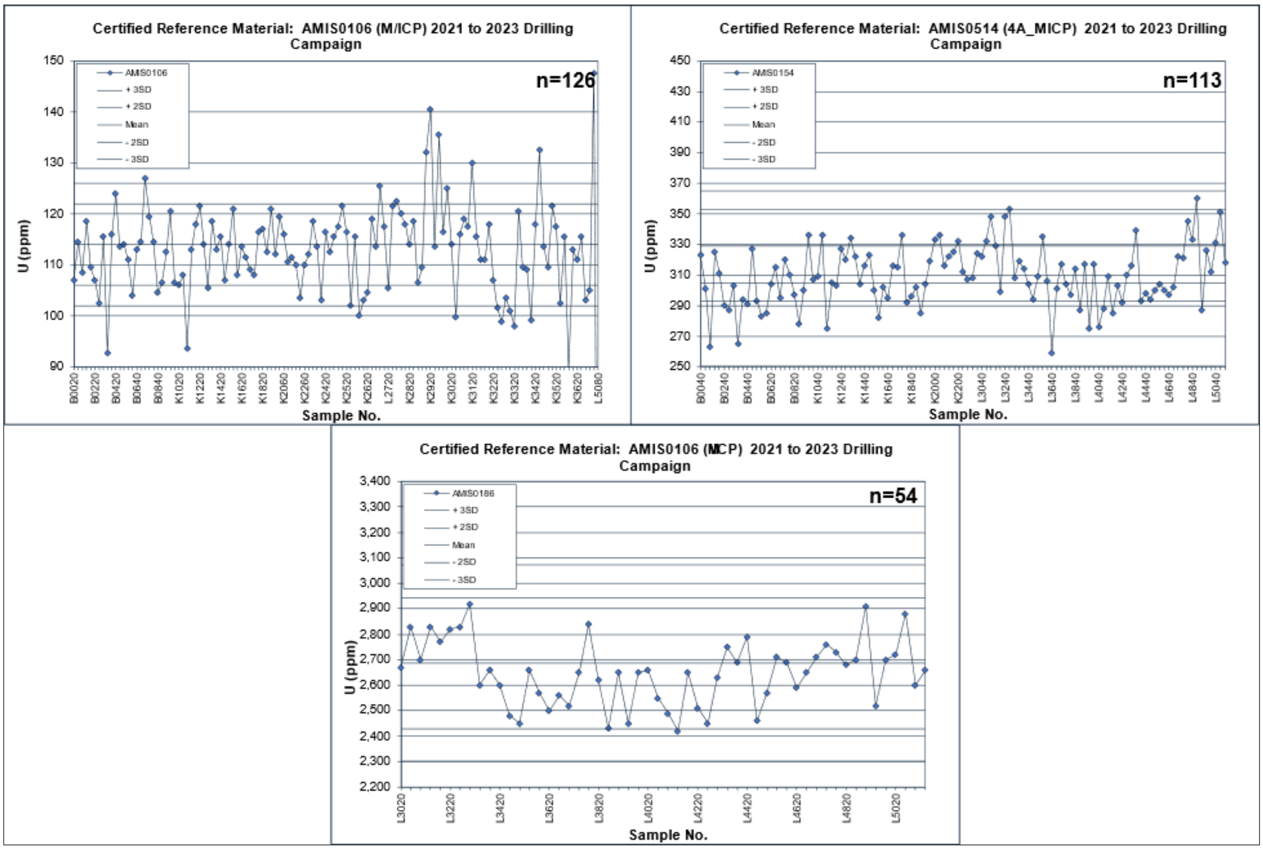

Figure 11-6: CRM sample performance chart

11.3.3.3. Duplicates

A total of 293 field duplicate samples were collected during the 2021 to 2023 drilling campaigns. Two duplicate samples did not return any results. Field duplicates were collected by sampling the remaining half of the core interval selected for the original assay sample. A comparison of assay results between the field duplicates and original assay samples is provided in Figure 11-7 to Figure 11-9.

The results of the duplicate analysis demonstrate an acceptable correlation between the original and field duplicate sample pairs, however, an observed marginal bias towards underreporting of grade can be seen in field duplicate samples for higher-grade samples >300 ppm U.

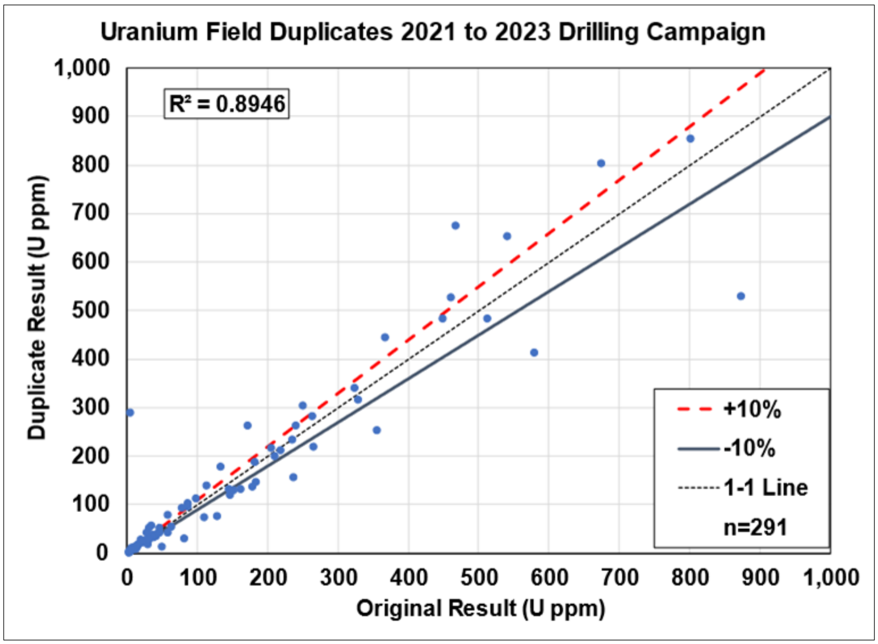

Figure 11-7: Scatter plot of original and duplicate assay samples

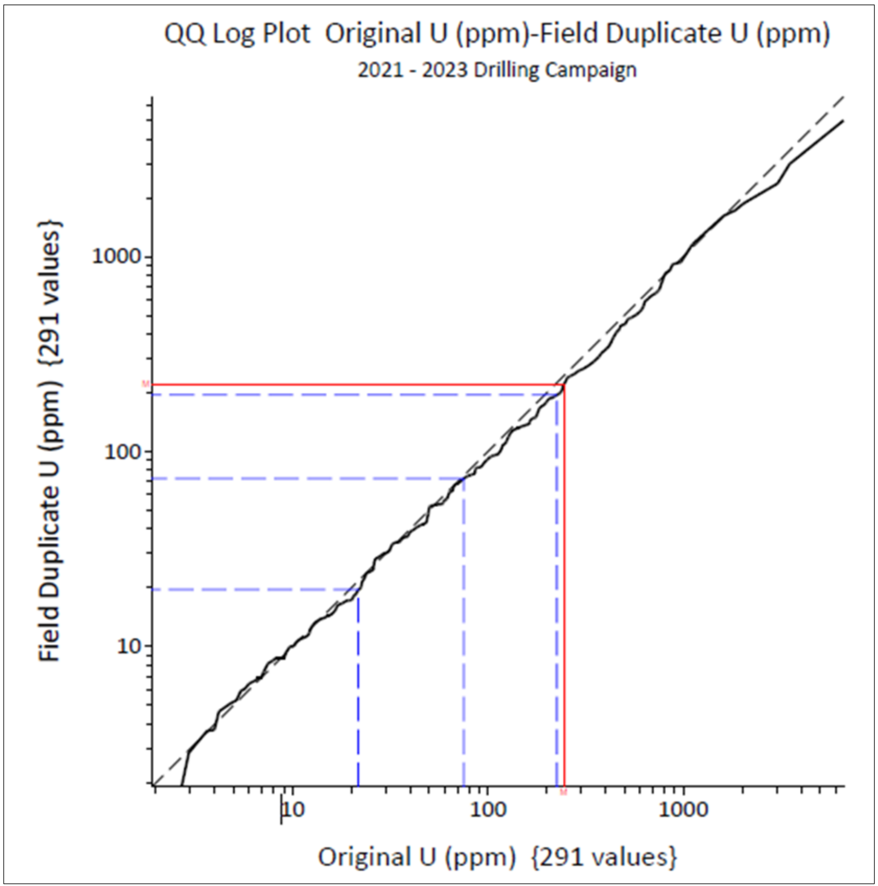

Figure 11-8: Q-Q plot of assay duplicate pairs

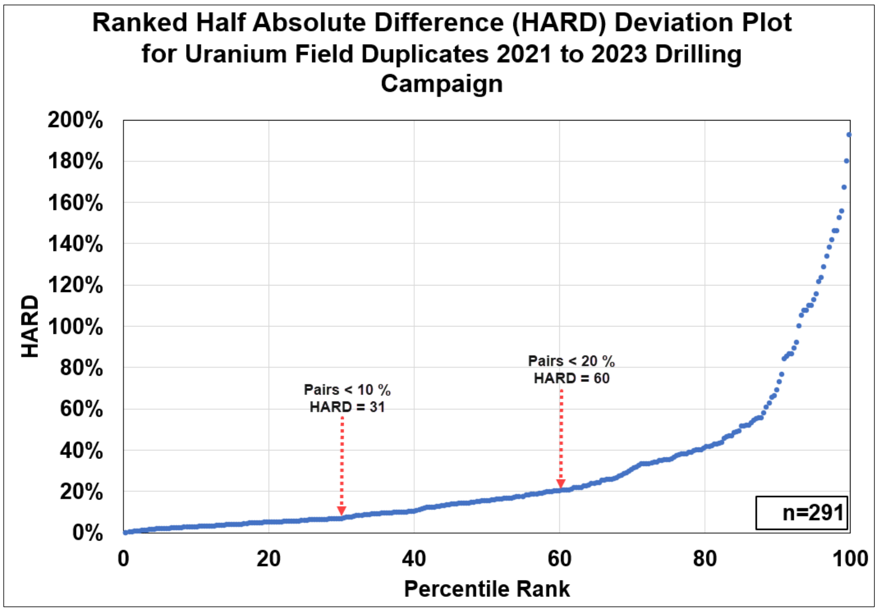

Figure 11-9: Ranked half absolute relative deviation plot

11.3.4.Specific gravity determinations

A total of 450 valid bulk density measurements have been collected from DD cores across the Muntanga, Dibbwi and Dibbwi East deposits. After the core was dried the density was determined by calculating the core volume which was then divided into the weighed dry mass to calculate the in-situ dry bulk density. A wax coating was used in 88 % of the volume displacement density determinations, taking the rock's porosity into account to prevent overstating the density.

The mean and median density values are 2.1 t/m^3 with very low variance and coefficient of variation ("CV") values as summarised in Figure 11-10. There was no recognisable correlation between density and depth or lithology. A global dry bulk density of 2.1 t/m^3 was used for the estimation of the Muntanga, Dibbwi and Dibbwi East Mineral Resources.

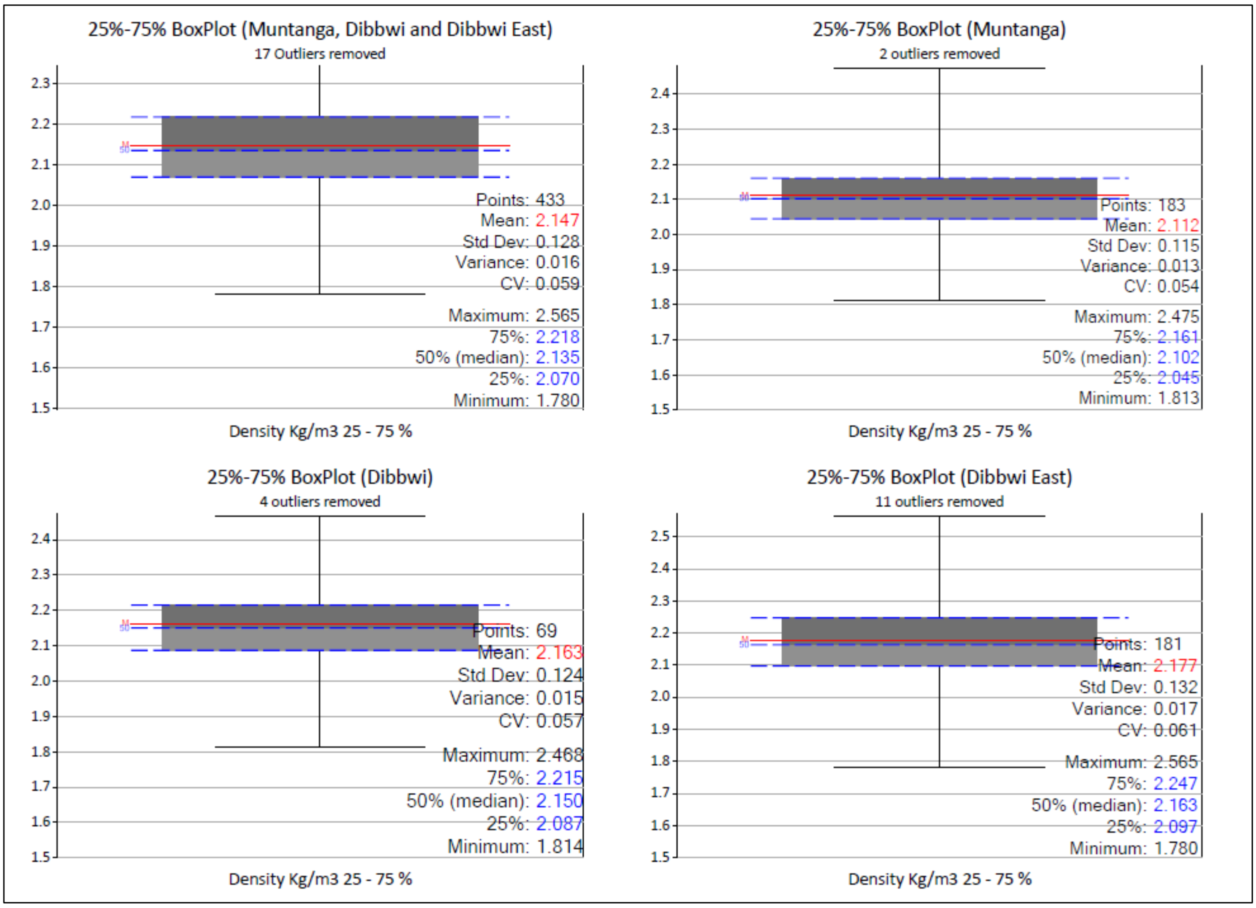

Figure 11-10: Specific gravity sample summary statistics

11.3.5. Geophysical probe calibration, down-hole logging and quality assurance and quality control

During the 2021, 2022, 2023 and 2024 drilling campaigns, an external service provider provided all down-hole geophysical logging services. Terratec Geophysical Services Namibia was contracted to provide all down-hole logging equipment and personnel, conduct probe calibration and initial QAQC of down-hole geophysical data.

Calibration of all down-hole probes was carried out at the Pelindaba test facility in South Africa prior to arriving on site.

In-field QC measures consisted of weekly probe checks using drill hole MTC51600-04 to ensure consistent and reliable operation of the probe used for down-hole gamma logging. Figure 11-11 provides an example of repeat logging results showing consistent readings between logging runs. Only one gamma probe was used during the 2021 to 2024 drilling campaigns.

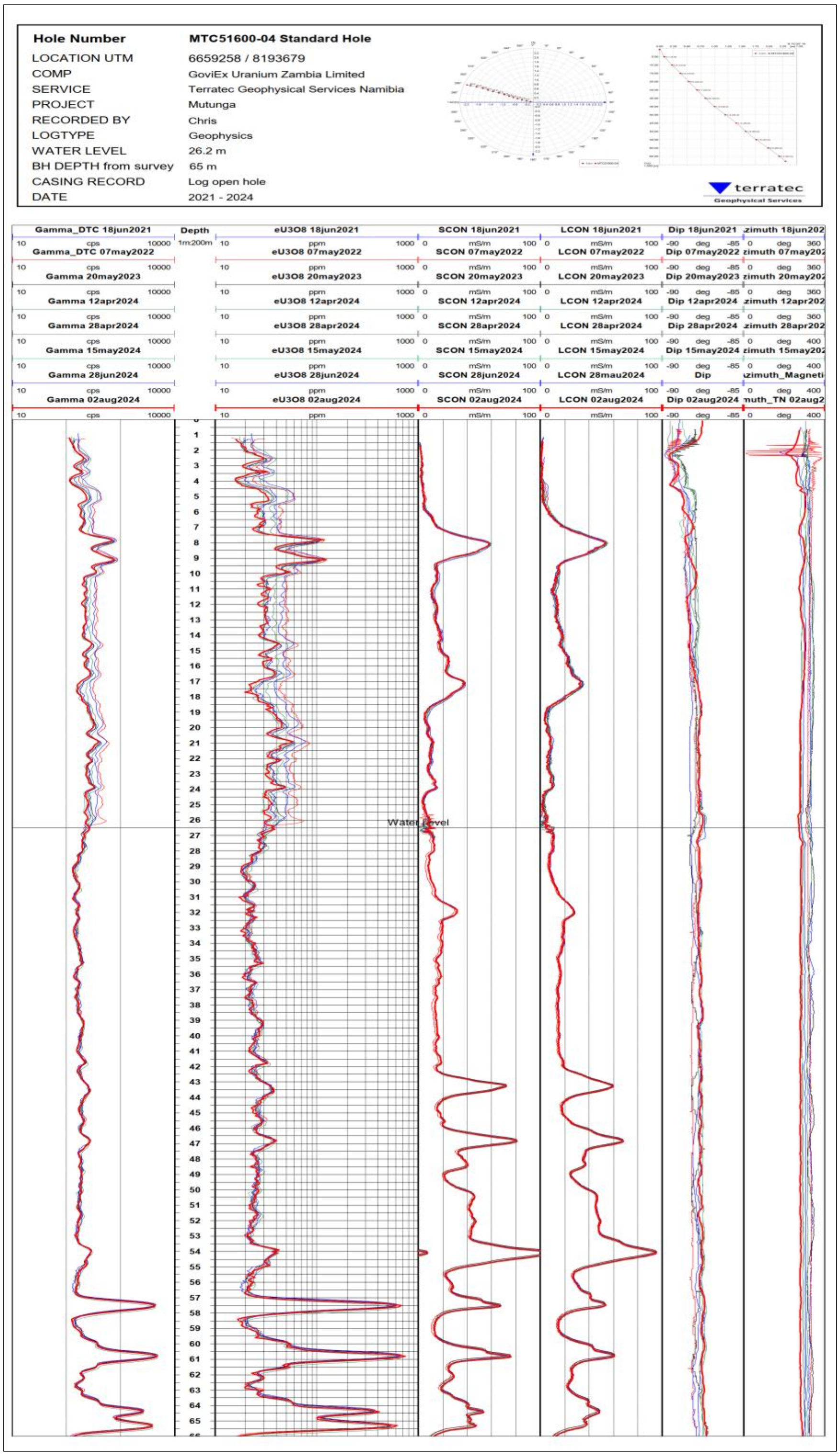

Figure 11-11: Example of multiple runs to determine repeatability of logging at test hole MTCS51600-04

11.4. Qualifying persons' comments on Section 11

In Mr. Deiss's opinion the sample preparation, security, and analytical procedures meet industry standards, and the QAQC programmes, as designed and implemented by GoviEx and past operators, are adequate; consequently, the assay and down-hole probe data within the drill hole database are suitable for MRE purposes. The 2024 drilling was primarily outside of the Muntanga, Dibbwi and Dibbwi East mineralised zones, and drilled for sterilisation, hydrological, and geotechnical purposes and as such not used in the MRE.

12. Data verification

12.1. Data verification by previous companies

12.1.1. Denison down-hole radiometric quality assurance/ quality control

Limited down-hole radiometric QAQC data are available to support the historical drilling completed prior to 2006, however Denison's drilling campaigns, which represent the majority of historical data for the Muntanga, Dibbwi and Dibbwi East deposits, used a variety of systematic checks and standards for routine checking and calibration of down-hole radiometric logging tools.

Probe calibration was undertaken initially in the USA, using the Grand Junction DOE pits prior to delivery to site. Further periodic checks were undertaken using drill hole MTC51600-04 as a standard. If problems were detected in the probes in the test hole located at Muntanga, the equipment was sent back to the USA for repair and calibration.

An exercise of repeat down-hole probing was completed by Denison on 14 selected drill holes to review the repeatability of the results from the down-hole radiometric probe. Although the exercise was based on a relatively small eU_3O_8 database, results of the study suggested that the down-hole probe was performing within acceptable limits, as illustrated in
Figure 12-1.

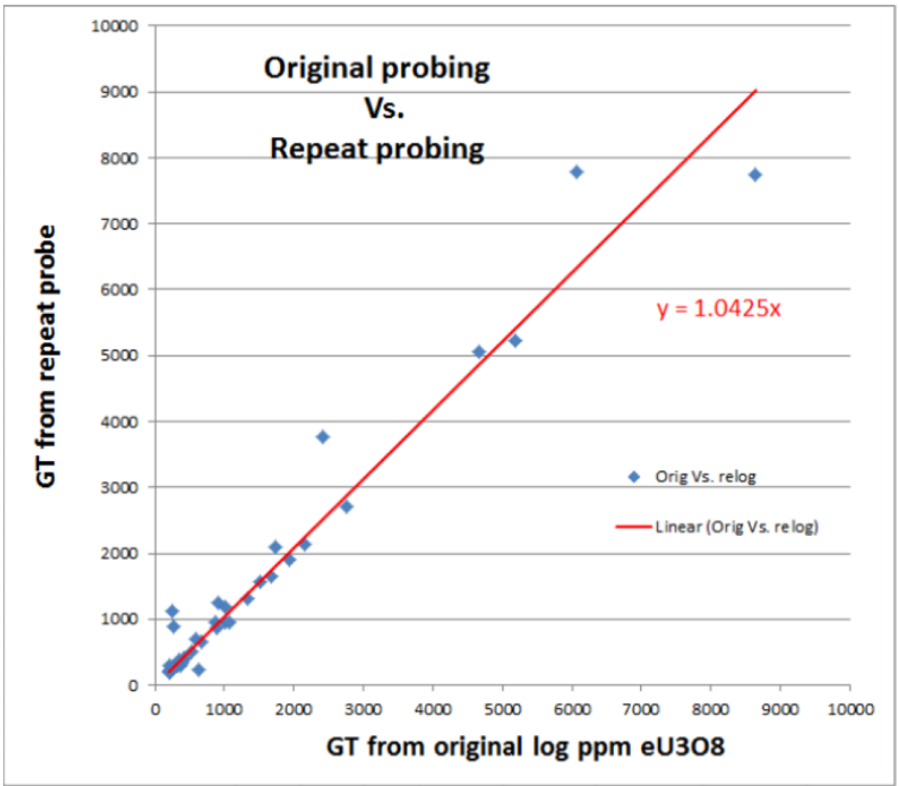

Figure 12-1: Repeat radiometric logging of selected drill holes by Denison

12.1.2. Data verification by CSA Global

CSA Global ("CSA") conducted data verification exercises in 2009 and 2012 to support the historical MRE updates completed by CSA. The following items were included in their data verification process, including exploration protocols used by Denison:

- Core sampling, sample preparation and assaying
- QAQC control procedures
- Drill hole collar and down-hole deviation surveys
- Down-hole radiometric logging procedures and results and
- Database validation.

No material issues were identified by CSA regarding data collected by Denison. For drill holes completed prior to Denison (circa 1980) on the Muntanga and Dibbwi deposits with collar prefixes 'DDH' and 'DWD', a number of data concerns were identified which could not be resolved due to insufficient information available. Therefore, these drill holes were excluded from use within the MRE process.

12.1.3. Data verification by AFR

AFR completed twin hole drilling of RC and DDH to confirm AC holes, as well as DDH to confirm RC holes. A total of 23 twins were completed and compared versus the original holes during the exploration programmes at Njame and Gwabi. Although some of the holes were not directly comparable due to extra sampling requirements, the results indicate that the comparison between twin holes is generally acceptable.

12.2. Data verification by SRK

12.2.1. Site visit

Mr. Deiss did not complete a site visit. Mr. Deiss worked directly with Mr. Revering on the project until early 2024 until Mr. Revering was no longer in SRK's employ. Subsequently, Mr. Deiss took over the role of QP for to ensure the continuation and consistency in the technical work. Mr. Revering who visited the Muntanga project twice in 2022, from May 8 to May 11, and October 17 to October 20. During the site visits, he observed drilling and down-hole logging activities, core and drill chip logging and data collection, and assay sampling and chain of custody protocols. He can confirm that the description of the geology, mineralisation and mineralisation controls, and the drilling, logging, sampling and data collection techniques described are consistent with observations made in the field during these site visits.

The QP, Mr. Deiss, is satisfied that the site visits completed by Mr. Revering, while they were both directly engaged on the project, have verified the information upon which the Mineral Resource is based.

12.2.2. Drill hole collar coordinate verification

As part of the 2021 and 2022 drilling campaigns, check surveys were conducted on a limited number of historical drill hole collars to verify the location and relative position of the historical collars to drill holes completed by GoviEx. Through this verification exercise, it was determined that the UTM WGS84 drill hole collar coordinates for the historical drill holes were on average approximately 7.25 m off in the easting coordinate and 0.15 m off in the northing coordinate. Therefore, all historical collar coordinates for drill holes located on the Muntanga, Dibbwi and Dibbwi East deposits were shifted to align with the 2021 to 2023 survey locations.

In addition, all drill hole collar elevations were adjusted to align with the 2023 LIDAR survey conducted on the Muntanga Project area in Q1 2023. All drill hole collar adjustments were completed in preparation for mineral resource estimation purposes.

12.2.3. Drill hole assay database review

SRK conducted a review of the Project drill hole assay database, comparing database entries to the original Lab assay certificates. Approximately 10 % of historical assay database entries and 85 % of recent assay database entries were validated against the original Lab assay certificates, and no errors were noted.

No data validation was conducted on historical drill holes completed prior to 2006, as insufficient documentation and details were available for review. Therefore, SRK excluded all historical data collected prior to 2006 from the MRE process.

12.2.4. Radon contamination

Radon is a naturally occurring radioactive gas that is generated during the normal radioactive decay of uranium into stable lead. Radon is produced by the radioactive decay of radium-226, which is a daughter product within the uranium decay chain found in uranium deposits. Because of its gaseous form, it can easily migrate through fractured rock masses and concentrate in catchment areas such as caves, underground mines, reservoirs and open drill holes. During the 2021 and 2023 drilling campaigns on the Dibbwi East deposit, radon contamination was identified within some drill holes, causing inflated down-hole radiometric signatures and overestimated eU_3O_8 grades within those holes. Examples of identified radon contamination in 2021 to 2023 drill holes are provided in Figure 12-2. The down-hole location and extent of the radon contamination was found to be associated with the presence of fracturing within the drill hole and depth of the water table. Where fractures were encountered above the water table, radon contamination was generally limited to above the water, and vice versa.

SRK reviewed the down-hole radiometric and eU_3O_8 profiles for all 2021 and 2023 drill holes, and where radon contamination was identified, adjusted (corrected) the eU_3O_8 profiles to produce a more robust eU_3O_8 grade profile.

SRK also reviewed the down-hole radiometric and eU_3O_8 profiles for all historical drill holes (circa 2006 to 2012), and where radon contamination was identified, adjusted (corrected) the eU_3O_8 profiles to produce a more robust eU_3O_8 grade profile as illustrated in Figure 12-2.

A total of 167 drill holes were identified as having variable degrees of suspected radon contamination and were adjusted accordingly to produce more robust eU_3O_8 grade profiles.

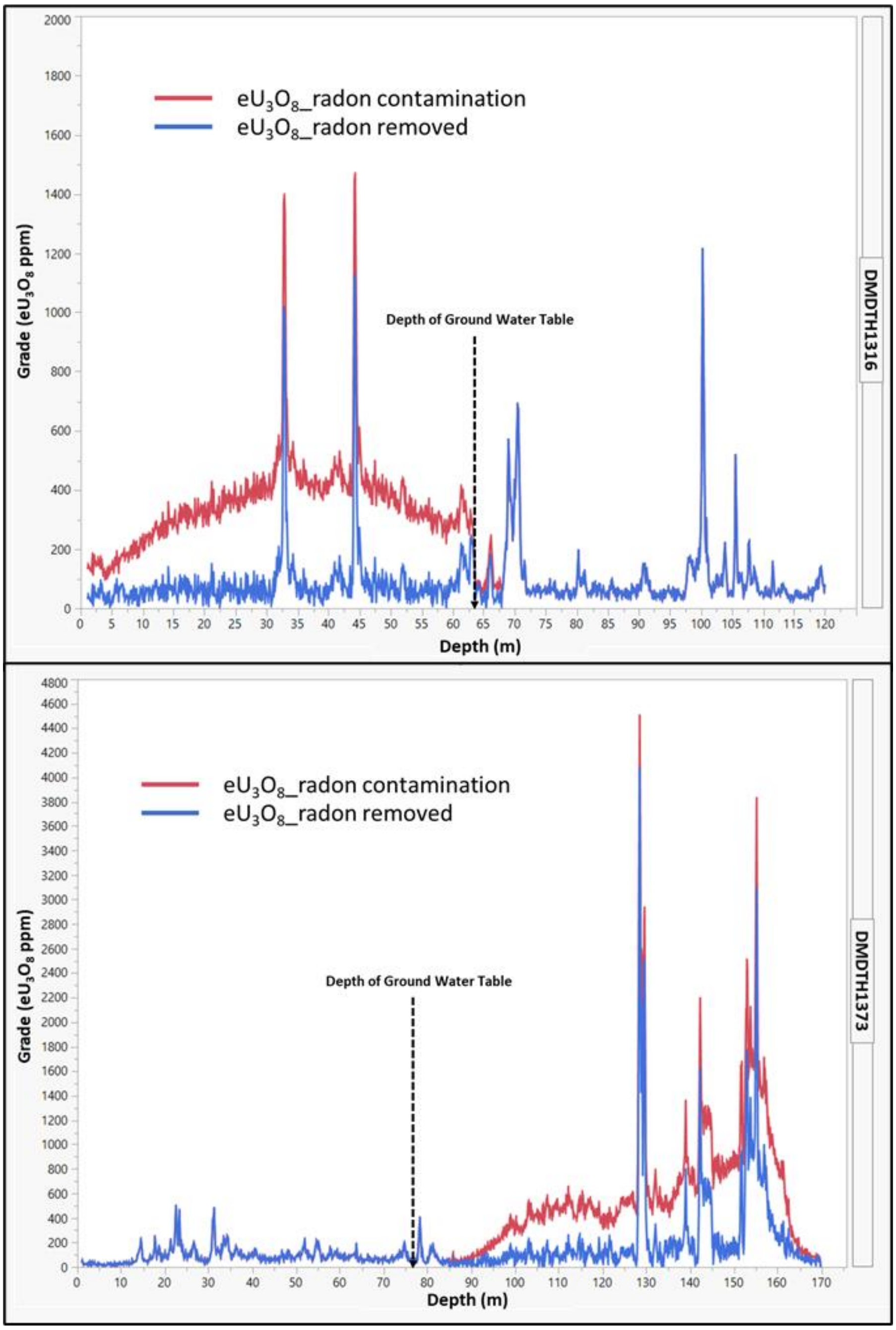

Figure 12-2: Radon contamination and correction of down-hole eU_3O_8 grades for 2021-2023 Drill Holes DMDTH1316 and DMDTH1373 (Dibbwi East deposit)

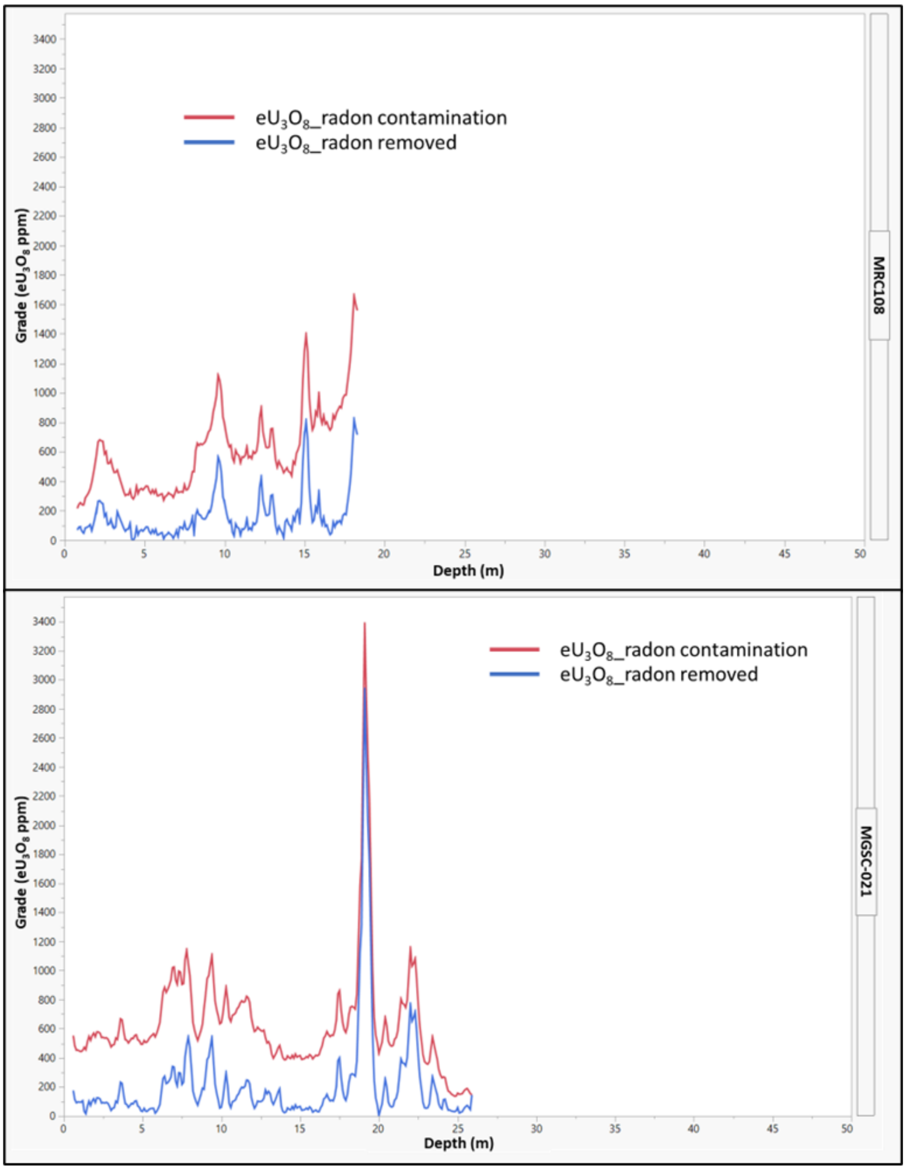

Figure 12-3: Radon contamination and correction of down-hole eU_3O_8 grades for historical drill holes MRC108 and MGSC-021 (Muntanga deposit)

12.2.5. Down-hole radiometric probing vs assay comparison

SRK compared down-hole radiometric probe eU_3O_8 grade data to corresponding geochemical assays for drill holes located on the Muntanga, Dibbwi and Dibbwi East deposits. The comparison was conducted for each deposit separately and data were segregated into historical data collected by Denison and recent data collected by GoviEx. This analysis was completed to establish a radiometric-grade correlation to use for mineral resource estimation purposes, details of which are provided in Section 14.5.2.

12.3. Qualified person comment on data verification

Mr. Deiss has reviewed and analysed the results of data verification programmes conducted by previous companies and accepts the results of these programmes. Based on this review and analysis, along with the additional data verification conducted directly by SRK, Mr. Deiss is of the opinion that the Project drill hole database is adequate to support the current geological interpretation of the Project uranium deposits and to support the estimation of Mineral Resources.

13. Mineral processing and metallurgical testing

13.1. Introduction – History and summary

Several bench scale mineralogical studies and column testwork have been completed on the Muntanga and Chirundu deposits by the previous owners. The work is summarized in this section, with the majority of the information extracted from: NI 43-101 Technical Report prepared by CSA Global in 2013 (CSA, 2013) on the Muntanga Uranium Project; AFR Pre-Feasibility Report in 2008 (AFR, 2008) prepared for the Chirundu deposit; the report prepared by Mintek for the Chirundu deposit bankable feasibility study "Determination of Uranium Heap Leach Process Design Criteria for the Chirundu Project in Zambia" (Mintek, 2010); Muntanga Project Feasibility Study (MDM Engineering, 2009); and the Dibbwi East NI 43-101 Technical Report prepared by Denison Mines (USA) Corp and Roscoe Postle Associates (Denison Mines (USA) and Roscoe Postle Associates (RPA) Inc, 2012.

The following is summarised from information provided in a report prepared by Mintek, Randburg, South Africa (May 2013) titled "Heap Leach Feasibility Testwork on Mutanga and Dibbwi Ores". Denison submitted to Mintek 1 170 kg and 1 400 kg of diamond drill core samples from the Muntanga and Dibbwi uranium ore deposits respectively. The drill cores were divided into groups according to the production periods planned for the two ore bodies. Composite samples were also prepared. Chemical head assays showed uranium contents (as U) of 200 ppm and 210 ppm for the Muntanga and Dibbwi composite ores.

Both ores were composed of mainly silica (86 %) and alumina (8 %) which are known to exhibit low reactivity to acid media. Iron at between 1.3 % and 1.9 % was found to be the main impurity in both ores.

A summary of 2013 testwork is given below:

- At a crush size of 100% <25 mm, both ore types could be stacked to a height of 6 m and still be permeable to reagent (lixiviant) at an application rate of 10 $L/m^2/h$
- Acid leach bottle roll tests indicated that uranium extraction rates for Muntanga ore are reasonable, with final acid consumption of 3 kg/t and could be leached within three weeks yielding extraction of 88 %
- The optimum conditions to leach the Muntanga ore were concluded to be the addition of 2.5 kg of concentrated sulfuric acid per ton of dry ore during agglomeration, three days curing time and irrigation of the ore with 3 g/L acid solution at an irrigation rate of 6 $L/m^2/h$
- The Dibbwi composite sample exhibited higher acid consumption (12.3 kg/t) and required a longer period of time (80 days) for completion of the leach cycle. A maximum uranium extraction of 79 % was achieved for the Dibbwi ore
- The Dibbwi sample was agglomerated with 10 kg/t of acid, followed by a curing period of seven days and was then irrigated using leach solution containing of 3 g/L acid at an application rate of 15 $L/m^2/h$. Under these conditions, the uranium extraction was improved such that a maximum extraction of 82 % was achieved, most of it in less than two weeks
- The acid consumptions expressed in terms of kg acid consumed per pound of U_3O_8 extracted for the Muntanga and Dibbwi ores were 3.7 kg/lb and 37.3 kg/lb, respectively.

The Chirundu project had bottle roll testing completed and detailed mineralogy. According to a mineralogical report completed previously at SGS, the uranium was observed as a combination of U-Ti oxides (presumably such as brannerite and betafite) and uraninite. The uranium content varied typically between 300 ppm to 400 ppm U_3O_8. Successful leaching of uraninite would require oxidative leaching to oxidise the U(IV) to U(VI). The U-Ti oxides can be very refractory to leaching, but their leaching behaviour is difficult to predict and is best determined experimentally. The association with, and even occlusion of U-minerals in pyrite was mentioned in that report. The liberation of uranium from pyrite would require oxidation and solubilisation of the pyrite, but uranium locked in pyrite was not frequently observed and would therefore probably not be a major consideration for U extraction. During the testwork programme, it was therefore considered important to include tests under both oxidising and non-oxidising conditions. The Gwabi deposit contains about 2 % calcite and ankerite which, being carbonate minerals, are acid consuming. The Njame deposit contains virtually none of these carbonate minerals and would therefore be expected to be less acid-consuming than the Gwabi ore. For this programme of testwork, it was therefore considered important to include leaching tests under both acidic and alkaline conditions.

Both Gwabi and Njame deposits occur at shallow depths with minimal dip, which permits mining by surface mining equipment. It was the intention to construct a heap leach facility ("HLF") at each of the deposits. Heap height of 10 m is currently envisaged, to be irrigated at 10 $L/h/m^2$ with mild acidic ferric liquor, being re-oxidised in the ponds using hydrogen peroxide if oxidising conditions are deemed to be required. The counter-current flow of the ore with intermediate leach solution ("ILS") and raffinate will be employed to increase the solution U-tenor. The current assumption has been that the resin can be loaded to 30 g/l U_3O_8.

From the results obtained by the programme of rolling bottle and column leach tests using acid described in the previous reports, the following conclusions were drawn:

- The Ca content of the Njame ore is mostly <0.1%, whereas the Gwabi ore contains >0.1% Ca and variability samples containing up to 1% Ca have been found. This could indicate a higher acid-consuming calcite content in the Gwabi ore, compared to the Njame ore. Several silicates are reactive to acid and could further increase the acid consumption of the Gwabi ore, but the reaction of acid-consuming silicates during heap leaching can often be controlled somewhat by manipulating the acid concentration in the irrigation liquor, highlighting the importance of continual testwork during operations
- Because both ores contain siltstone, it was suspected there could be a risk of a large proportion of fines occurring as hard lumps which decompose upon wetting which can impair the permeability of the ore during heap leaching. Several tests described in the report, however, indicated that this ore does not exhibit that problem, and additional pre-treatment of the ore like dry scrubbing would be unnecessary
- It is concluded that the uranium that was extracted from both the Njame and Gwabi ore samples leached by chemical dissolution (be it in acid or alkaline medium), and oxidative leaching does not offer any advantage over non-oxidative leaching.

During acidic leaching, the maximum uranium extraction from both the Njame and Gwabi ore is independent of the acid strength, between pH values of 1.2 to 1.8.

For both ores, the maximum uranium extraction is higher during acid leaching than during alkaline leaching. From the Njame ore, a maximum of 80 % to 90 % extraction can generally be obtained by acid leaching (although individual variability samples yielded close to 100 % extraction), but 70 % to 80 % by alkaline leaching. From the Gwabi ore, a maximum of 70 % to 80 % extraction can be obtained during acid leaching, but about 60 % by alkaline leaching.

The acid consumption of blends of both ores increases with increasing acidity, increasing from 12 kg/t to 70 kg/t on the Njame ore and increasing from 75 kg/t to 140 kg/t on the Gwabi ore as the pH is lowered from 1.8 to 1.2. On both ore sample blends, it was possible to keep the acid consumption in rolling bottle tests below 3kg/t during acidic leaching at pH = 1.8 under non-oxidising conditions.

During alkaline leaching, both ore samples consumed zero alkali leach reagents.

A comparison of the rolling bottle leach results on variability samples and their respective lithologies reveals that the Gwabi ore exhibits greater variation in both acid consumption and uranium extraction amongst the different ore lithologies than the Njame ore.

During acid column leaching of Njame ore crushed to <20mm, a final uranium extraction of around 85 % was obtained.

At a stacking height of 2 m, uranium extraction during percolation leaching of Njame ore with irrigation liquor at pH = 1.5 is completed within 30 days, or an irrigation ratio of 2 m^3/t. Using irrigation liquor at pH = 1.5, the final acid consumption was around a very low 3 kg/t. The ore compacted within five days by 12 % to 15 % and then stabilised.

The unscrubbed ore compacted noticeably less than the scrubbed ore. No benefit could be observed from the dry scrubbing of the ore before leaching, thereby confirming that fines occurring as hard lumps are not a significant problem with the Njame ore. The final moisture hold-up in the ore at a steady state was a relatively high 20 %, but ponding never occurred and the ore material seems to exhibit adequate permeability to irrigation liquor to sustain the irrigation rate of 10 L/h/m^2 applied.

Two process options have been investigated: alkaline leach and acid leach. Acid heap leaching was selected based on giving a slightly better overall recovery and leaching rate for all six deposits and it has lower operating and capital costs. Test work has indicated that heap permeability would be good, and that acid consumption would be relatively low in a range of 3 kg/t to 9 kg/t for all deposits except Gwabi which requires 18 kg/t. The process is robust, simple and has a low environmental profile. The nature of the operation will support greater participation by the local labour force. Work has been completed including bottle roll and column testwork, mineralogy and metal recovery and precipitation.

The materials tested in columns represent the broad lithotypes of the Muntanga and Dibbwi resource materials. Additional testwork is required to optimise leach conditions and confirm extractions for composites representing the ore supply for the full resource indicated by the mine plan but results to date are considered to confirm the technical feasibility of acid heap leach technology for the Muntanga and Dibbwi resource materials.

Additional testwork is recommended to optimise the leach parameters (initial acid addition, lift height, etc) and to confirm leach performance on all scheduled resource materials.

Additional testwork is required to better define the accumulation of coextracted metals as a basis for the determination of the solution bleed and treatment which may be required to maintain zero water discharge, leach performance and uranium product quality.

13.2. Sample descriptions/ Ore description

Metallurgical test work was performed on five composites, from all the prospects, with Muntanga and Dibbwi East accounting for approximately 80 % of M&I Mineral Resources. The samples were sourced from existing drillcore from the 2023 drilling programme. They were:

- **Composite 1:** Dibbwi East oxidised; coarse oxidised sandstone containing visible secondary uranium minerals of autunite, umhoite and carnotite
- **Composite 2:** Dibbwi East reduced; black reduced siltstone-sandstone mixed containing finer-grained groundmass with some lithoclasts
- **Composite 3:** Muntanga + Njame; both samples were fine-grained grey to green sandstone-siltstone mix with some graphitic and pyritic material
- **Composite 4:** Gwabi oxidised; medium-grained sized oxidised siltstone
- **Composite 5:** Dibbwi Main oxidised; coarse oxidised sandstone containing visible secondary uranium minerals of autunite, umhoite and carnotite.

13.3. Work scope

Five feed composites were prepared and crushed to 100 % -25 mm. The scope of test work included the following:

- Particle size distribution ("PSD") and chemical head assay
- Curing acid optimisation (agglomeration and soaking) tests
- Iso-pH (constant pH) acid consumption tests
- Uni-axial compression (stacking) tests and hydrodynamic column tests
- Leach column tests (6 m tall, 160 mm ID)
- Ion exchange/ neomembrane filtration/ acid neutralisation/ uranium precipitation
- Geochemical assays on residues and leach liquors.

A 30 kg sub-sample was split out from each composite, and a wet and dry PSD was performed at the following screens: 19, 12.5, 9.5, 6.7, 4.75, 3.35, 2.36, 1.7, 1.18, 0.85, 0.6, 0.425, 0.3, 0.212, 0.15, 0.106, 0.075 and 0.053 mm. The screened masses were combined into four size classes (+12.5 mm, +3.35 mm, +1.18 mm and -1.18 mm). Each of the four size classes was analysed for U_3O_8 by X-ray fluorescence ("XRF") 15 and multi-element inductively coupled plasma-optical emission spectroscopy ("ICP-OES") (Al, Ca, Co, Cr, Cu, Fe, Mg, Mn, Ni, Pb, Ti, V and Zn).

A composite head sample was reconstituted from the four size classes, pulverised and analysed for the following:

- XRF15 (U_3O_8)
- ICP-OES: Solid samples are fused with sodium peroxide and sodium carbonate and dissolved with HCl and H_2O
- Iso-pH (constant pH) acid consumption tests
- Acid generation potential (AP): Calculated by multiplying Sulfide S by a factor
- Neutralising potential (NP): The sample is contacted with 0.1 M HCl until pH 2-2.5. Excess acid is neutralised with NaOH
- Acid-base accounting (ABA) = AP – NP
- Total S: combustion
- Sulfate S: gravimetric method
- Sulfide S: Sample is dissolved in trichloroethylene to extract elemental S. The residue is boiled in sodium carbonate, filtered and the filtrate is complexed with barium sulfate to determine sulfate S by gravimetry. Sulfide S is determined by combustion from the filtered solids
- TCLP (toxicity characteristic leaching procedure) by EPA method 1301. TCLP solutions were analysed for the following
  - ICP-OES (Al, Ca, Fe, Mg, S, Si)
  - ICP-MS (Ti, V, Cr, Mn, Co, Ni, Cu, Zn, As, Se, Sr, Zr, Mo, Ag, Cd, Sn, Sb, Ba, Hg, Pb, Th and U)
  - Cl, F and PO_4 by wet chemistry methods.

A second 30 kg (-25 kg) sub-sample was removed and rotary-split into smaller masses (~3 kg) for curing acid optimisation tests and geomechanical tests. A 30 kg sub-sample was split out from each composite, and a wet and dry particle size distribution ("PSD") was performed at the following screens: 19, 12.5, 9.5, 6.7, 4.75, 3.35, 2.36, 1.7, 1.18, 0.85, 0.6, 0.425, 0.3, 0.212, 0.15, 0.106, 0.075 and 0.053 mm. The screened masses were combined into 4 size classes (+12.5, +3.35, +1.18 and -1.18 mm). Each of the four size classes was analysed for U_3O_8 by XRF15 ("X-ray Fluorescence") and multi-element ICP-OES ("Inductively coupled plasma-optical emission spectroscopy") (Al, Ca, Co, Cr, Cu, Fe, Mg, Mn, Ni, Pb, Ti, V and Zn).

A composite head sample was reconstituted from the four size classes, pulverised and analysed for the following:

- XRF15 (U_3O_8)
- ICP-OES: Solid samples are fused with sodium peroxide and sodium carbonate and dissolved with HCl and H_2O
- Iso-pH (constant pH) acid consumption tests
- Acid generation potential ("AP"): Calculated by multiplying Sulfide S by a factor
- Neutralising potential ("NP"): The sample is contacted with 0.1 M HCl until pH 2-2.5. Excess acid is neutralised with NaOH
- Acid-base accounting ("ABA") = AP – NP
- Total S: combustion
- Sulfate S: gravimetric method
- Sulfide S: Sample is dissolved in trichloroethylene to extract elemental S. The residue is boiled in sodium carbonate, filtered and the filtrate is complexed with barium sulfate to determine sulfate S by gravimetry. Sulfide S is determined by combustion from the filtered solids.

TCLP by EPA method 1301. TCLP solutions were analysed for the following:

- ICP-OES (Al, Ca, Fe, Mg, S, Si)
- ICP-MS (Ti, V, Cr, Mn, Co, Ni, Cu, Zn, As, Se, Sr, Zr, Mo, Ag, Cd, Sn, Sb, Ba, Hg, Pb, Th and U)
- Cl, F and PO_4 by wet chemistry methods.

A second 30 kg (-25 kg) sub-sample was removed and rotary-split into smaller masses (~3 kg) for curing acid optimisation tests and geomechanical stacking tests ("STs").

The column tests aimed to determine maximum uranium dissolutions, leach kinetics and reagent consumptions at the target stacking height of 6 m. In addition, the curing acid was first optimised with agglomeration/ soaking tests to maximise initial uranium dissolution. The pregnant leach liquor accumulated over the first three weeks was treated by ion exchange, neomembrane filtration and uranium precipitation. The barren solution was recycled over the columns to study the build-up of impurities. However, since the leach kinetics was fast, only one pass of the barren solution was recycled. Geochemical assays were performed on the feed solids, accumulated liquors, wash liquors and final residues to measure the potential release of impurities from the residues.

13.4. Mineralogy and geometallurgy

13.4.1. Geometallurgy

Selected uranium samples were evaluated by bulk mineralogical analysis ("BMA") and trace mineral search ("TMS") using QEMSCAN at SGS Lakefield Oretest in Brisbane, Queensland/ Australia.

The tests aimed to characterise:

- The natural liberation of quartz from the conglomerate phases and
- The occurrence and mineralogy of uranium phases, including grain size, association and liberation.

Key findings from the evaluation:

1. The majority of the uranium (~95 %) was contained in the U-Ca-P phase, nominally referred to as autunite. The 'Other Minerals Group" (which makes up approximately 5 % of the U elemental deportment) was comprised predominantly of brannerite and coffinite
2. The vast majority (>90 %) of the U-bearing mineral particles studied in the test program were liberated whilst <10 % remained unliberated. The U-bearing minerals in the latter category were predominantly attached to the quartz boundaries
3. The U-bearing minerals generally appeared to be discrete grains (not intergrown with other minerals), suggesting that it should be possible to achieve high levels of liberation of the U-bearing minerals
4. Between 50 % to 60 % of the U-bearing particles in the test programme were associated with quartz, but the average grain size was small, so the proportion of the total deportment was low at ~2 %. The dominant U-bearing mineral autunite was associated with the pores of the host rock (sandstone) not within the clay cement
5. The U-bearing mineral autunite does not occur within the quartz grains. This suggests that it should be possible to upgrade the ore by preferential removal of the quartz grains (~1 mm diameter and more)
6. The data suggests that the timing of the U mineralisation was post-depositional, which is supported by the low association between the U-bearing minerals and the quartz grains and clay cement.

13.4.1.1. Dibbwi East

Uranium occurs mainly as U-phosphate and UAlSi-phosphate, with uranium as autunite, coffinite, Ti-coffinite, uraninite, U-phosphate and UAlSi-phosphate in Zone 3. The samples show similar bulk mineralogical compositions; for example, gangue minerals are dominated by albite, kaolinite, microcline, muscovite and quartz, as determined by X-ray diffraction ("XRD"), but with varying proportions.

The source of the uranium is believed to be the surrounding Proterozoic gneisses and plutonic basement rocks. Having been weathered from these rocks, the uranium was dissolved, transported in solution and precipitated under reducing conditions in siltstones and sandstones. Post-lithification fluctuations in the groundwater table caused dissolution, mobilisation and redeposition of uranium in reducing, often clay-rich zones and along fractures.

Mineralisation is not strictly associated with a particular unit in the stratigraphic section. It was observed to occur in both the fine-grained and coarser material and mudstones especially where fractures and mud balls occur. Some mineralisation occurred in association with Mn oxide or disseminated with pyrite. Mineralisation in some bore holes was seen to occur where there was a grey alteration, limonite and feldspar alteration and in dark grey mudstones. The strata dip in the south-easterly direction and mineralisation seems to occur along dip.

In 2011, Denison Mines Zambia Limited requested ALS Chemex Johannesburg to conduct a mineralogical analysis of four uranium ore samples shown in Table 13-1 to identify the uranium and gangue minerals present in the various strata, including both low and high-grade zones. The samples were in the form of drill cores.

Table 13-1: Sample list for mineralogical study (Source: (Denison-RPA, 2012), 2012)

Sample number	Depth from [m]	Depth to [m]	Sample type	Weight [kg]	U grade [ppm]
F000988	96.85	96.95	SPOT	0.3694	2 988
F000989	93.7	93.8	SPOT	0.4562	1 958
F000990	54.3	54.4	SPOT	0.231	724
F000991	17.3	17.4	SPOT	0.2996	1 608

The mineralogical analysis, using an automated Mineral Liberation Analyzer ("MLA"), was used to determine the uranium minerals (Table 13-2 and Table 13-3) present along with the associated gangue (ALS Minerals, 2011).

The data indicates that the main uranium phase in sample F00988 (Muntanga) was coffinite, which accounted for 97 wt% of the uranium ore minerals in the sample. There was also some Ti-bearing coffinite in the sample.

Coffinite was the most abundant ore mineral in F00989, accounting for nearly 67 wt% of the ore minerals. It was predominantly Ti-coffinite (55 wt%), with lesser coffinite (11 wt%). Autunite (28 wt%) was a major ore mineral in this sample, which contained a significant amount of Brannerite (6 wt%). Despite having the second-highest grade of the samples submitted, there was difficulty in finding the ore minerals in this sample, hence the lower particle counts recorded.

Sample F00990 had less coffinite (26 wt%) than the other two samples, with the most abundant ore mineral being phurcalite (72 wt%). There was a small amount (2 wt%) of gastunite present.

Phurcalite accounted for almost all the uranium ore minerals in sample F00991, with minor coffinite and autunite making up about 1 wt% of the ore minerals.

Table 13-2: Relative uranium mineral abundance (Source: Denison and RPA, 2012)

Mineral	Relative abundance [wt%]				Particle count			
	F00988	F00989	F00990	F00991	F00988	F00989	F00990	F00991
Brannerite	0.1	5.9	0.3	0.0	6	1	23	0
Coffinite	97.3	11.2	23.4	0.6	785	5	296	85
Ti-Coffinite	2.2	55.4	2.6	0.2	239	7	164	37
Phurcalite	0.1	0.0	71.8	98.9	4	0	556	427
Curite	0.0	0.0	0.0	0.0	1	0	0	0
Gastunite	0.4	27.5	2.0	0.3	79	10	134	57
Total	**100.0**	**100.0**	**100.0**	**100.0**				

Table 13-3: Uranium distribution (%) (Source: Denison and RPA, 2012)

Mineral	F00988	F00989	F00990	F00991
Brannerite	0.03	4.74	0.15	0.00
Coffinite	98.23	15.47	22.53	0.59
Ti-Coffinite	1.33	45.69	1.46	0.09
Phurcalite	0.06	0.00	74.14	99.10
Curite	0.01	0.00	0.00	0.00
Gastunite	0.35	34.10	1.72	0.22
Total	**100.00**	**100.00**	**100.00**	**100.00**

13.4.2.Type of mineralisation

13.4.2.1. Disseminated uranium mineralisation

Disseminated uranium mineralisation occurs in sandstones, conglomerates, and within mud layers, mud balls and mud flakes. The uranium is present as interstitial fine-grained crystals or small amorphous masses constituting less than 1 % by volume, if visible at all.

Figure 13-1: Mineralisation associated with Mn oxide (Black) (Source: Denison and RPA, 2012)

Grades vary considerably between zones of disseminations, approximately 20 ppm to 2 052 ppm U_3O_8 (geochemical) in mineralisation is thought to be solely of a disseminated nature, although mud replacement material may have been contained within the core and therefore not visible during logging leading to higher values.

Lithological units containing Fe-oxide and uraniferous mineralisation returned moderate to high assays, as did material containing sulfides (pyrite). Samples from MR05, MR08, MR09, MR10 and MR11 contain both sulfides and micas and disseminated U_3O_8 and were expected to return low assays.

The presence of sulfides alongside uranium oxides may indicate a transitional zone and/or preferential replacement/ reduction of uranium compounds by one chemical route over another (such as decaying organic matter over oxidation of sulfides) as uraniferous groundwaters moved through the lithologies.

13.4.2.2. Uranium mineralisation associated with mudstones and siltstones

An association between uranium mineralisation (as replacements and selvages) is evident at all prospects. The muddy lithologies include mud balls (within sandstones), flakes and interbeds. In some cases, mud balls may be completely replaced by mineralisation (Figure 13-2).

The degree of replacement varies from fully replaced mud balls to those with a thin selvage of mineralisation whilst others are unmineralised. This is attributed to:

- Different ground water chemistry
- Differing volumes of reducing matter within the mud (fully replaced material may have been a peat-like material) and
- The porosity of the muddy lithology during the influx of uraniferous ground water.

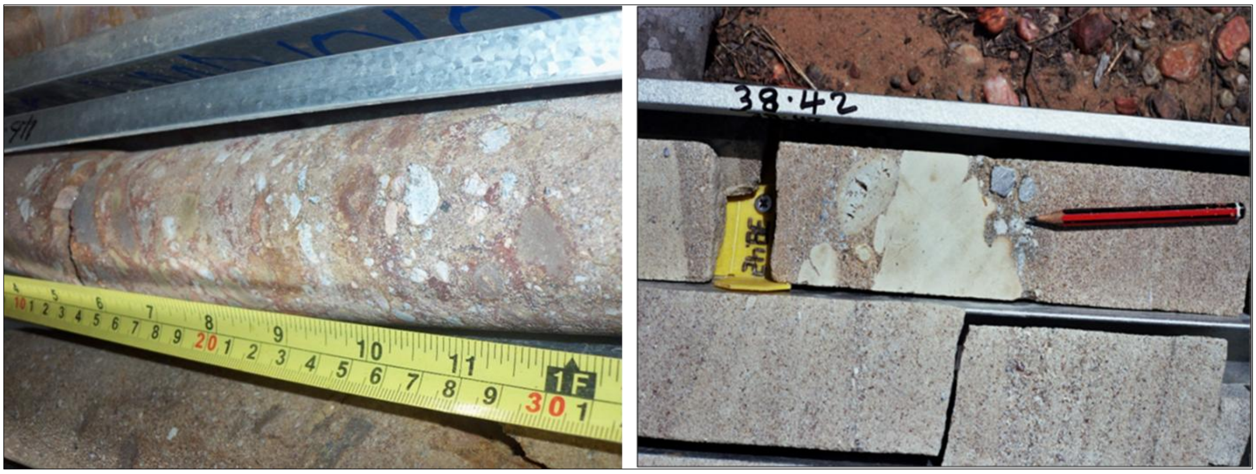

Figure 13-2: Mudclasts (Source: Denison and RPA, 2012)

13.4.2.3. Fracture-hosted uranium mineralisation

Drilling intersected several fractures and fault rocks. The fractures intersected in the core were generally steep (although several shallower-angled fractures were logged). Mineralisation is seen as crystal coatings on surfaces and as concentration close to surfaces (Figure 13-3). Most notably at the Dibbwi-Muntanga-Dibbwi corridor, these fractures are coated with black Fe/ Mn oxides which in turn may be coated with secondary uranium phosphate mineralisation (autunite, meta-autunite and selenite).

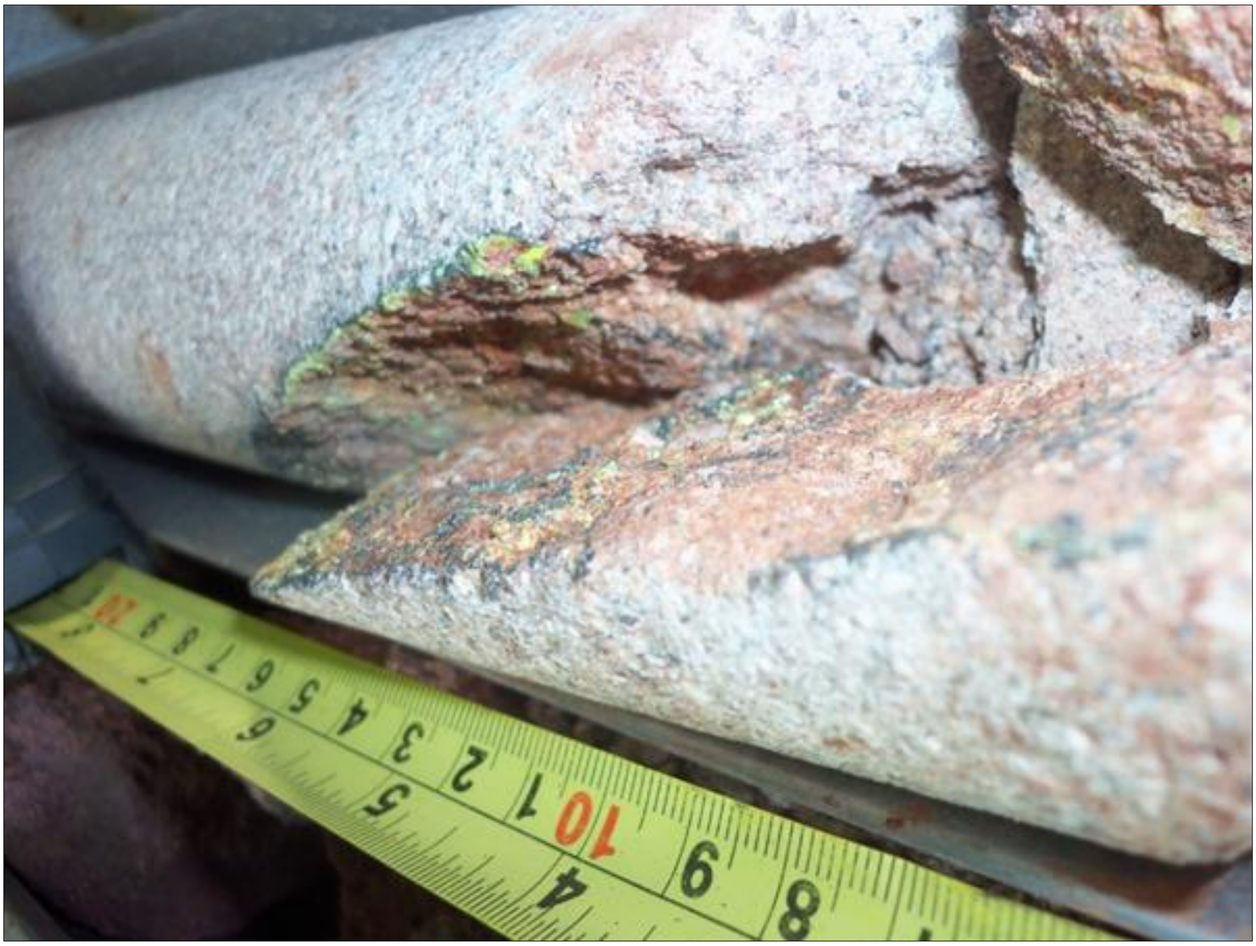

Figure 13-3: Mineralisation in a fracture with the presence of Mn oxide (Source: Denison and RPA, 2012)

13.4.2.4. Uranium mineralisation associated with pyrite

Grains and poorly defined blebs of pyrite occur throughout all the sedimentary lithologies of the Project area. Uranium mineralisation may be elevated in some (relatively) pyrite-rich zones.

The presence of sulfides near uranium oxides may indicate a transitional zone and/or preferential replacement/ reduction of uranium compounds by one chemical route over another (such as decaying organic matter over oxidation of sulfides) as uraniferous groundwaters moved through the lithologies.

13.4.2.4.1. Njame and Gwabi

At Njame and Njame South, the uranium mineralisation occurs at the interface between siltstones and sandstones at redox boundaries. Approximately 25 % of the Njame mineralisation is siltstone hosted, with the balance in coarser-grained sandstones and grits.

Drilling conducted by AFR (AFR, March 2008; April 2012) identified two main mineralised horizons; the thickest, most consistent and highest grade is the lower horizon within the second sequence from the base. Drilling was carried out along the entire length of the 5km long system, with uranium mineralisation encountered along the entire length. Unlike the high-energy sandstone and grit horizons, which show very rapid changes over several tens of metres, the siltstone horizons are generally laterally continuous for hundreds of metres, except where younger grit/ sandstone channels have cut through them. There is a clear stratigraphic control on mineralisation at the deposit scale, although structural control may be present on a larger scale.

Similarly to Njame, the uranium mineralisation at Gwabi is related to the redox front; there is one main mineralised horizon which appears to be controlled by both lithology and the redox boundary. It is hosted by the coarse-grained sediments that are interpreted to be the along-strike continuation of the Escarpment Grits which host the Njame uranium mineralisation. Uranium mineralisation at the Gwabi deposit occurs in red, oxidised, coarse-grained sandstones, grits and pebble conglomerates which overly a green, non-mineralised, reduced silty-shale horizon. This is interpreted to represent a major redox boundary and may be the regional unconformity between the Upper and Lower Karoo.

Uranium mineralisation at the Gwabi deposit is strata bound and occurs in red, oxidised, coarse-grained sandstones, grits and pebble conglomerates which overly a green, non-mineralised, reduced silty-shale horizon. This is interpreted to represent a major redox boundary and may be the regional unconformity between the Upper and Lower Karoo. The mineralisation forms a broadly tabular body, which dips very gently to the southeast and occurs at very shallow depths between 3 m and 29 m below surface.

Uranium occurs largely as uraninite (up to 5 % of all uranium in any sample), with accessory autunite (up to 30 % in the oxidised zone), coffinite (0 % to 25 %), davidite and brannerite (up to 25 %) and is associated with pyrite. Dolomite is present in both materials but is in more abundance in Gwabi.

13.4.2.5. Particle size distributions

The PSDs of the head samples (wet and dry basis) and the column leach residues (wet basis) are plotted as shown in Figure 13-4. The head samples (both wet and dry) meet the PSD criteria proposed by Garcia and Jorgensen (1997) of less than 10 % to 15 % -75 µm and by Scheffel (2017) of less than 10 % and 14 % -150 µm. Although the samples contain high percentages of sand (-4.75mm), this does not necessarily inhibit permeability, but it has been shown that the presence of large amounts of silt + clay (-75 µm) may result in poor permeability as it may cause pores to collapse and flow channels to block. The fines content increased during the column leach to between 14.7 % and 15.4 % -150 µm. This is close to the limit proposed by Scheffel (2017). The residues contained between 66.8 % and 75.1 % -1.18 mm material indicating high levels of fine sand.

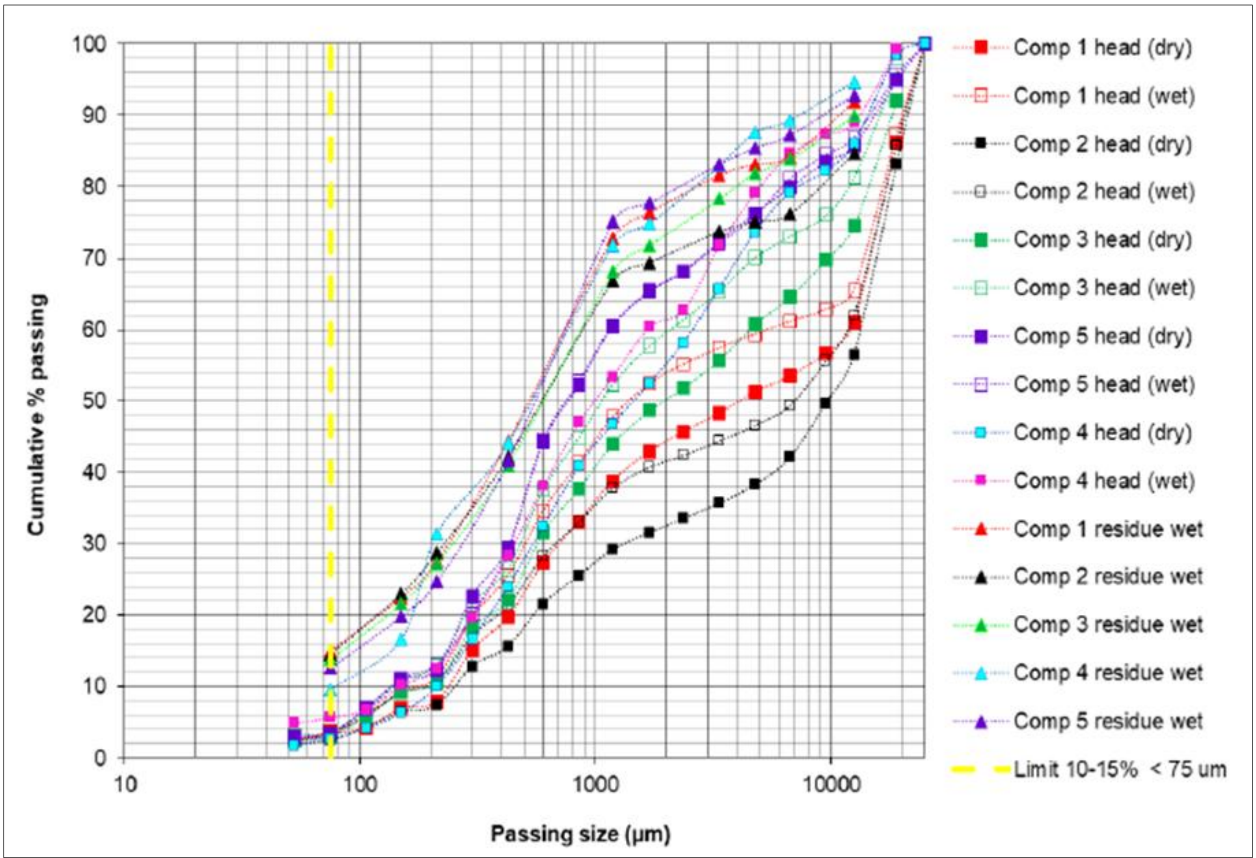

Figure 13-4: Particle size distribution

13.5. Comminution and beneficiation

13.5.1. Comminution

Core samples were combined into six composites, as shown in Table 13-4, and submitted for the following tests:

- Uniaxial compressive strength (“UCS”)
- Bond Crushability (Impact) Work Index (“CWI”).

The ore was too soft to obtain a reliable UCS result. Results of the CWI tests are shown in Table 13-4:

Table 13-4: Bond ball work index results

Description	Sample ID	Minimum [kWh/t]	Maximum [kWh/t]	Average [kWh/t]	75th Percentile	S.G [t/m^3]
Dibbwi East Oxi	Composite 1	2.5	6.6	4.1	5.1	2.31
Dibbwi East Red	Composite 2	3.4	11.7	6.8	8.1	2.49
Njame Red	Composite 3	2.1	8.9	4.6	5.5	2.52
Gwabi Oxi	Composite 4	1.4	10.3	5.0	6.3	2.37
Dibbwi Main Oxi	Composite 5	2.6	7.2	4.3	4.6	2.25
Njame Oxi	Composite 6	4.0	8.3	6.3	7.7	2.27

All the composites are classified as very soft withe average Wi values less than 10 kWh/t.

13.6. Radiometric sorting

13.6.1.Introduction

Rados International Technologies completed on-site testing for particle sorting (“PS”) and bulk ore sorting (“BOS”) using Rados XRF and drill core analyser (“DCA”) on site. The Rados XRF+ DCA is a mobile unit developed for ore sorting amenability testing on-site for data capturing for process design on highly variable multi-mineral ore deposits, before mining access to bulk ore samples.

Figure 13-5: Rados XRF + DCA

The analyses from the XRF+ DCA are equivalent to those obtained from a full-size Rados ore sorter and allow an assessment of the ore's sorting potential.

13.7. Methodology

Testwork was conducted onsite by GoviEx technicians with support from Rados staff. The automated test process included:

- Analysing the drill cores using the DCA automated motion control and X-ray system, with scans taken approximately every 10 mm
- Removing the tray from the analysis chamber once the analysis was completed.

The average time interval between the analysis of each tray during steady-state operation was five minutes. The Rados DCA recorded essential data, including relative concentrations of U, Fe, Ca, Rb, and Pb. These elements were selected due to their significant variations between samples.

13.8. Results

13.8.1. Summary of distribution

Rados analysed a total of 722 trays of drill core, covering 3 012 m from 77 drill holes. The average Rados uranium estimate for each tray and the coefficient of variance ("CoV" %) in uranium analyses within each tray are presented in Figure 13-6. The CoV (%) represents the ratio between the standard deviation of individual particle analyses and the average analysis for the whole sample. A larger CoV indicates greater interparticle variation in the sample, presenting a better opportunity for ore sorting.

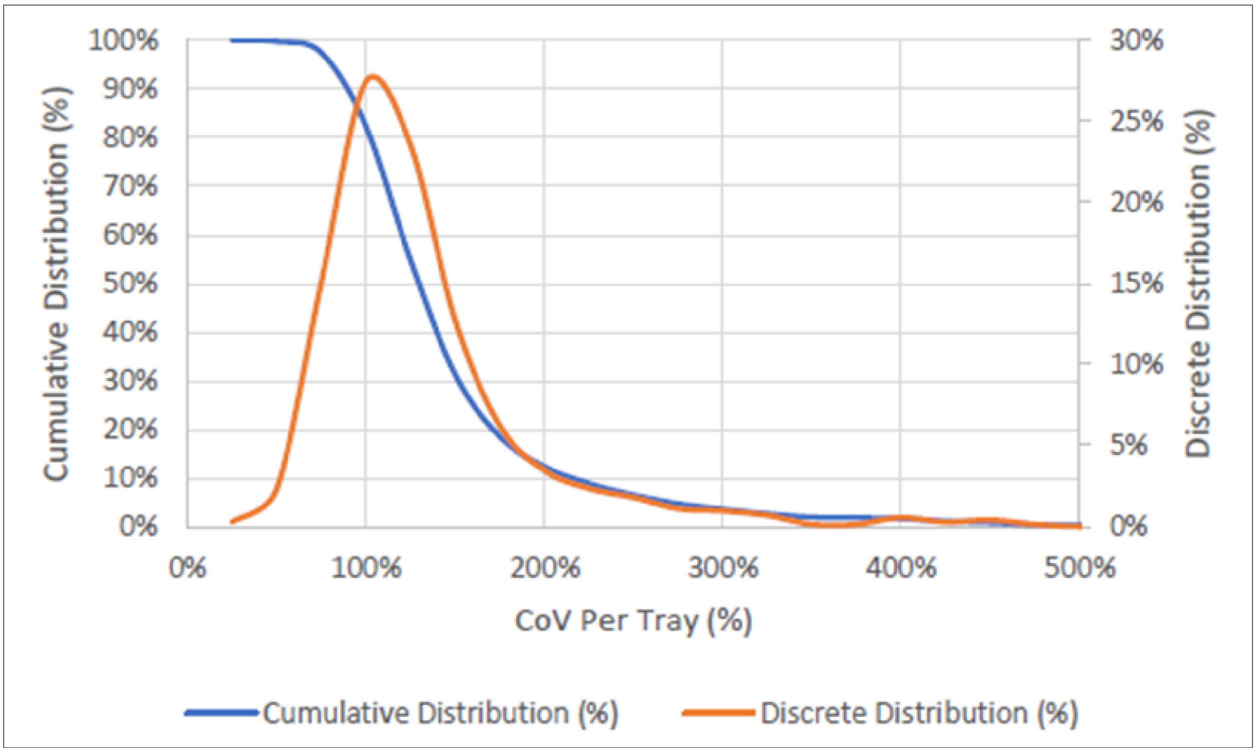

Figure 13-6: Distribution of CoV among Muntunga drill core trays (Source: RADOS, 2024)

The distribution shows that the median (50th percentile) CoV is 129 %, indicating significant variation in half of the samples. Additionally, the 95th percentile CoV is 264 %, highlighting that the top 5 % of the trays exhibit very high interparticle variation, which is highly favourable for ore sorting.

13.8.2. Calibration

Rados compared the analyses of 229 drill core sections to the associated assays provided by the client to generate a calibration relationship based on the Rados U analysis. The sections used for XRF analysis calibration ranged from 0.3 m to 1.35 m, with a weighted average length of 0.62 m. The initial calibration indicates a strong correlation, with an R^2 of 0.99 for uranium when the data is aggregated into approximately ten equivalent-length sections (around 6.2 m).

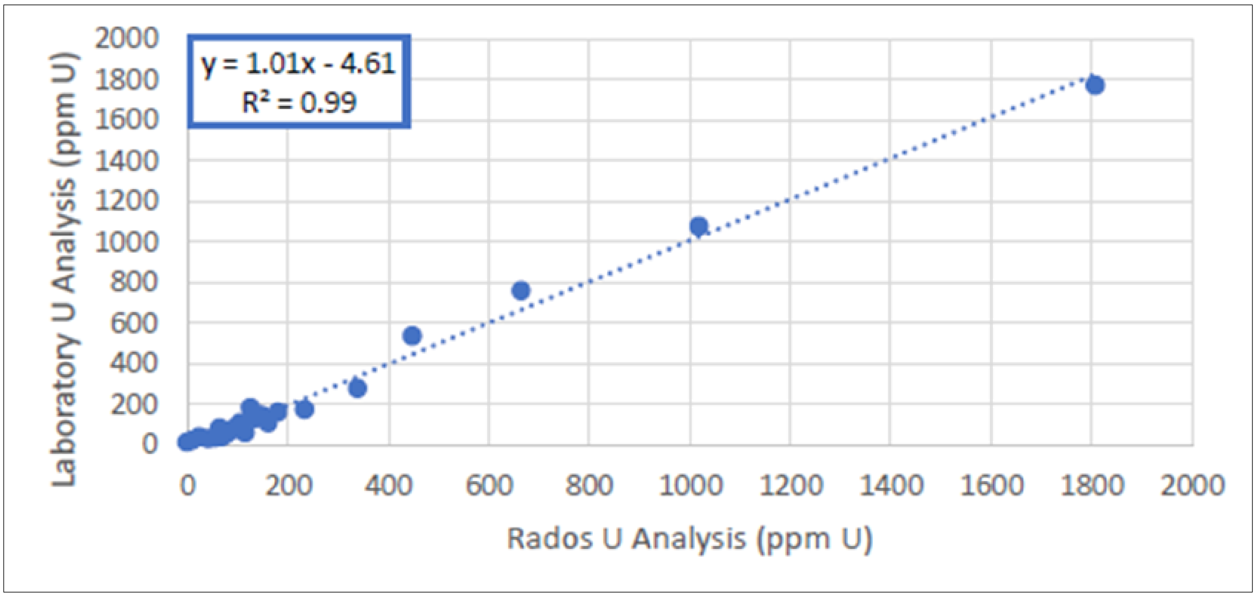

Figure 13-7: Rados DCA calibration results for uranium

The large calibration dataset allowed for the development of machine learning algorithms to provide calibrations for the Fe, Ca, Rb and Pb. The results of the calibrations are presented in the figures below and demonstrate robust correlations between the Rados analyses and the laboratory grades with correlation coefficients >0.9. The accuracy of the calibration in regions below 50 ppm U is demonstrated in Figure 13-8. The sorter identified all four particles as having a U analysis of 50 ppm or below.

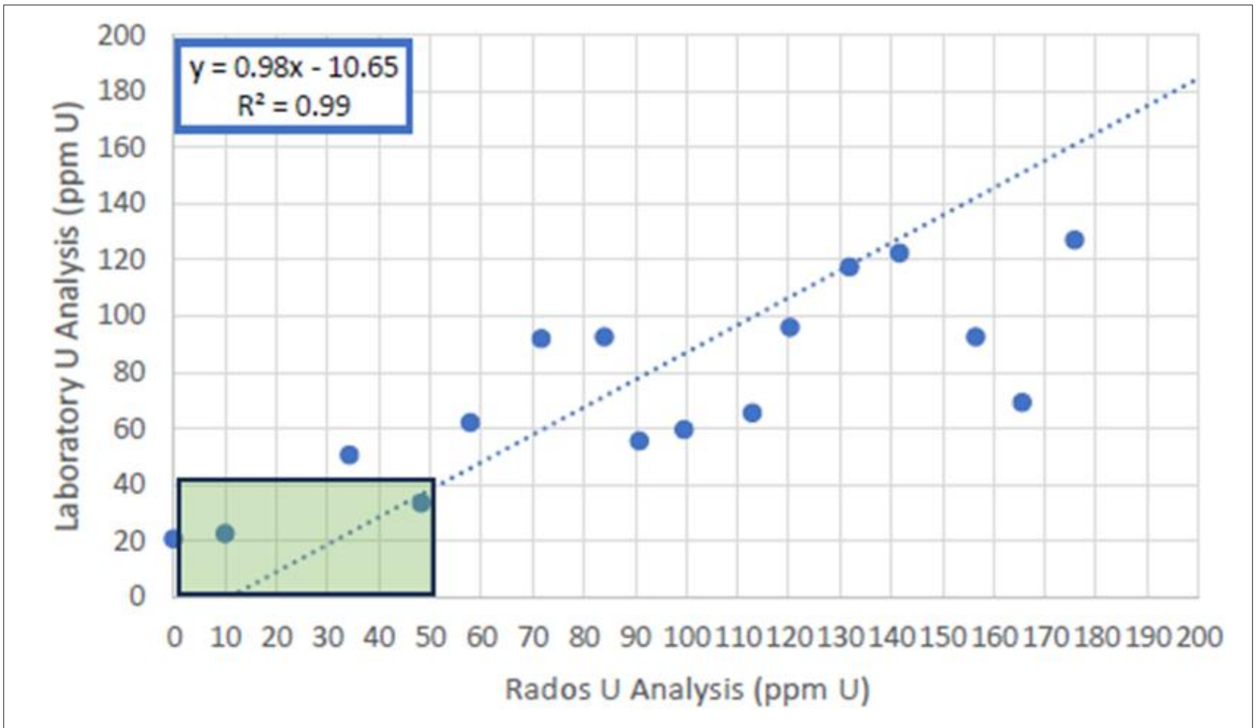

Figure 13-8: Calibration of Rados uranium analysis (Source: Rados, 2024)

When the calibration was redone for shorter lengths of drill core (3.1 m equivalent length), it was noted that out of eight data points that the sorter identified as <50 ppm, one data point was actually 80 ppm and was therefore would be incorrectly discarded by the Rados sorter if a threshold of 50 ppm was selected.

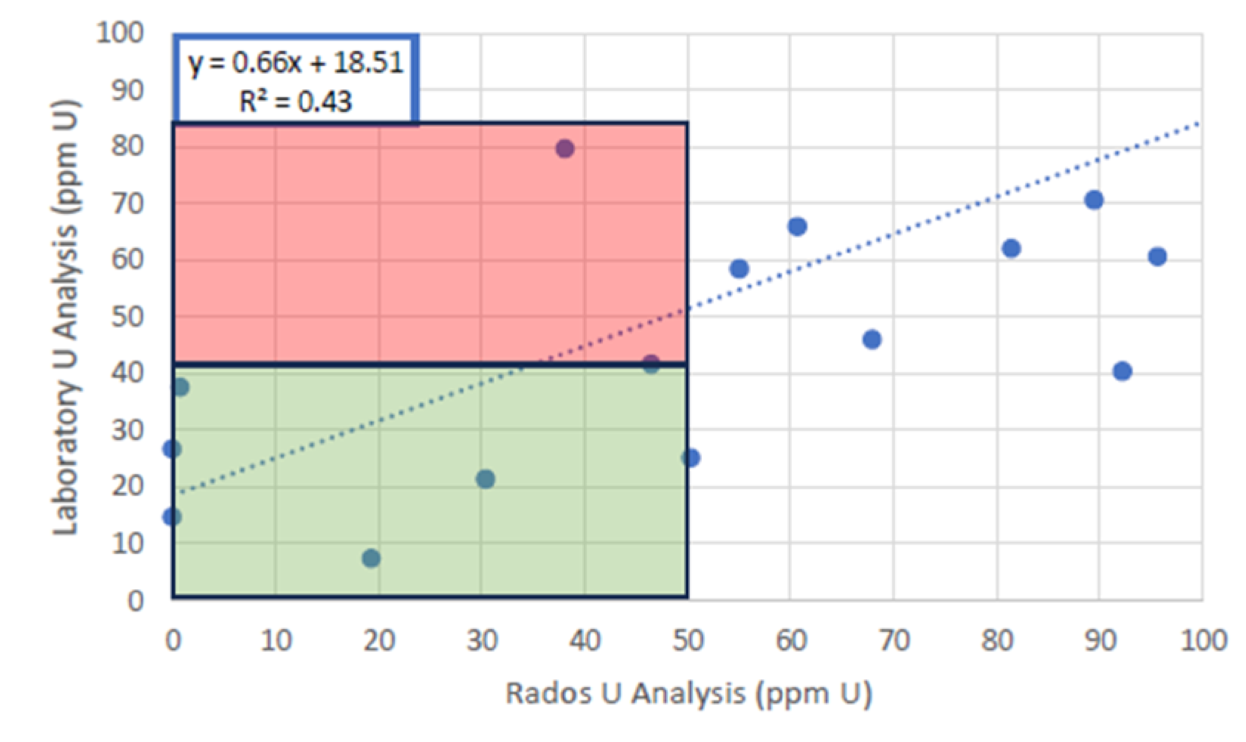

Figure 13-9: Results from calibration (per 3.1m of drill core)

However, even when sorting at the smallest interval available (0.61 m equivalent length) with a cut-off of 50 ppm U, the average grade of the sorter discard was 32.2 ppm U, demonstrating that the sorter can maintain a discard grade below a 50 ppm U cut-off.

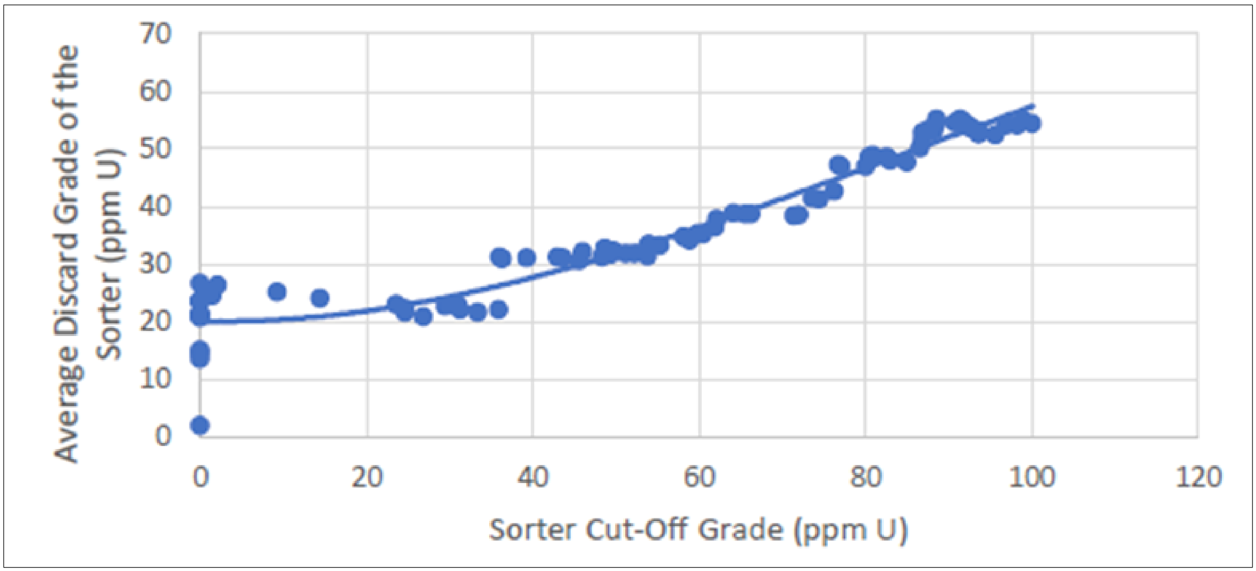

Figure 13-10: Relationship between Rados COG and discard grade for uranium assay sections (Source: Rados, 2024)

The graphs below show the comparison between the laboratory assay results and the Rados-determined uranium content for all 229 calibration data points. The trend indicates that the Rados analyser can effectively track the uranium grade across a wide range of concentrations. For ore sorting, the calibration can be optimised even further around specified COGs, improving the accuracy of classification.

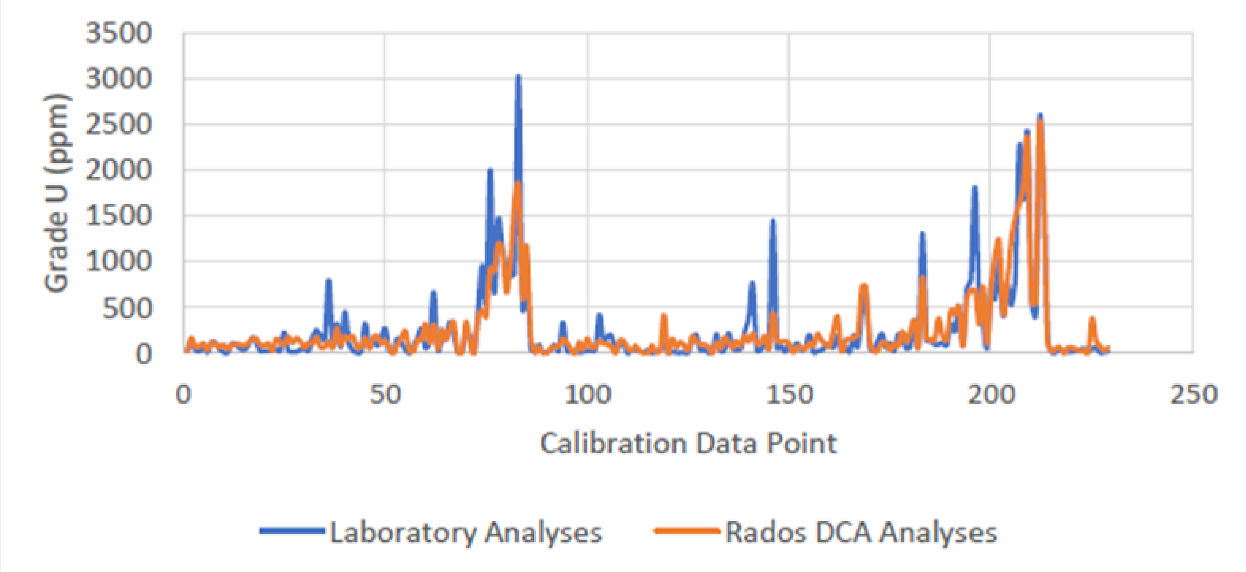

Figure 13-11: Comparison of Rados and laboratory assays (Source: Rados, 2024)

Rados identified the following distribution of uranium analyses for 10mm sections of all the drill cores analysed. This distribution is based on 364 931 data points collected. The distribution highlights the presence of waste and low-grade uranium in the dataset, which can be exploited and removed by ore sorting. The 50th percentile (median) is 90 ppm U, meaning half of the samples have a uranium grade below 90 ppm. Similarly, the 95th percentile is 664 ppm U, meaning that the top 5 % of the samples have a uranium grade above 664 ppm.

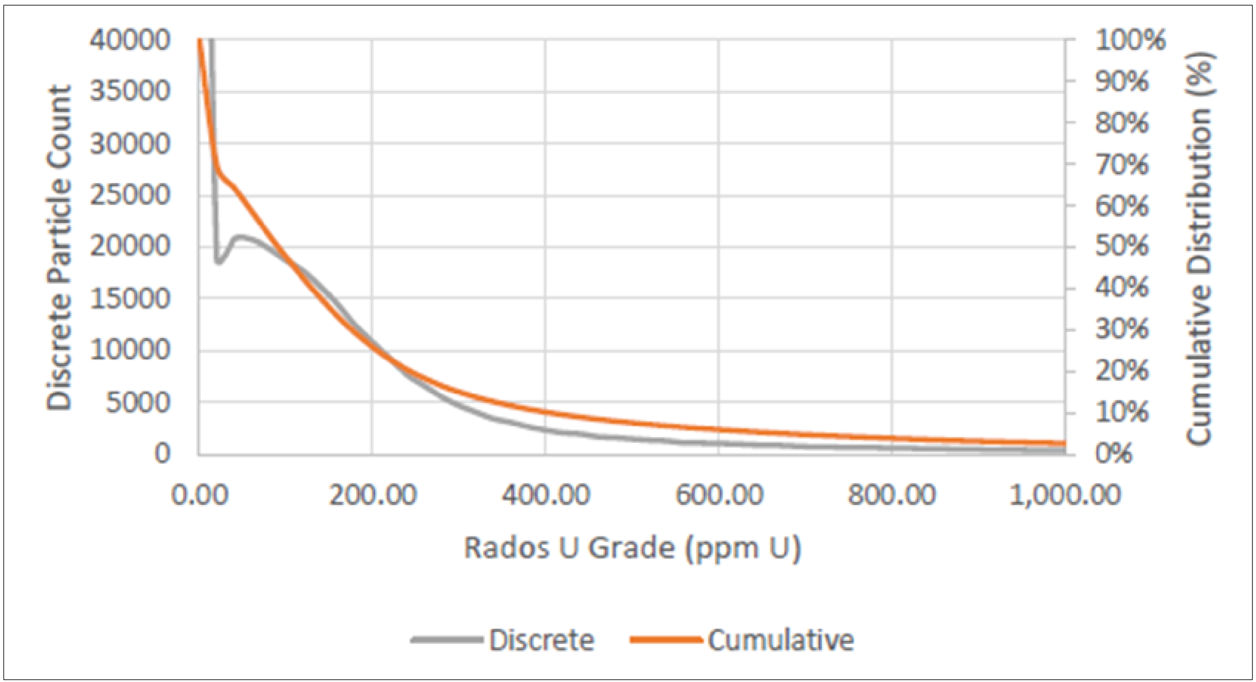

Figure 13-12: Distribution of Rados U analyses (Source: Rados, 2024)

13.8.3.Simulated particle sorting

The Rados analysis was conducted on each 10mm section of the drill core, enabling evaluations for each tray, each hole, and the entire analysed drill core database. Each 10 mm section represents ore particles of 10 mm in size. By accumulating these 10 mm portions into sections of larger portions (30 mm, 50 mm 80 mm, 100 mm and 1 000 mm), it becomes possible to simulate the in-situ heterogeneity of the orebody at various sorting parcel sizes. This provides valuable insights into the potential for different ore sorting options, allowing for the determination of the trade-off between particle size and sorter performance The results show that the Rados particle sorter can effectively upgrade the uranium content of the ore from Muntunga while maintaining high uranium recoveries. The results illustrate various operating points that the Rados sorter can be set at to upgrade the ore.

For example, with a threshold of >80 ppm, the Rados sorter can upgrade the U content of the feed >1.87 times, reduce the mass that needs to be processed by ~30 %, and recover 95 % of the uranium.

Data from the DCA allows for simulation to understand how the sorting of different-size fractions would perform. The graphs below summarise the results for sorter concentrate uranium grade, sorter discards uranium grade, and recovery achieved for five different particle sizes (Rados, 2024). Generally, the results indicate that as particle size increases, uranium recovery at the same mass pull decreases,

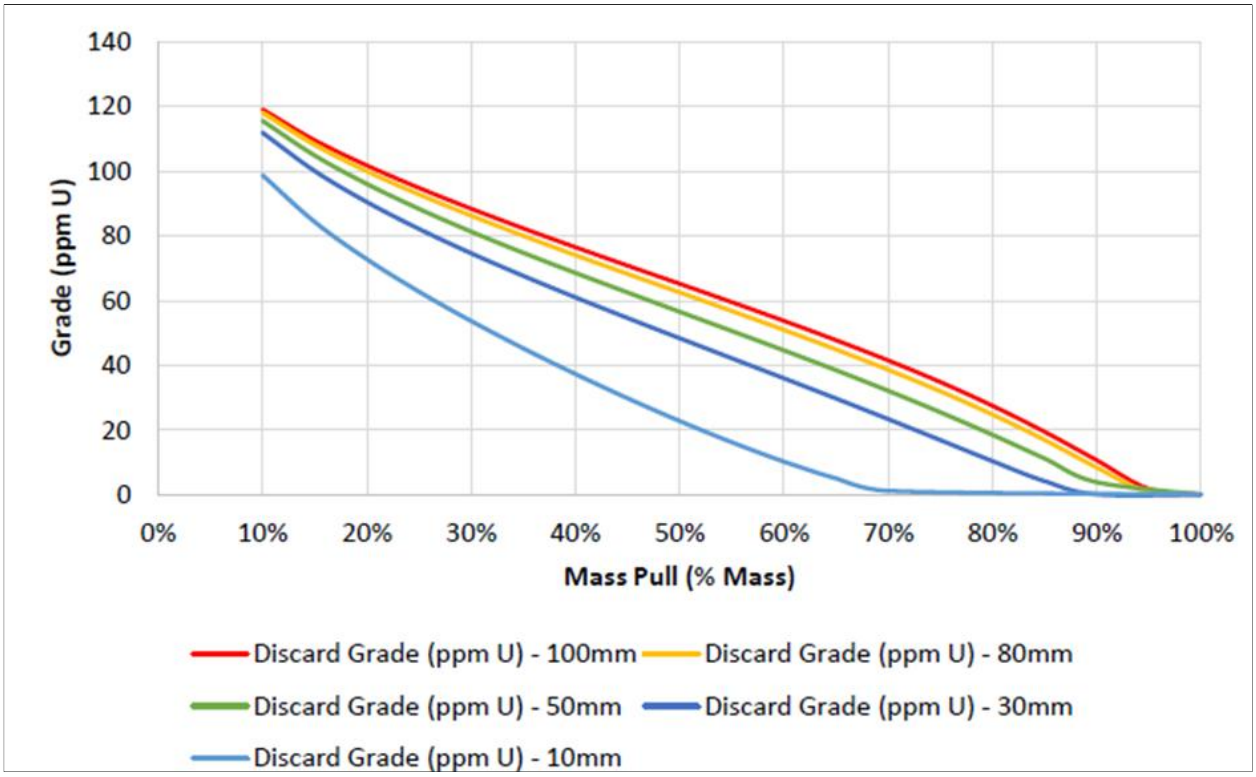

Figure 13-13: Simulated Rados XRF particle sorter uranium recovery performance at 100 mm, 80 mm, 50 mm, 30 mm and 10 mm for Muntunga uranium ore as a function of mass pull to sorter concentrate (Source: Rados. 2024)

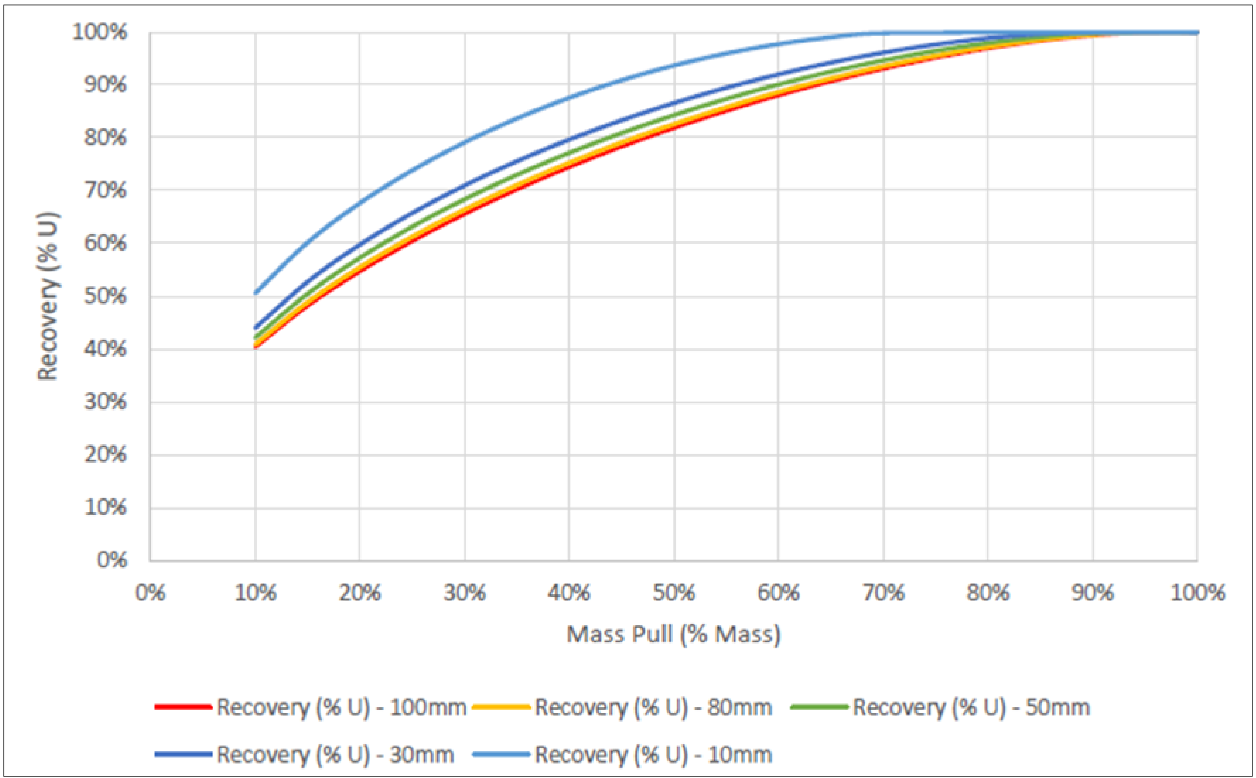

Figure 13-14: Simulated Rados XRF particle sorter uranium recovery performance at 100 mm, 80 mm, 50 mm, 30 mm and 10 mm for Muntunga uranium ore as a function of mass pull to sorter concentrate (source: Rados. 2024)

13.8.4.Bulk ore sorting results

For bulk ore sorting ("BOS"), the sorting packet is significantly larger than for particle sorting. A calculation must be made based on grade differences between larger portions of the ore body, using data from the DCA. However, BOS did not work as well with Muntanga ore with 80 % mass pull with little rejection and therefore particle sorting was selected.

13.9. Summary

The key findings of the study were:

- A total of 722 trays of drill core, covering 3 012 m from 77 drill holes was analysed
- Calibration against client assays showed a strong correlation (R^2 = 0.99 for uranium), confirming the accuracy of Rados XRF+ technology
- Both particle sorting and bulk sorting show that a 95 % Uranium recovery could be achieved while depleting the discard grade to below 50 ppm Uranium
- To achieve 95 % Uranium recovery, particle sorting shows a mass pull to concentrate of 50 % at 10 mm, 65 % at 30 mm, 70 % at 50 mm, 70 % at 80 mm and 75 % at 100 mm average particle sizes
- For design purposes, 95 % uranium recovery at 50 mm will result in a mass pull of 70 %
- The test work demonstrates the waste rejection and significant uranium upgrade potential for Rados XRF and ore sorting technology for the project.

13.10.Curing acid optimisation testwork

13.10.1. Method

Soaking tests were performed on crushed head samples as ~3kg agglomerated material developed by blending on a plastic sheet.

Approximately 50 % of the moisture was added as a synthetic solution, comprising tap water with 5 g/L Fe adjusted to 450 mV with 50 % H_2O_2 and pH 1.5 with 98 % H_2SO_4. Composite 1, however, was agglomerated with Rand Water Board tap water adjusted to pH 1.5 with 98 % H_2SO_4 only. The curing acid was added as 98 % analar grade sulfuric acid.

The remaining moisture was added until agglomeration was complete as per visual inspection. The samples were allowed to cure in a plastic bucket (closed but not sealed) for 24 hours. The samples were repulped twice with tap water adjusted to pH2 at a liquid: solid ratio of 1:1.

The first wash filtrate was analysed for U by ICP-OES and for pH, redox potential and sulfuric acid by titration. The residues were dried, crushed, pulverised and analysed for U_3O_8 by XRF15. Uranium mass balances were performed and the acid consumptions were calculated (Table 13-5).

The test matrix is shown below.

Table 13-5: Matrix of curing acid conditions

Test	Curing acid dosages [kg/t]	Agglomeration solution	Agglomeration moisture including acid [
Composite 1	5, 10, 23	Tap water, adjusted to pH 1.5	7.9 to 9.3
Composite 2	25, 40, 57	Tap water containing 5g/L Fe, adjusted to pH 1.5 and 450mV w.r.t Ag/AgCl	9.6 to 10.6
Composite 3	2, 10, 30		7.9 to 9.2
Composite 4	1, 5, 9		8.4 to 9.6
Composite 5	25, 40, 57		9.9 to 11.8

13.10.2. Results

The curing acid optimisation test results are summarised in Figure 13-15, Figure 13-16 and Table 13-6. These results were used to specify the curing acid additions in the tall leach columns:

1. **Composite 1:** Curing acid additions were 5 kg/t, 10 kg/t and 23 kg/t, respectively. The acid consumed over this range was between 1.6 kg/t to 5 kg/t and it appeared that the uranium dissolution had already peaked at a curing acid addition of 10 kg/t. Hence, a curing acid addition of 5 kg/t (the maximum acid consumption over the range) was specified for the column tests
2. **Composite 2:** Curing acid additions were 25 kg/t, 40 kg/t and 57 kg/t, respectively. Acid consumptions over this range were between 17.6 kg/t to 23.1 kg/t and it appears that the uranium dissolutions had already started to flatten off. Hence it appears that any acid above 20 kg/t was in excess, and a curing acid addition slightly below this value, namely 18 kg/t, was specified
3. **Composite 3:** Uranium dissolutions already appeared to have flattened off at the lowest curing acid addition of 2 kg/t, hence this value was specified for the columns
4. **Composite 4:** Uranium dissolutions increased with curing acid addition and was still increasing at the maximum curing acid addition of 9 kg/t tested, with a corresponding acid consumption of 8.3 kg/t. Hence a curing acid addition of 9 kg/t was specified for the columns, as lower acid additions were not yet optimal with respect to the initial uranium dissolution
5. **Composite 5:** Curing acid additions were 25 kg/t, 40 kg/t and 57 kg/t, respectively. The uranium dissolution profiles were very flat over this range, indicating that the optimal curing acid addition was below 25 kg/t. Acid consumption was between 11 kg/t and 16.9 kg/t. Hence an average curing acid addition of 14 kg/t was specified.

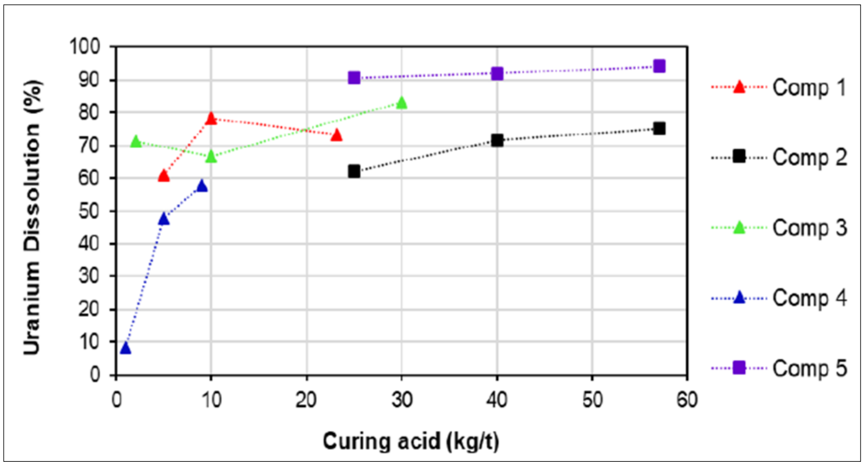

Figure 13-15: Uranium dissolution profiles

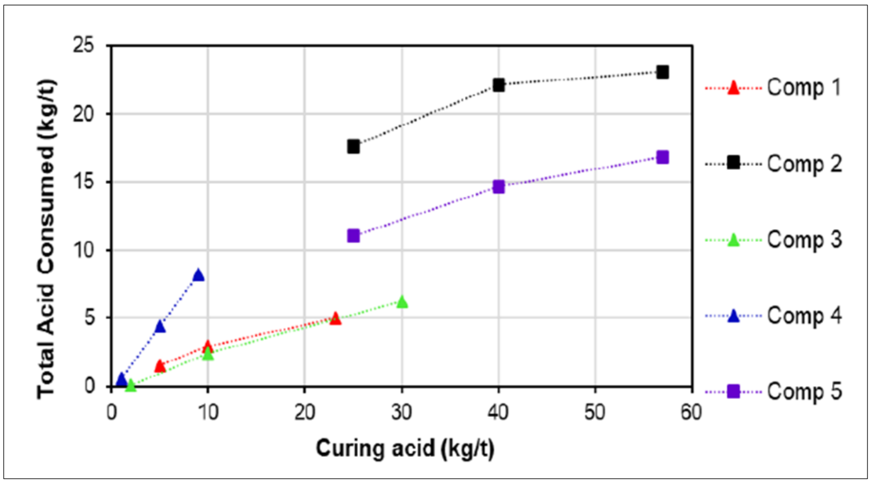

Figure 13-16: Total acid consumption profiles

Table 13-6: Summary of curing acid test results

Sample	Curing acid [kg/t]	U dissolution [%]	Acid consumed [kg/t]	Residual acid [kg/t]	Curing acid specified for columns [kg/t]
Composite 1	5	61	1.6	3.4	5
	10	78	2.9	7.1	
	23	73	5.0	18.2	
Composite 2	25	62	17.6	7.4	18
	40	72	22.1	17.9	
	57	75	23.1	33.9	
Composite 3	2	71	0.1	1.9	2
	10	67	2.4	7.6	
	30	83	6.2	23.8	
Composite 4	1	8	0.6	0.4	9
	5	48	4.4	0.6	
	9	58	8.3	0.7	
Composite 5	25	91	11.0	14.0	14
	40	92	14.7	25.3	
	57	94	16.9	40.1	

13.11. Iso-pH test

13.11.1. Methodology

Iso-pH tests were carried out on the pulverised head sample to determine the total acid consumption ("TAC") at constant pH. The iso-pH test apparatus is shown in Figure 13-7. The test procedure was as follows:

- The head sample (100 g) was placed in a 250 ml glass beaker
- 200 ml of de-ionised water, acidified with H_2SO_4 to pH 1.5, 1.8 or 2.0, was added
- The sample was stirred on a magnetic stirrer plate
- A pH sensor connected to an auto-titrator was inserted into the glass beaker and the pH was adjusted to the set point with 3M sulfuric acid solution
- The pH was maintained at the set-point for four hours
- Acid additions and pHs were logged at regular intervals
- The final slurry mass was measured and the solids were filtered and dried
- The final filtrate was analysed by XRF for U_3O_8
- The acid addition and filtrate assays were used to calculate the TAC.

Figure 13-17: Setup, iso-pH test

13.11.2. Results

The TAC after four hours at each pH (1.5; 1.8 and 2) are summarised in Figure 13-18 and Table 13-7. Composites 2 and 4 had the highest acid consumption (39.1 kg/t and 34.5 kg/t) at pH 1.5. Maximum acid consumptions for Composites 1, 3 and 5 were between 15.6 kg/t and 17.3 kg/t at pH 1.5. The curing acid additions specified for the column tests are generally lower than the TAC determined in the iso-pH tests. This is because the iso-pH tests were performed on milled material, whereas the curing acid tests were performed on coarse (-25 mm) material with a lower surface area.

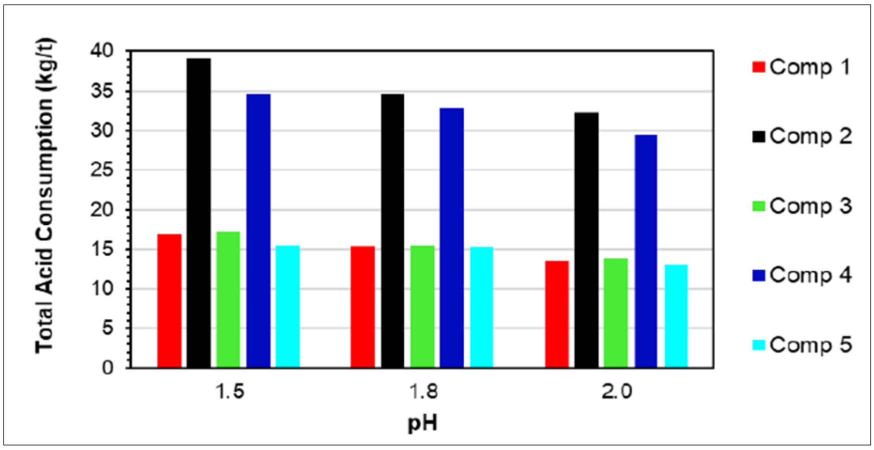

Figure 13-18: TAC profiles

Table 13-7: Results of acid curing

Total acid consumption [kg/t]					
pH	Composite 1	Composite 2	Composite 3	Composite 4	Composite 5
1.5	16.8	39.1	15.5	32.9	15.3
1.8	15.4	34.6	15.5	32.9	15.3
2.0	13.6	32.3	13.9	29.4	13.1
Curing acid specified for columns	5.0	18	2.0	9.0	14

13.12.Stacking tests

Uni-axial compression (stacking tests) was performed in a 150 mm ID steel pot (31.6 cm tall). The agglomerates were loaded into the pot while hammering on the sides to compact the bed. Since most of the slumping in heaps and leach columns typically occurs after initial pre-wetting, the agglomerates were first pre-wetted by irrigation with a synthetic solution for 24 hours at 6 L/m^2/h, followed by draining for another 24 hours. The agglomerates were then subjected to an incremental mechanical load (simulating overburden weight) to measure the change in bulk density and porosity as a function of stacking height (Figure 13-9). After compression to the maximum load, the bed was saturated, and the saturated hydraulic conductivity was measured by passing synthetic leach solution through the bed from a constant head reservoir. Finally, the bed was allowed to drain, and the drain-down profile was generated. The residual moisture content was determined by drying the solids. As indicated in Table 13-8, experience has shown that the saturated hydraulic conductivity should be at least 100 times the target.

Table 13-8: Target physical and hydrological criteria

Parameter	ST	Source
Bulk density	<1.9t/m^3	Robertson et al., 2013
Porosity	>30 % >0.3m^3/m^3	Robertson et al., 2013 Guzman et al., 2013
Saturated hydraulic conductivity (Ks)	>1 000 L/m^2/h	Robertson et al., 2013 Milczarek et al., 2013
Air conductivity (Ka)	>36L/m^2/h	Guzman et al., 2013
Mobile: Immobile moisture	50: 50	Robertson et al., 2013
PSD criteria	<10 % to 15 % - 75µm	Garcia and Jorgensen, 1997
PSD criteria	<10 % to 14 % -150µm <40 % -5mm	Scheffel, 2017

Figure 13-19: Stacking head test

13.12.1. Results

The stacking (uni-axial compression) test results are presented in Figure 13-20 to Figure 13-24 and Table 13-9. The feed composites were compressed under mechanical loads equivalent to 6 m and 50 m stacking heights, respectively. Insufficient sample was available for Composite 4, hence only a 6 m compression was performed. The residues were removed wet from the leach columns and loaded into the compression cells while hammering on the sides of the cell to compact the samples. Both head and residue samples were pre-wetted by irrigation for 24 hours at 6 L/m^2/h with a synthetic solution and drained for a further 24 hours before compression.

After compression to a 6 m equivalent stacking height, the final bulk densities of the head samples were between 1.55 t/m^3 and 1.62 t/m^3 and the residues between 1.58 t/m^3 and 1.63 t/m^3, which is well below the target of 1.9 t/m^3. The samples compressed to a 50 m equivalent stacking height had final bulk densities of between 1.72 t/m^3 and 1.79 t/m^3. Experience has shown that for the sample to be suitable for percolation leaching, the total bed porosity after compression should remain above 30 %. In all the tests, the final porosity was above this target (33 % to 42 %). Bulk densities in the leach columns were between 1.46 t/m^3 and 1.49 t/m^3 with final bulk densities after leaching between 1.36 t/m^3 and 1.4 t/m^3.

Experience has shown that for the sample to be suitable for percolation leaching, the saturated hydraulic conductivity (Ks) should be at least 100 times the target application rate. Hence for a target irrigation rate of 10 L/m^2/h, the saturated hydraulic conductivity should be >1 000 L/m^2/h. The head samples after compression to a 6 m equivalent load measured saturated hydraulic conductivities >1 000 L/m^2/h. The residues measured between 84 L/m^2/h and 726 L/m^2/h. This is due to the increase in fines during the leach. The head samples which were compressed to 50 m did not conduct solution during the saturated hydraulic conductivity measurements.

During drain-down, 26 % to 40 % of the saturation moisture drained out from the composite samples tested at 6 m stacking height, indicative of moderate to high fine content. The residual moisture content typically increases with sand content, as an increase in sand translates into an increase in surface area and capillary suction. A ratio of 50:50 is optimal to provide a ratio between drained and residual moisture. The samples tested at 50 m equivalent compression could not conduct solution during the saturation test and hence were not drained.

The stacking tests therefore indicate that all head samples are amenable to heap leaching at the lower lifts of 6 m, but not at a taller lift of 50 m. However, due to fines increasing during leaching, the residues are marginal, although no permeability restraint was observed in the leach columns. The leach columns, however, had lower bulk densities than the compressed samples in the compression tests.

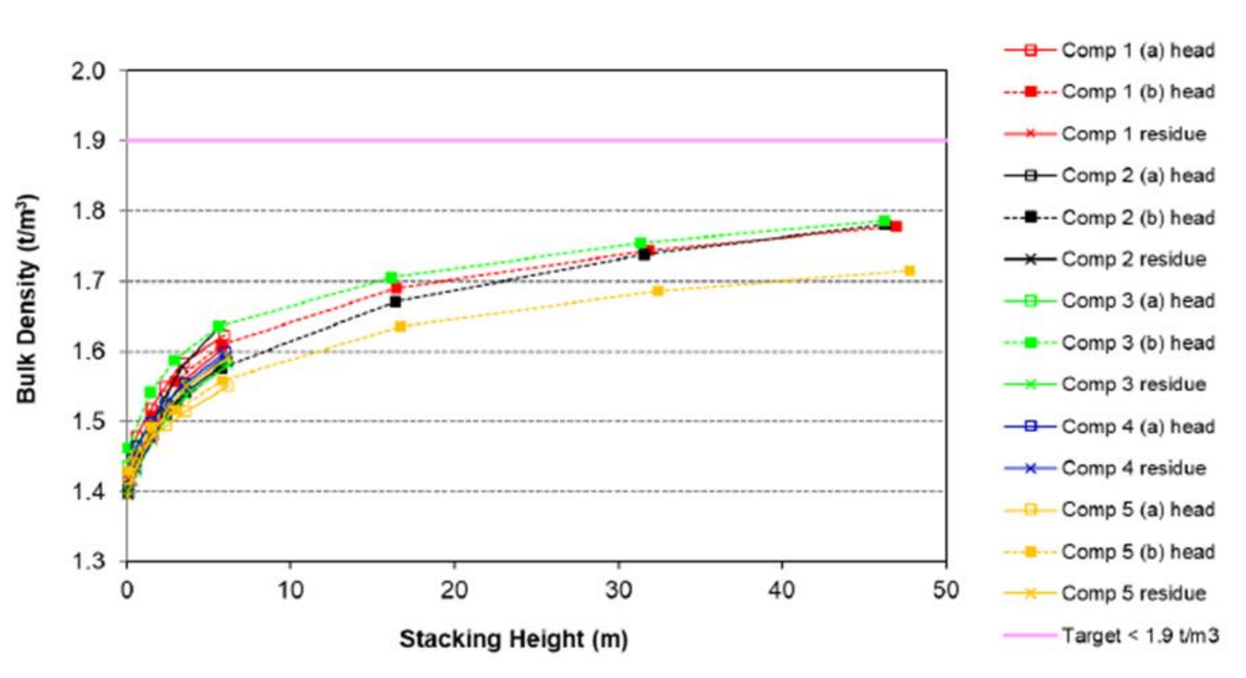

Figure 13-20: Bulk density profiles

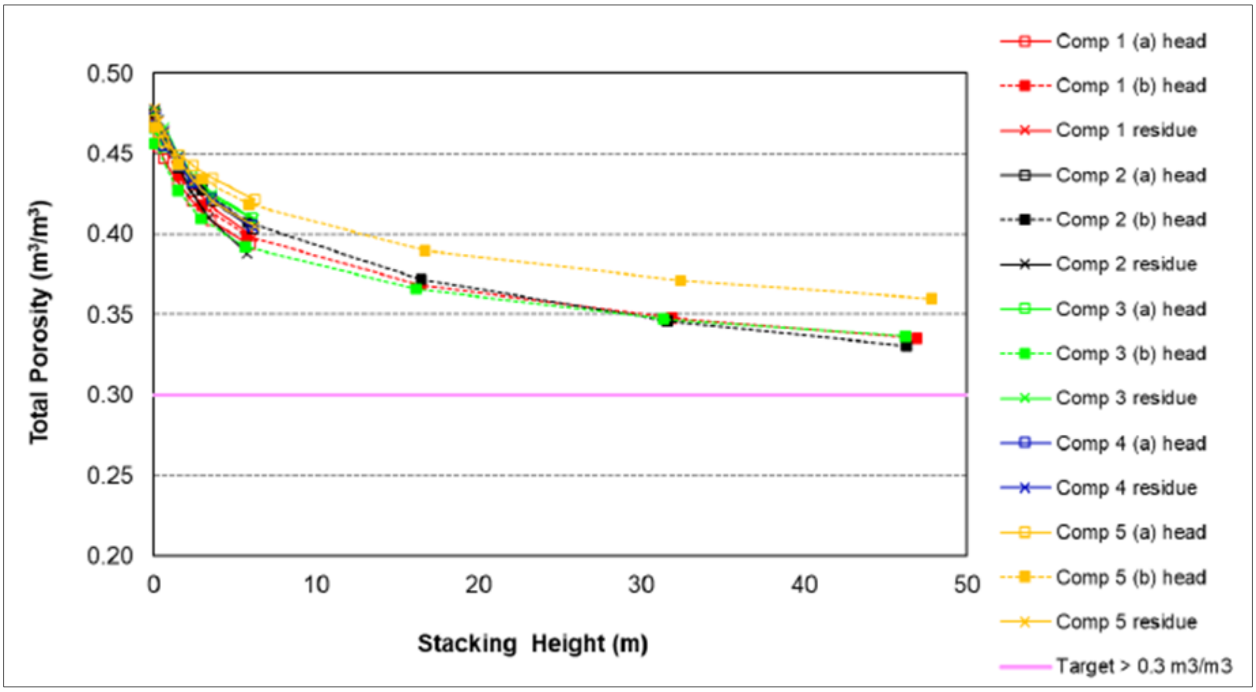

Figure 13-21: Porosity profiles

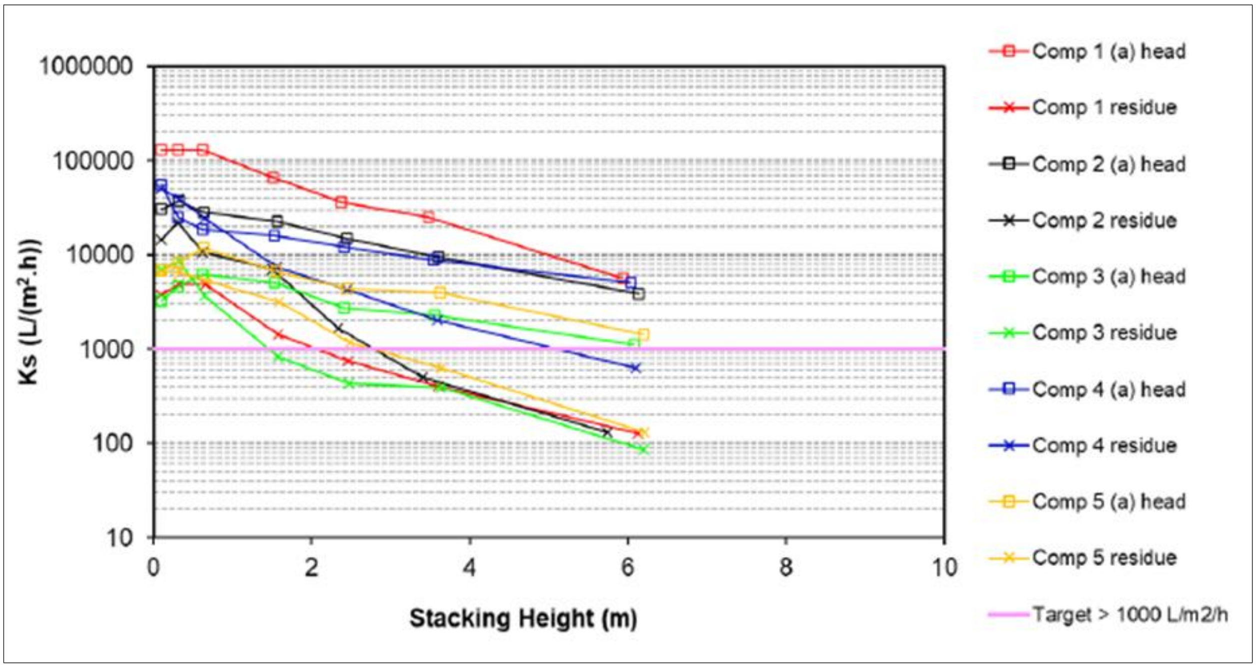

Figure 13-22: Saturated hydrologic profiles

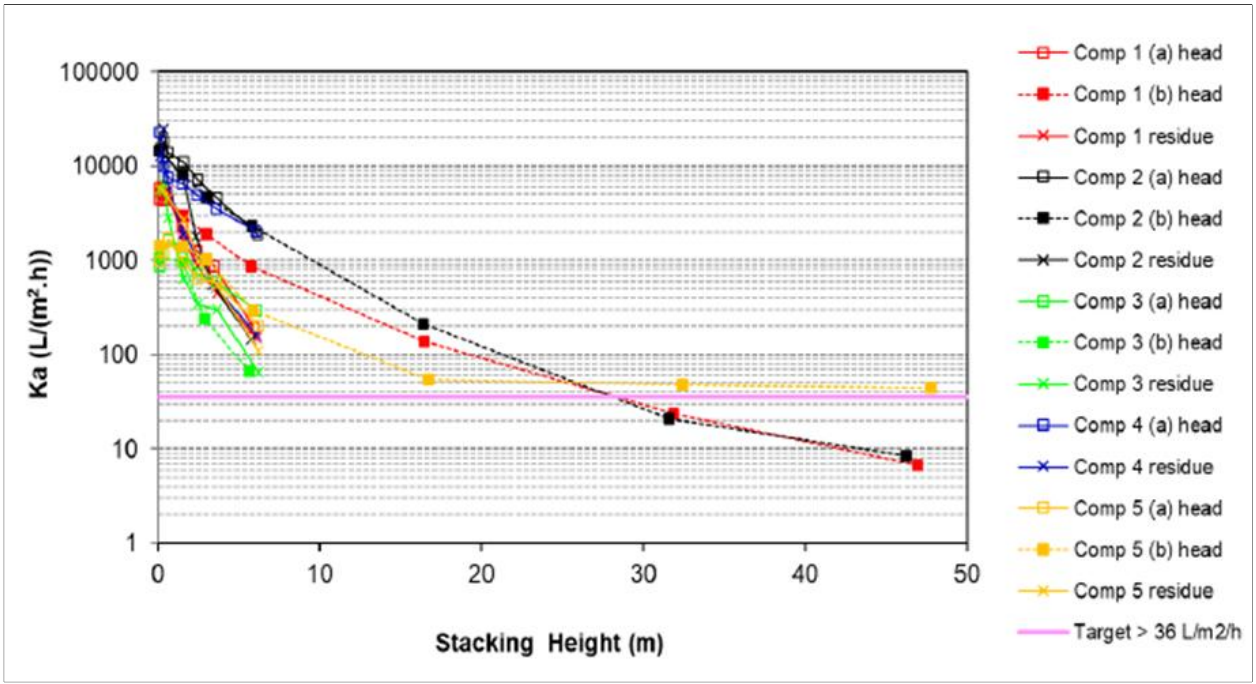

Figure 13-23: Air conductivity profiles

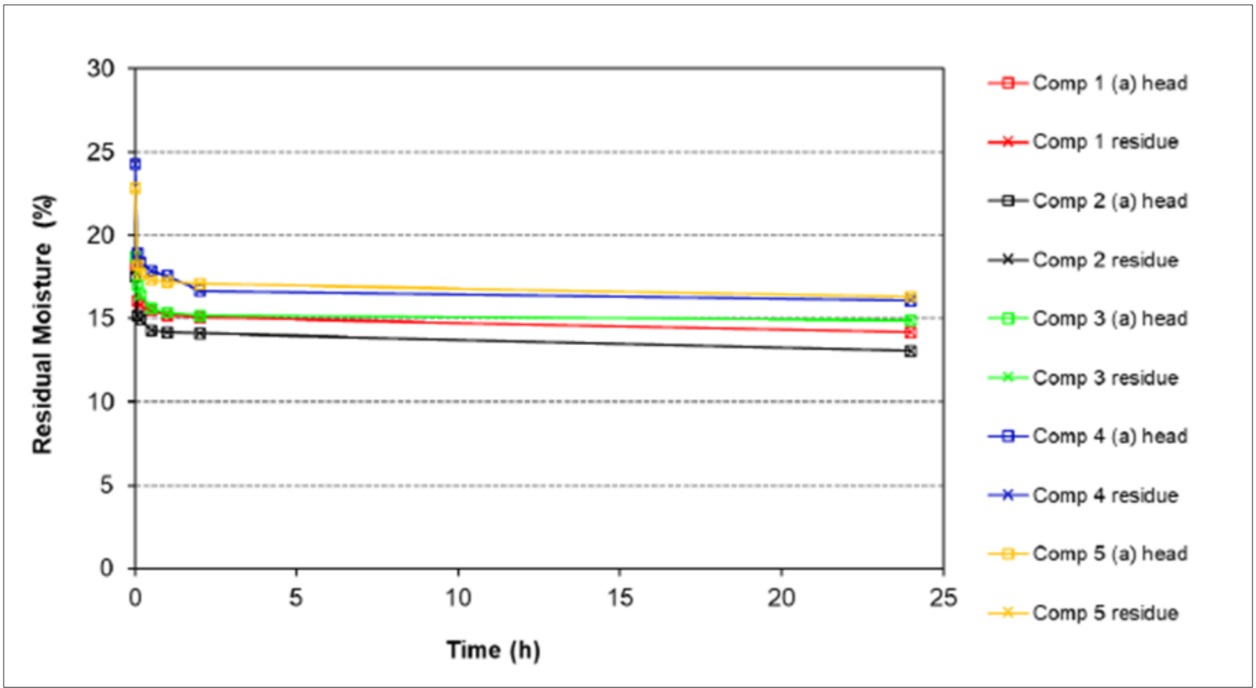

Figure 13-24: Residual moisture profile

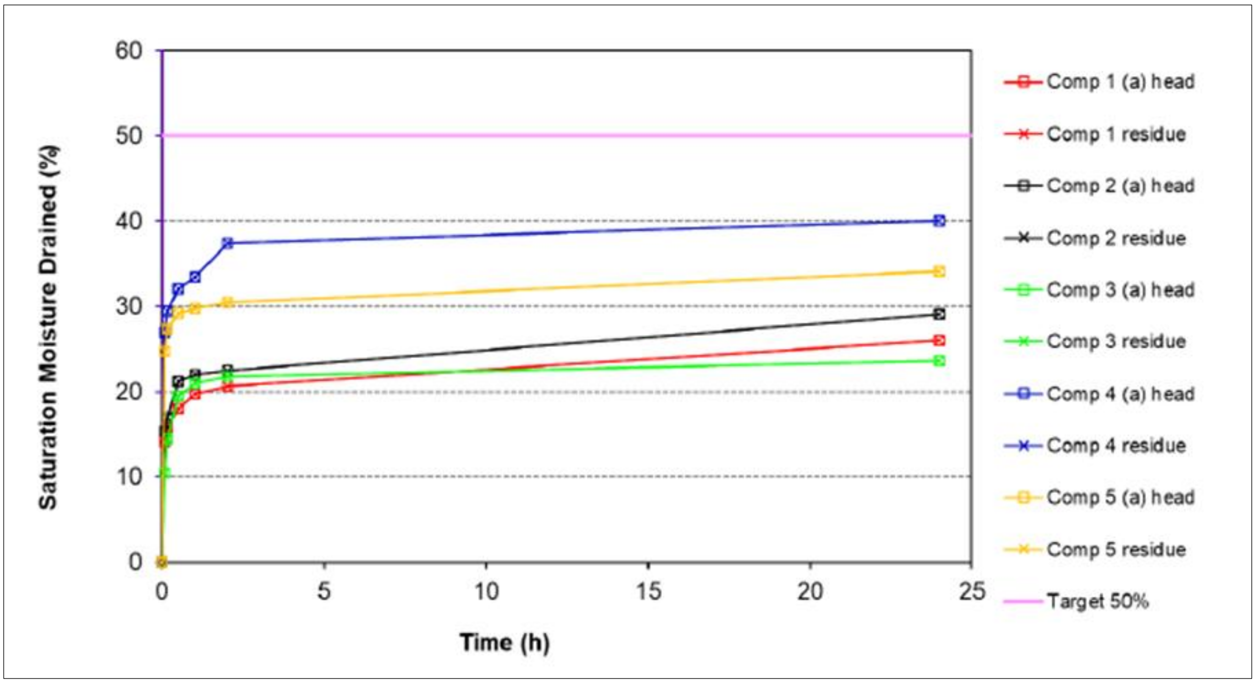

Figure 13-25: Drain down profile

Table 13-9: Stacking test results

Test	Unit	Comp 1 ST	Comp 2 ST	Comp 3 ST	Comp 4 ST	Comp 5 ST	Comp 1 STb	Comp 2 STb	Comp 3 STb	Comp 5 STb	Comp 1 residue	Comp 2 residue	Comp 3 residue	Comp 4 residue	Comp 5 residue
Target height	m	6	6	6	6	6	50	50	50	50	6	6	6	6	6
Topsize	mm	25	25	25	25	25	25	25	25	25	25	25	25	25	25
Curing acid	Kg/t	5	18	2	9	11	5	18	2	11	-	-	-	-	-
Agglomeration moist	%	7.8	7.1	7.5	9.1	7.6	7.7	7.1	7.5	7.6	-	-	-	-	-
Solids SG		2.68	2.66	2.69	2.69	2.68	2.68	2.66	2.69	2.68	2.68	2.66	2.69	2.69	2.68
Initial dry bulk density	t/m^3	1.42	1.41	1.44	1.42	1.42	1.43	1.40	1.46	1.43	1.40	1.40	1.40	1.40	1.40
Final dry bulk density	t/m^3	1.62	1.59	1.59	1.60	1.55	1.78	1.78	1.79	1.72	1.61	1.63	1.58	1.59	1.60
Initial porosity	m^3/m^3	0.47	0.47	0.47	0.47	0.47	0.47	0.47	0.46	0.47	0.48	0.48	0.48	0.48	0.48
Final porosity	m^3/m^3	0.39	0.40	0.41	0.41	0.42	0.34	0.33	0.34	0.36	0.40	0.39	0.41	0.41	0.40
Moisture at saturation	%	18.2	17.5	18.7	24.3	22.8	-	-	-	-	18.2	18.5	18.1	19.6	18.5
% of saturation moisture drained	%	26.0	29.1	23.6	40.1	34.1	-	-	-	-	26.0	27.3	25.4	32.1	27.3
Final cake moisture	%	14.2	13.1	14.9	16.1	16.3	-	-	-	-	14.2	14.2	14.2	14.2	14.2
Sat'd hydraulic cond.	$L/m^2/h$	5 555	3 886	1 116	5 023	1 415	-	-	-	-	128	132	84	627	130

13.13. Hydrodynamic column tests

13.13.1. Methodology

The hydrodynamic column test was performed by loading column leach residues into a 150 mm ID column (76 cm tall) at a uniform target bulk density. After curing for seven days in the column, the bed was irrigated with tap water (at incremental rates until ponding occurred. The moisture hold-up and degree of saturation profiles as a function of irrigation flowrate were calculated from the solution balance. The air permeability was measured at each flowrate by passing air through the bed and measuring the pressure drop. Finally, a drain-down profile was generated. The ratio between drained and residual moisture was measured. The residual moisture content was determined by drying the solids.

Figure 13-26: Hydrodynamic column tests

13.13.2. Results

The hydrodynamic column test results are presented in Figure 13-27 to Figure 13-30. The column leach residues were loaded at bulk densities equivalent to a 6m stacking height, based on the ST profiles. Therefore, the HCT tests were more conservative than the leach columns, as they represent a more compacted condition at the bottom of the heap (Table 13-10). However, the leach columns were compacted by hammering on the side of the columns during loading, but the compression tests experienced further compaction on account of the application of a mechanical load.

The degree of saturation versus application rate is plotted in Figure 13-27 and indicates that all residue samples operated below the limit of geotechnical stability (85 % saturation) at application rates of up to 20 L/m^2/h. Especially Composite 1 residue and Composite 2 residue operated close to saturation. The ratio between drained and residual moisture is shown in Table 13-10 which is indicative of a low to medium ratio between mobile and immobile moisture. A low ratio is indicative of higher fine content, as fines have a larger surface area and increased capillary suction.

Table 13-10: Drain down results

Sample	Ration drained: residual moisture	HCT loaded dry bulk density [g/cm^3]	Column leach dry loaded bulk density [g/cm^3]
Composite 1 residue	37:63	1.61	1.46
Composite 2 residue	33:67	1.64	1.49
Composite 3 residue	34:66	1.58	1.46
Composite 4 residue	35:65	1.59	1.49
Composite 5 residue	20:80	1.59	1.49

The moisture content increased to between 18.4 % and 24.3 % at saturation. The HCT results therefore confirm that a 6 m stacking height can be accommodated. However, the presence of high fine contents due to precipitation and decrepitation in residues have resulted in operation closer to saturation, potentially reducing the permeability over time. The HCT tests represent a “worst-case scenario” of high fine content (due to decrepitation in the residues) and compression at the bottom of the heap. Hence the results are probably conservative compared with the leach columns, where permeability was good.

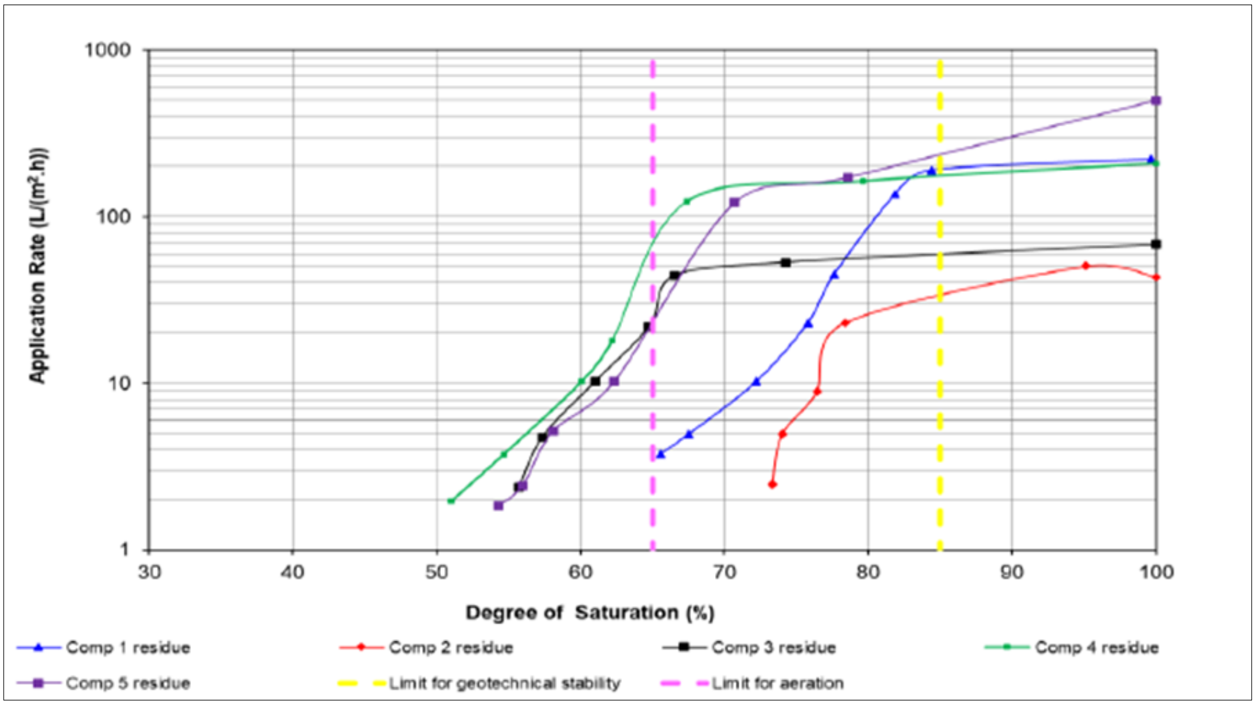

Figure 13-27: Application rate versus degree of application

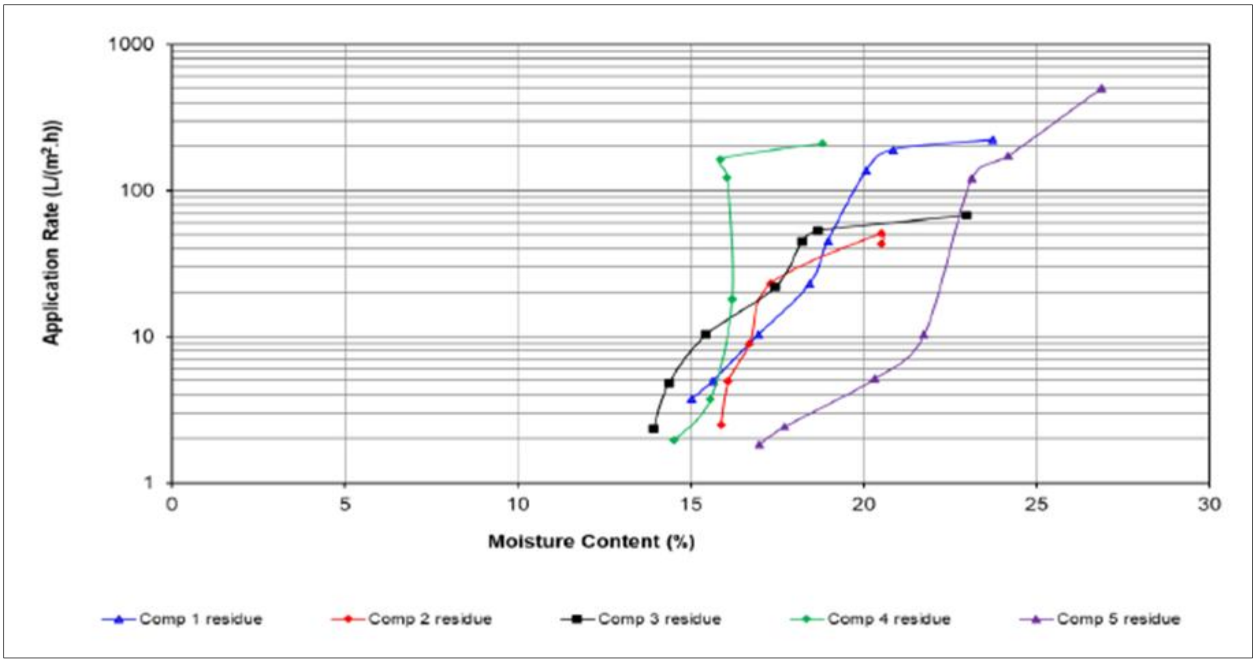

Figure 13-28: Application rate versus moisture content

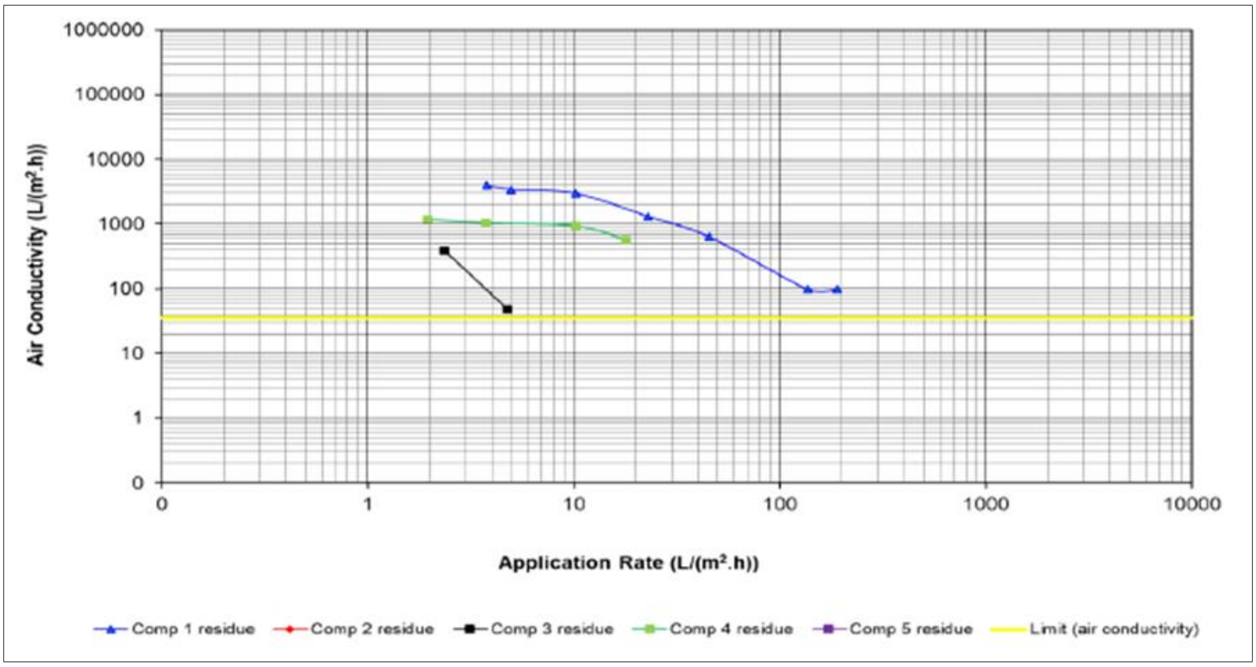

Figure 13-29: Air conductivity versus application rate

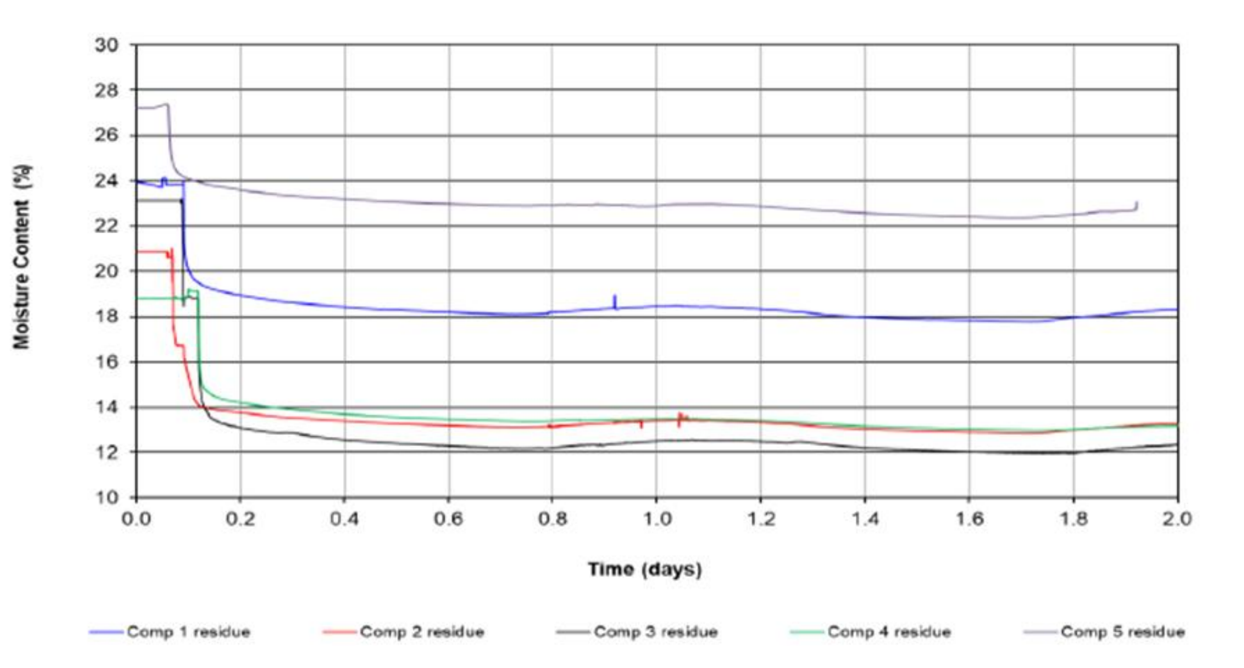

Figure 13-30: Draindown profiles

13.14. Column leach tests

13.14.1. Methodology

A series of tall (6 m) columns were run under the conditions summarised in Table 13-11.

Table 13-11: Summary conditions of the tall (6m) columns

Test	Crush size	Height	Temp.	Curing acid	Agglomeration moisture excl. acid	Irrigation rate	Lixiviant
	mm	[m]	[°C]	[kg/t]	[%]	[L/m²/h]	
Comp 1	-25	6	25	5	8.1	10	Tap water adjusted to pH 1.5
Comp 2	-25	6	25	18	8.4	10	Tap water, containing 5g/L Fe, adjusted to pH 1.5 and 450mV (vs. Ag/AgCl)
Comp 3	-25	6	25	2	7.0	10	
Comp 4	-25	6	25	9	9.8	10	
Comp 5	-25	6	25	11	8.6	10	

Tests were performed in 6 m tall, 160 mm ID, jacketed polypropylene columns at 25 ºC. The feed material was agglomerated in a rotating drum mixer while adding synthetic barren solution and curing acid (98 % sulfuric acid). For Composite 1 the agglomeration solution comprised tap water adjusted to pH 1.5. For Composites 2 to 5 the agglomeration solution comprised tap water containing 5 g/L Fe(II) adjusted to 450 mV (vs. Ag/AgCl) with 50 % hydrogen peroxide and pH 1.5 with sulfuric acid.

The agglomerates were loaded into the columns and allowed to cure for seven days. Irrigation was started with a synthetic solution at 5 L/m^2/h until the drainage rate was steady, after which the irrigation rate was increased to 10 L/m^2/h, the proposed operational irrigation rate.

The masses, pHs, redox potentials (Ag/AgCl) and specific gravity of the feed and drainage solutions were recorded daily. The feeds and daily PLSs were titrated for ferrous iron and sulfuric acid acidity. Drainage solutions were analysed daily for the first 10 days, and after that weekly for U_3O_8 by ICP-OES and Fe by multi-element ICP-OES (Al, As, Ca, Cd, Co, Cr, Cu, Fe, Li, Mg, Mn, Mo, Ni, Pb, S, Si, T, V and Zn).

The PLS were accumulated and after the first three weeks, the solutions were treated with batch ion exchange to remove uranium. The barren solution was readjusted to pH 1.5 and 450 mV (vs. Ag/AgCl) and recycled as feed over the columns. After 55 days (Composite 4) and 58 days (Composites 1, 2, 3 and 5) irrigation was stopped and the columns were allowed to drain for five days. A wash irrigation cycle was performed consisting of pH 2 tap water for five days, five days drainage, neutral unadjusted tap water for ten days, five days drainage, and finally unadjusted tap water for 15 days, followed by five days drainage.

The columns were discharged in 1m sections. Each section was divided into three portions. The sections were then split to be analysed as:

- A third portion was dried, crushed to -1.7 mm, pulverised and analysed for U_3O_8 (XRF15) and multi-element ICP-OES on the solid residue
- One-third portion was stored wet for geomechanical and geotechnical tests
- The other portion was combined with the other 1m sections to generate a single composite for each column. A wet PSD was performed on the composite, and assay-by-size was performed in four size classes (+1.7 mm, +3.35 mm, +1.18 mm, -1.18 mm) for U_3O_8 (XRF15) and multi-element ICP-OES.

A composite residue was reconstituted from the four size classes for geochemical assays: TCLP leach, with leach liquors analysed by ICP-MS (Ti, V, Cr, Mn, Co, Ni, Cu, Zn, As, Se, Sr, Zr, Mo, Ag, Cd, Sn, Sb, Ba, Hg and Pb), ICP-OES (Al, Ca, Fe, Mg, S and Si), Cl, F, PO_4 and U-MS. Solids were also analysed for W-AP (acid generation potential), W-NP (neutralising potential), W-ABA (acid-base accounting), sulfate S, sulfide S and total S.

The following drainage solutions were analysed for geochemical assays including ICP-OES (Al, As, Ca, Cd, Co, Cr, Cu, Fe, Li, Mg, Mn, Mo, Ni, Pb, S, SI, T, V and Zn), ICP-OES (U), ICP-MS (Ag, B, Ba, Cd, Hg, Sb, Se and Sn), and K, Na, Cl, F and PO_4:

- Week 1 to 3 accumulated PLS
- Week 4 accumulated PLS (Composite 4) and weeks 4, and 5 accumulated PLS (Composites 1, 2, 3, 5)
- Weeks 5, 6, and 7 accumulated PLS (Composite 4) and weeks 6, 7, and 8 accumulated PLS (Composites 1,2,3,5)
- Wash cycle 1 accumulated PLS
- Wash cycle 2 accumulated PLS
- Wash cycle 3 accumulated PLS.

13.14.2. Results

Column leach test results are provided below. Final uranium dissolutions (based on a recalculated head) were between 74.6 % (Composite 4) and 94.9 % (Composite 3). Uranium dissolution curves are presented in Figure 13-31, Figure 13-32 and Figure 13-33. Leaching was essentially complete after 20 days (Composites 1, 3 and 5) and after 60 days (Composites 2 and 4). Final acid consumptions were between 7 kg/t (Composite 1) and 27 kg/t (Composite 2).

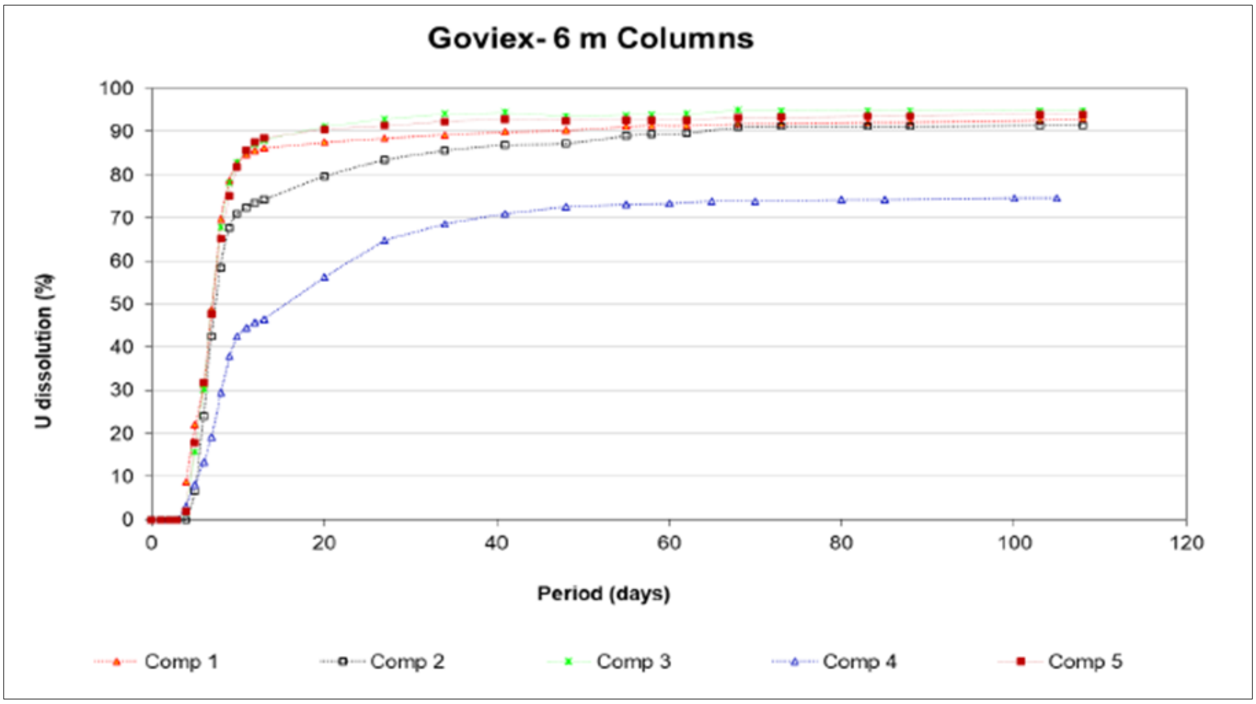

Figure 13-31: Uranium dissolution versus time

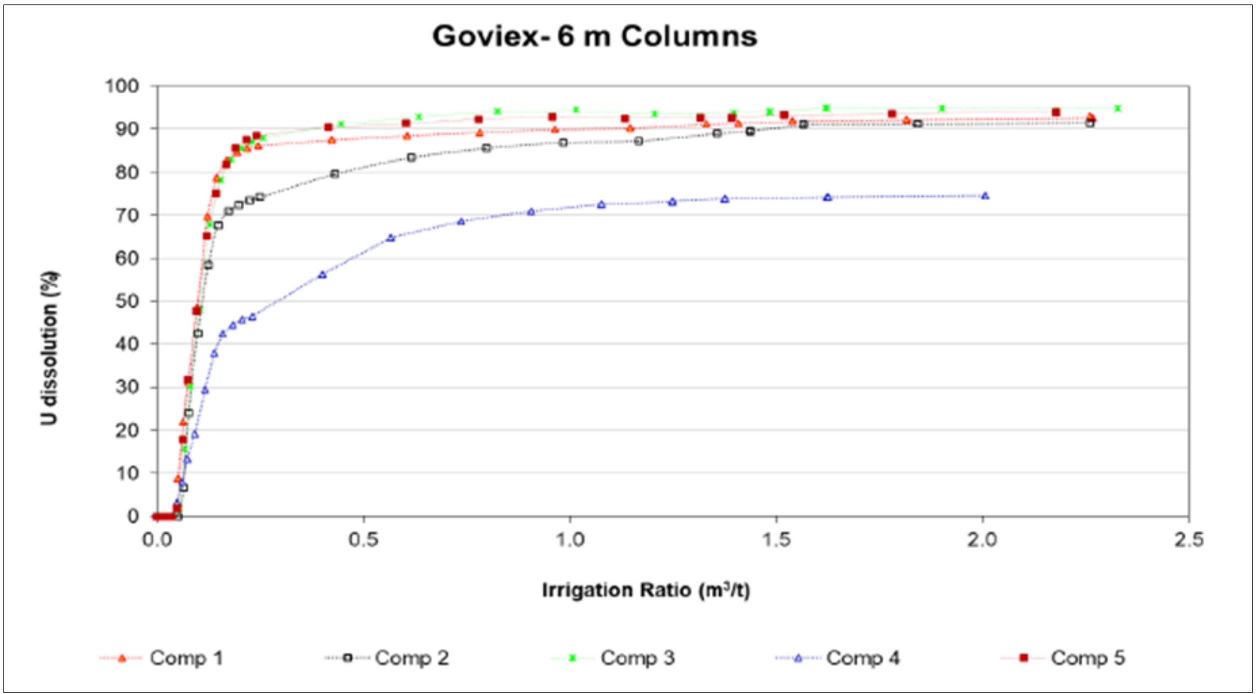

Figure 13-32: Uranium dissolution versus irrigation Ratio (recalculated head)

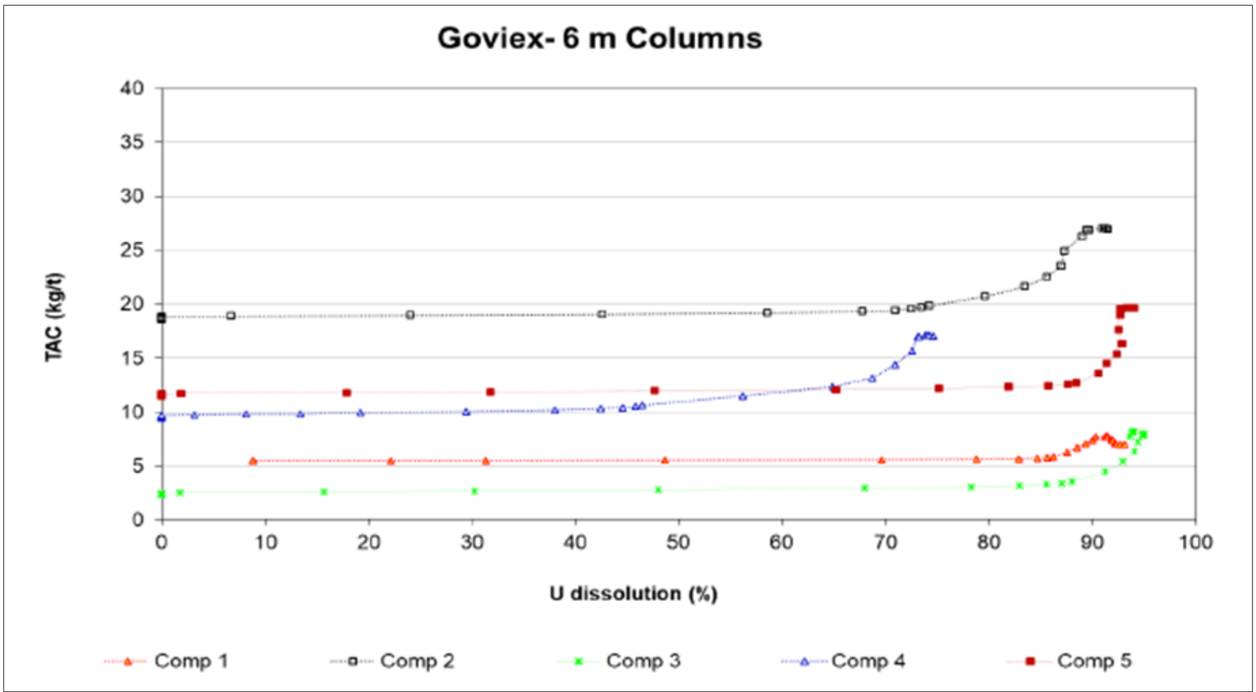

Figure 13-33: TAC versus uranium dissolution

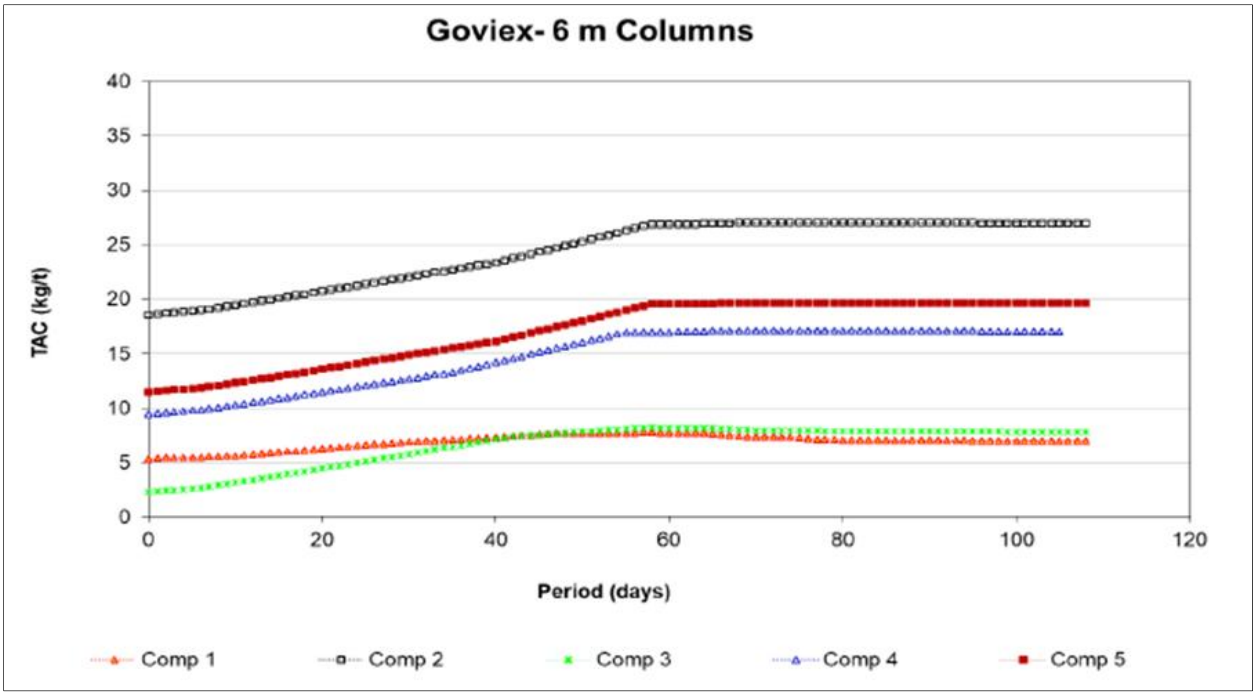

Figure 13-34: TAC versus time

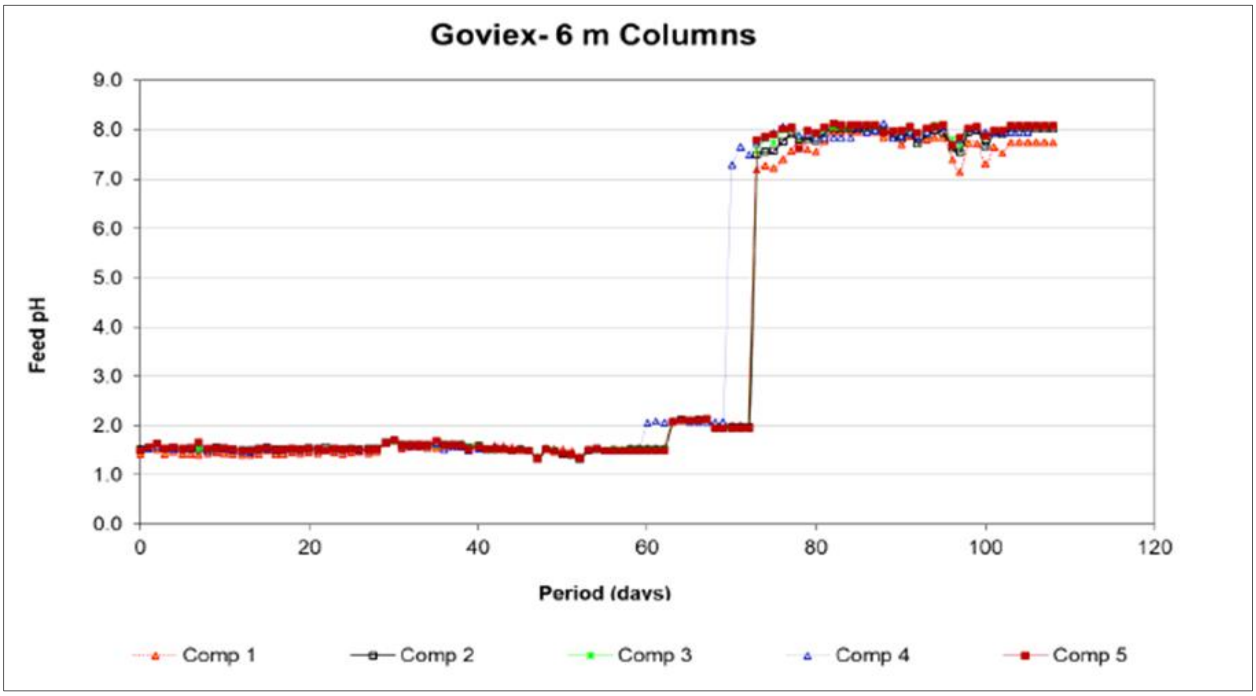

Figure 13-35: Feed pH profile

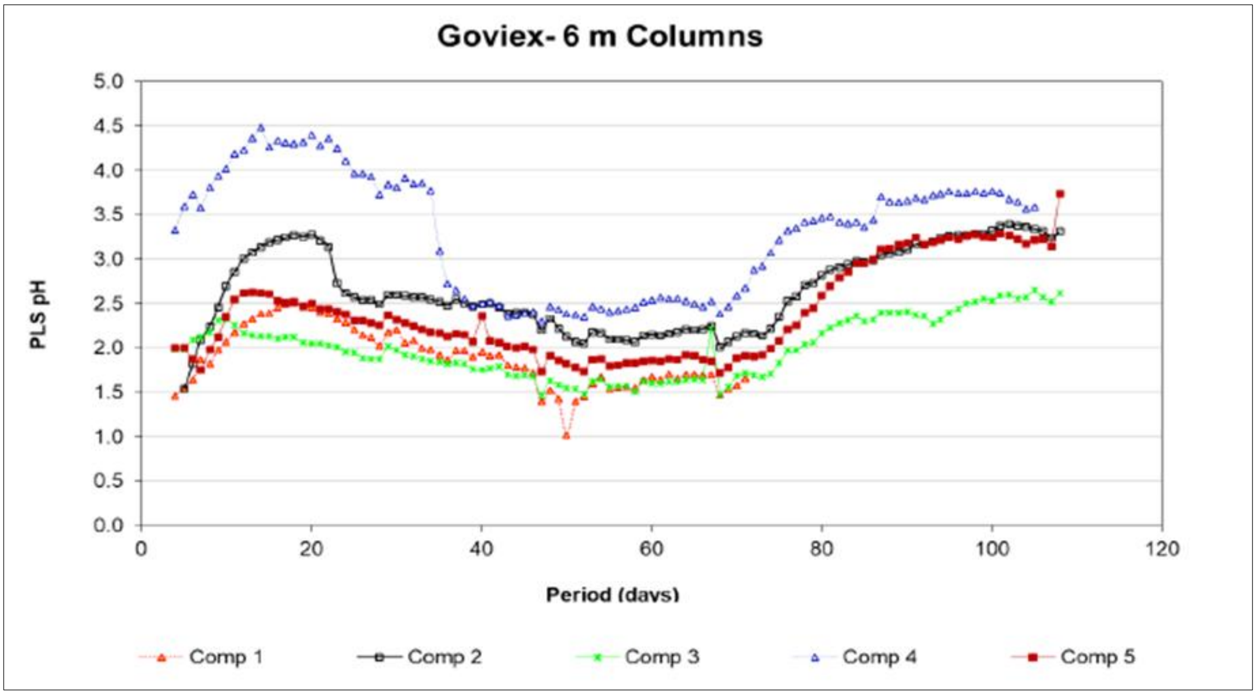

Figure 13-36: PLS pH profile

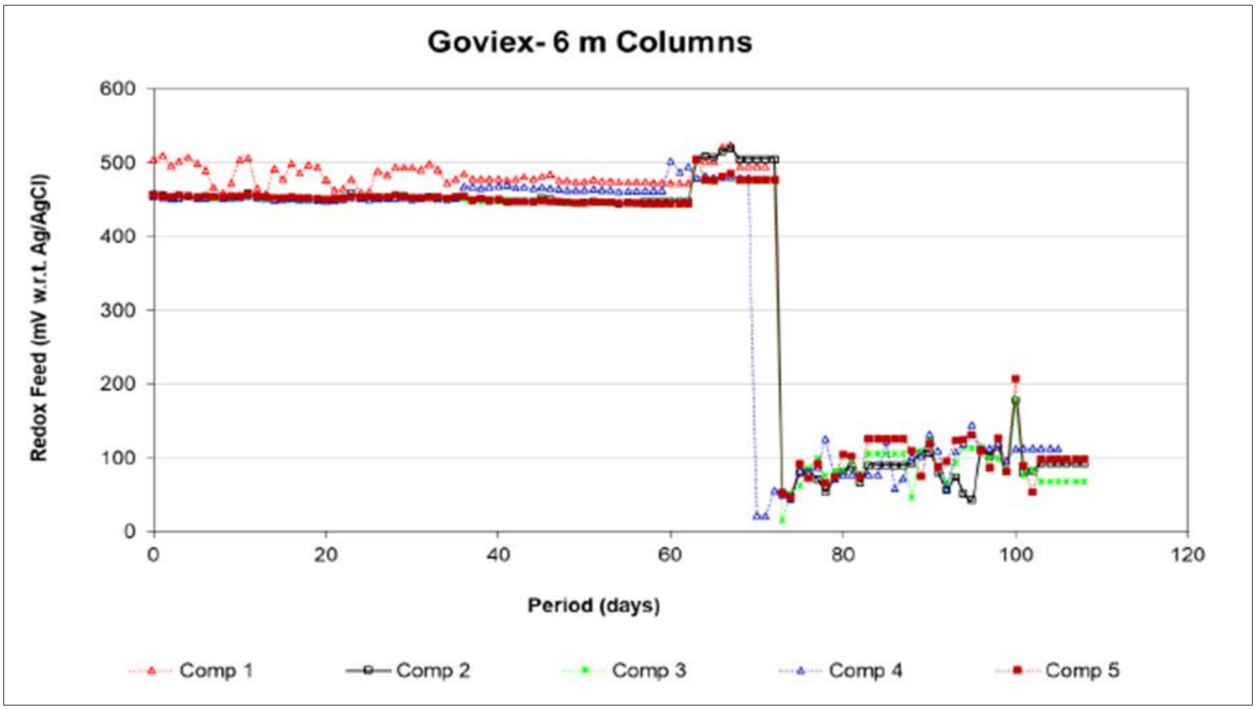

Figure 13-37: Feed Redox (Eh) potential over time

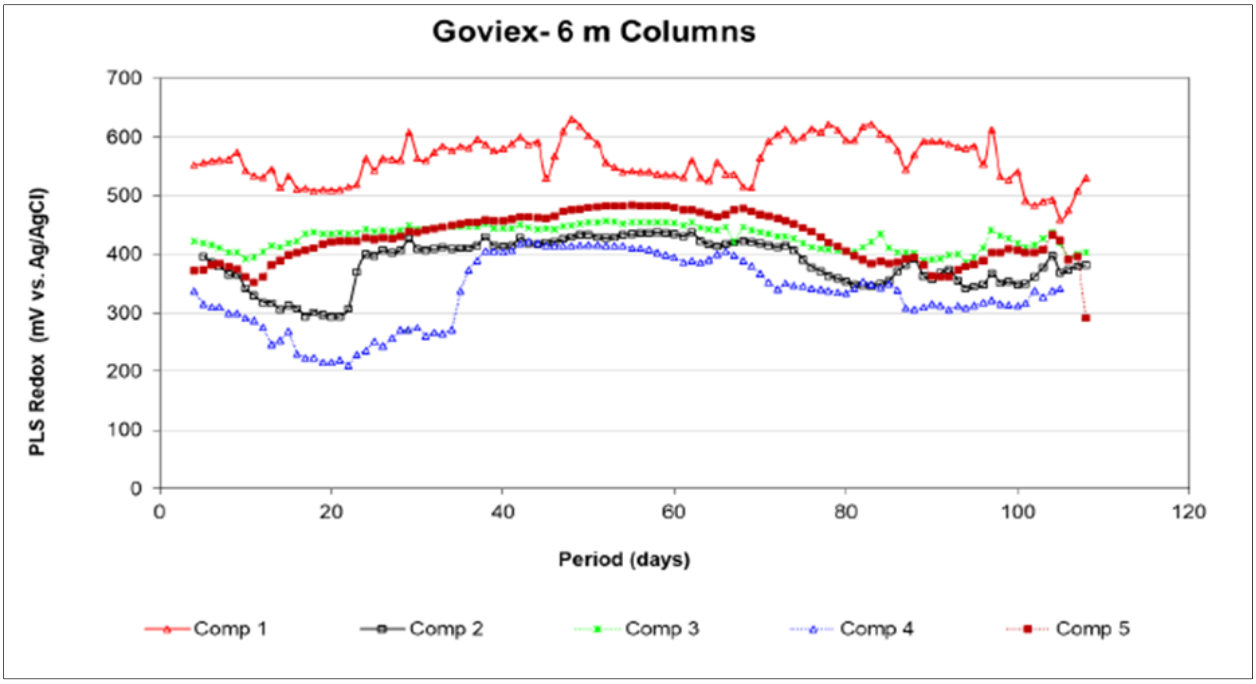

Figure 13-38: PLS Redox potential over time

During initial irrigation, the drainage pH showed an increase over the first 30 days, especially for Composite 4. This resulted in the precipitation of iron (III) from the solution over this period. As a result, redox potentials in the PLS dropped below 450 mV (wrt Ag/AgCl), especially for Composite 4, resulting in the PLS containing a lower ratio of ferric to ferrous iron than in the synthetic feed. PLS collected over the first three weeks was sent for bulk IX test work, and the barren solution was readjusted and recycled over the columns.

Columns showed minimal slumping (<2 %) although the bed may have received support from the column walls. No inhibition to percolation or ponding was observed throughout the leach tests. The low degree of compaction means that the columns maintained a high bed porosity and the bed typically was below 60 % void saturation (Appendix C, Figure C.5). Despite this, decrepitation occurred during the leach, resulting in fines generation.

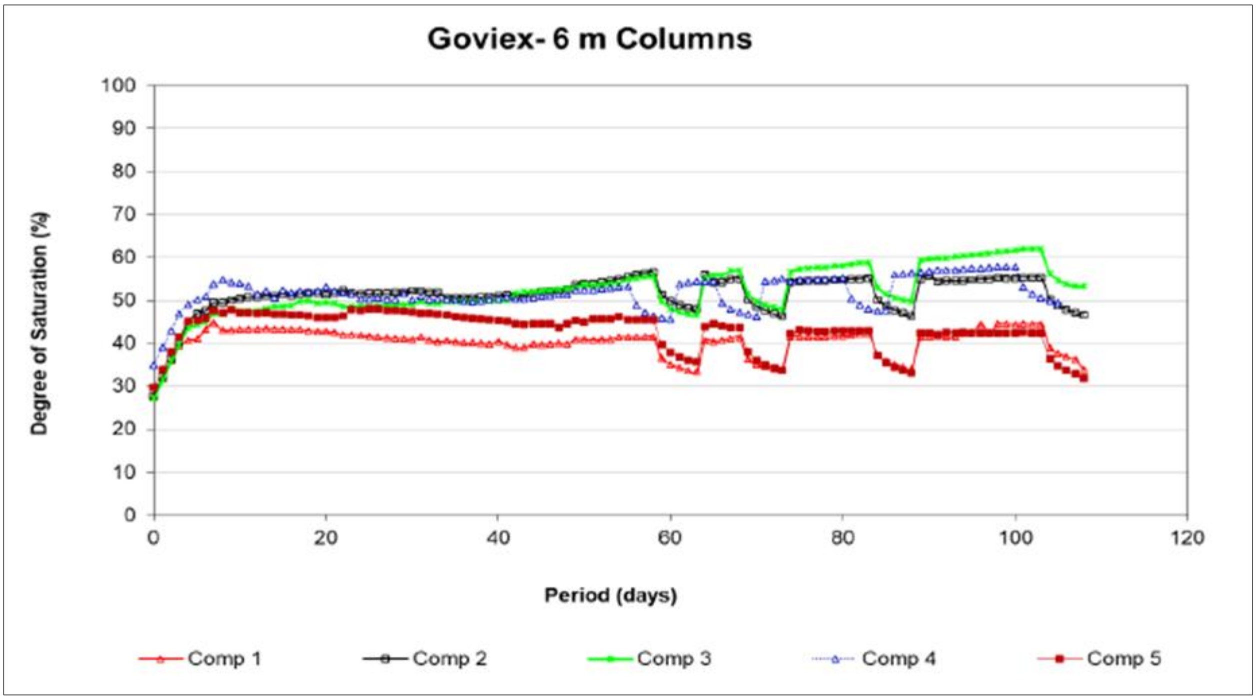

Figure 13-39: Degree of void saturation

Table 13-12: Column leach mass balance

Uranium mass balance

Test	Top size	Curing acid	Masses		U assays			U mass balance					U dissolution [head and residue]		Out/ In	Acid consumption
			Head	Residues	Head		Residue	IN head		OUT residue	IN solns	OUT solns				
	[P_{80}, mm]	[kg/t]	[kg]	[kg]	[ppm][1]	[ppm][2]	[ppm]	[g][1]	[g][2]	[g]	[g]	[g]	[%][1]	[%][2]	[%]	[kg/t]
Comp 1	25	5.0	176.4	166.0	146	185	3.5	26	33	2.2	0.3	31	91.3	93.2	127	7.0
Comp 2	25	18.0	179.3	161.2	251	331	31.3	45	59	5.0	1.6	56	88.8	91.5	132	27.0
Comp 3	25	2.0	175.9	164.8	266	306	16.6	47	54	2.7	2.2	53	94.2	94.9	115	7.8
Comp 4	25	9.0	179.3	170.1	116	188	50.3	21	34	8.6	0.0	25	59.0	74.6	161	17.1
Comp 5	25	11.0	179.1	164.5	159	185	12.0	28	33	2.0	0.8	32	93.1	94.1	117	19.6

Notes:

1) Assayed head (total uranium)
2) Recalculated head (total uranium)

Mg mass balance

Test	Top size	Curing acid	Masses		Mg assays			Mg mass balance					Mg dissolution [head and residue]		Out/ In	Acid consumption
			Head	Residues	Head		Residue	IN head		OUT residue	IN solns	OUT solns				
	[P_{80}, mm]	[kg/t]	[kg]	[kg]	[%][1]	[%][2]	[%]	[g][1]	[g][2]	[g]	[g]	[g]	[%][1]	[%][2]	[%]	[kg/t]
Comp 1	25	5.0	176.4	166.0	1.95	2.10	2.15	3 441	3 699	3 563	110	246	-3.5	3.7	108	7.0
Comp 2	25	18.0	179.3	161.2	1.98	2.15	2.26	3 546	3 858	3 640	1 428	1 646	-2.6	5.7	109	27.0
Comp 3	25	2.0	175.9	164.8	1.54	1.77	1.78	2 716	3 113	2 938	1 342	1 516	-8.2	5.6	115	7.8
Comp 4	25	9.0	179.3	170.1	2.08	3.30	3.36	3 729	5 922	5 710	925	1 136	-53.2	3.6	159	17.1
Comp 5	25	11.0	179.1	164.5	1.16	1.36	1.04	2 073	2 475	1 709	1 808	2 574	17.5	30.9	119	19.6

Notes:

1) Assayed head (total Mg)
2) Recalculated head (total Mg)

Fe mass balance

Test	Top size	Curing acid	Masses		Fe assays			Fe mass balance					Fe dissolution [head and residue]		Out/ In	Acid consumption
			Head	Residues	Head		Residue	IN head		OUT residue	IN solns	OUT solns				
	[P_{80}, mm]	[kg/t]	[kg]	[kg]	[%][1]	[%][2]	[%]	[g][1]	[g][2]	[g]	[g]	[g]	[%][1]	[%][2]	[%]	[kg/t]
Comp 1	25	5.0	176	166	0.12	0.17	0.17	213	298	280	52	70	-31.7	6.0	140	7.0
Comp 2	25	18.0	179	161	0.21	0.27	0.24	378	481	385	127	223	-1.6	20.0	127	27.0
Comp 3	25	2.0	176	165	0.06	0.11	0.10	103	19	159	37	71	-49.6	17.4	181	7.8

Fe mass balance

Test	Top size	Curing acid	Masses		Fe assays			Fe mass balance					Fe dissolution [head and residue]		Out/ In	Acid consumption
			Head	Residues	Head		Residue	IN head		OUT residue	IN solns	OUT solns				
	[P_{80}, mm]	[kg/t]	[kg]	[kg]	[%][1]	[%][2]	[%]	[g][1]	[g][2]	[g]	[g]	[g]	[%][1]	[%][2]	[%]	[kg/t]
Comp 4	25	9.0	179	170	0.21	0.25	0.16	378	448	272	140	316	28.2	39.3	118	17.1
Comp 5	25	11.0	179	165	0.10	0.14	0.11	184	243	176	93	160	4.3	27.7	132	19.6

Notes:
1) Assayed head (total Fe)
2) Recalculated head (total Fe)

Al mass balance

Test	Top size	Curing acid	Masses		Al assays			Al mass balance					Al dissolution [head and residue]		Out/ In	Acid consumption
			Head	Residues	Head		Residue	IN head		OUT residue	IN solns	OUT solns				
	[P_{80}, mm]	[kg/t]	[kg]	[kg]	[%][1]	[%][2]	[%]	[g][1]	[g][2]	[g]	[g]	[g]	[%][1]	[%][2]	[%]	[kg/t]
Comp 1	25	5.0	176	166	3.85	4.37	4.63	6 795	7 699	7 687	22	34	-13.1	0.2	113	7.0
Comp 2	25	18.0	179	161	4.44	4.57	5.14	7 962	8 203	8 284	114	34	-4.0	-1.0	103	27.0
Comp 3	25	2.0	176	165	3.85	4.58	4.88	6 778	8 062	8 041	29	50	-18.6	0.3	119	7.8
Comp 4	25	9.0	179	170	2.39	2.59	2.71	4 288	4 650	4 602	9	56	-7.3	1.0	106	17.1
Comp 5	25	11.0	179	165	3.67	3.90	4.23	6 580	6 985	6 963	29	51	-5.8	0.3	106	19.6

Notes:
1) Assayed head (total Al)
2) Recalculated head (total Al)

Mn mass balance

Test	Top size	Curing acid	Masses		Mn assays			Mn mass balance					Mn dissolution [head and residue]		Out/ In	Acid consumption
			Head	Residues	Head		Residue	IN head		OUT residue	IN solns	OUT solns				
	[P_{80}, mm]	[kg/t]	[kg]	[kg]	[%][1]	[%][2]	[%]	[g][1]	[g][2]	[g]	[g]	[g]	[%][1]	[%][2]	[%]	[kg/t]
Comp 1	25	5.0	176	166	0.14	0.19	0.17	250	338	279	97	157	-11.6	17.5	135	7.0
Comp 2	25	18.0	179	161	0.13	0.08	0.03	230	148	49	142	241	78.6	66.8	65	27.0
Comp 3	25	2.0	176	165	0.04	0.03	0.01	64	60	22	56	94	66.2	63.6	93	7.8
Comp 4	25	9.0	179	170	0.29	0.35	0.10	522	633	175	434	892	66.4	72.3	121	17.1
Comp 5	25	11.0	179	165	0.01	0.03	0.01	16	46	10	52	88	34.8	77.3	287	19.6

Notes:
1) Assayed head (total Mn)
2) Recalculated head (total Mn)

Dissolutions by column height are shown in Figure 13-40 and Figure 13-41 determined from solid head and residue assays per 1 m section. The apparent dissolutions were normalised to take into account the movement of ore between different sections and to take into account the mass change that occurs. The corrected residue assays were then used to calculate the normalised uranium dissolutions by height, which gives a better estimate of the actual uranium dissolutions per section. The overall uranium dissolution remains the same for the actual and normalised results since the total mass is conserved during the normalisation process. The results show little variation in uranium dissolution between the different 1 m sections, suggesting there was no reagent limitation down the length of the column. Furthermore, curing acid was evenly distributed during agglomeration, providing the acid requirement for initial dissolution and ensuring that parts of the bed were less likely to be acid limited.

Dissolutions by size class (4 size classes) are presented in Figure 13-42 and Figure 13-43. Apparent dissolutions by size class were similarly corrected to consider changes in masses (for example due to decrepitation) within the size classes. The corrected dissolutions are presented in Figure 13-43. Interestingly, the results show little variation between the size classes, except for Composite 4, which had lower dissolutions in the coarsest size class. This suggests that the uranium minerals were well liberated and accessible to leach solution, even in the coarser rocks. It suggests that crush size doesn't affect the extent of mineral dissolution.

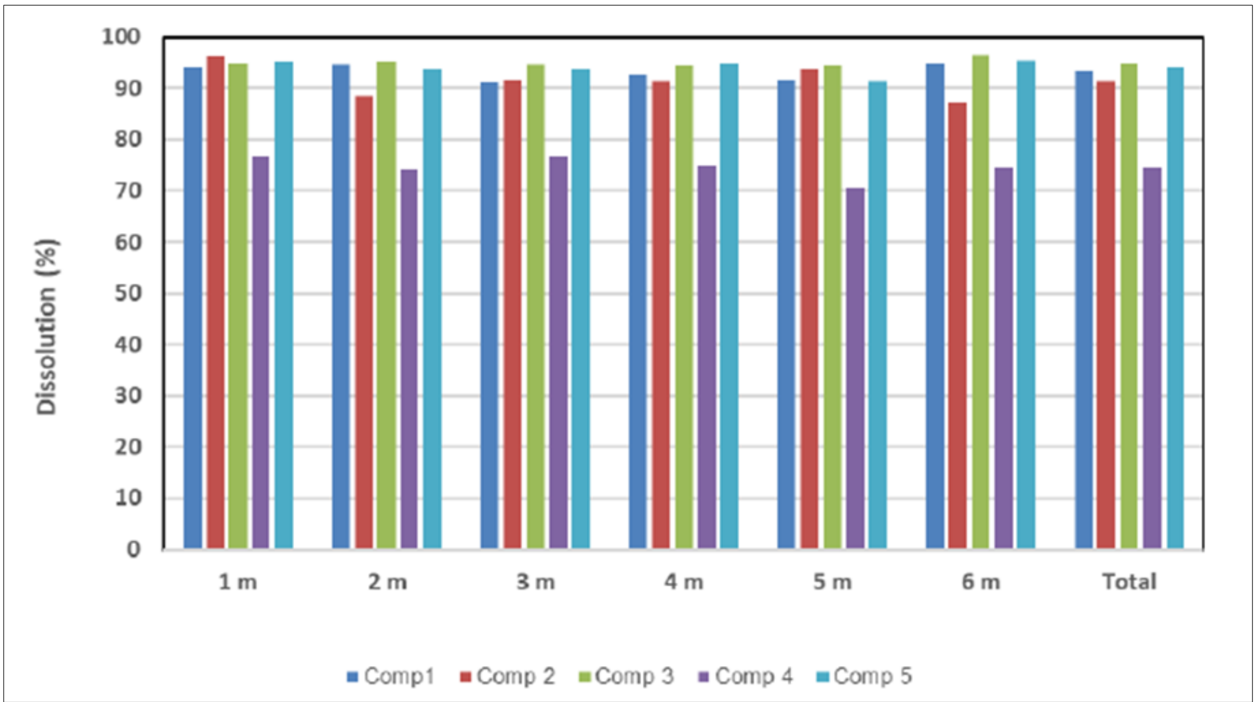

Figure 13-40: Apparent uranium dissolution in columns

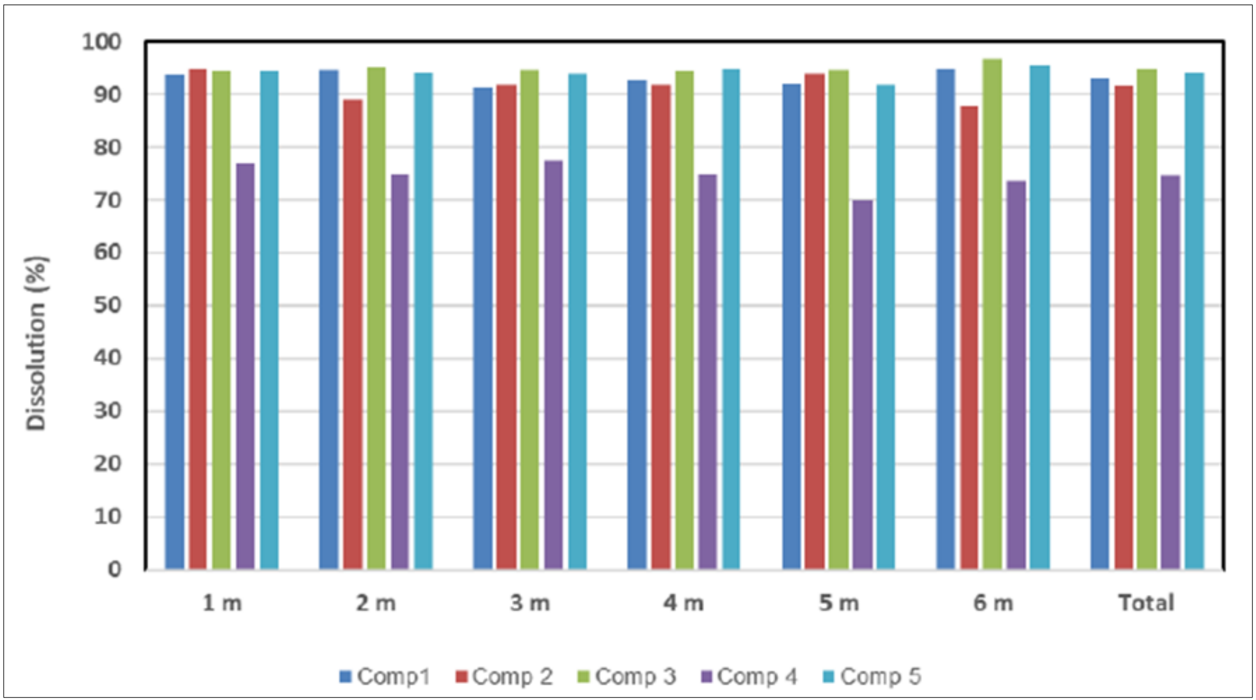

Figure 13-41: Dissolution by height (corrected)

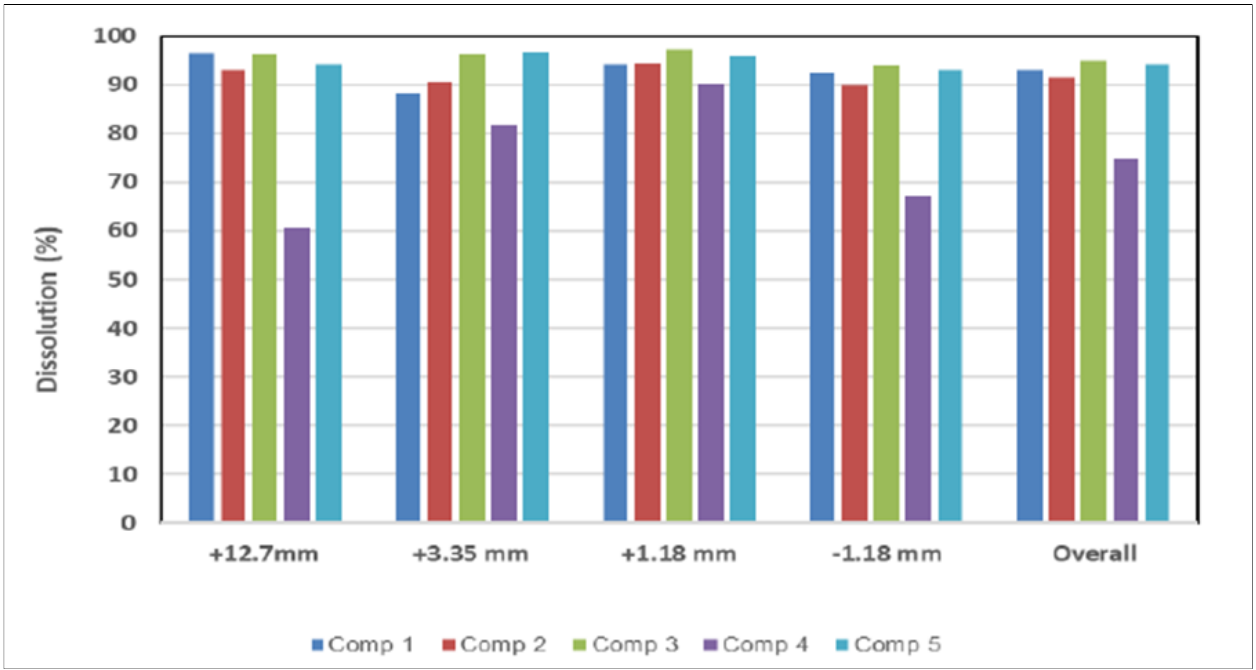

Figure 13-42: Dissolution by grain size class

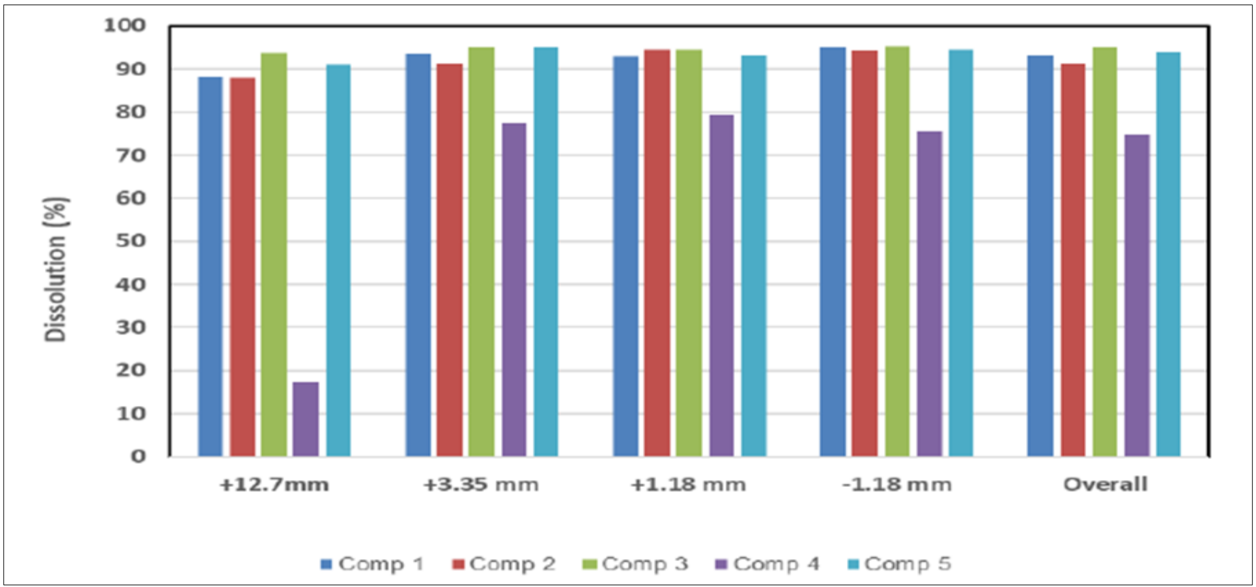

Figure 13-43: Dissolution by size class (adjusted)

13.15.Recovery – Ion exchange and precipitation

13.15.1. Methodology

The PLS generated during the first three weeks from each of column 1 (Dibbwi East Oxi), column 2 (Dibbwi East Red), column 3 (Muntanga and Njame), column 4 (Gwabi) and column 5 (Dibbwi) were composited and used for the recovery and precipitation testwork. The following tests were performed, using standard Mintek procedures. The full methodologies and detailed results of all the tests performed can be found in the Mintek external report.

- Bulk loading and elution tests on each composite using Purolite MTA8000PSO4 resin
- IX adsorption isotherm on a prepared PLS from composite 4
- Rinsing of loaded resin to remove iron
- NF scouting tests using two alternate Filmtech membranes; NF90 and NF245
- Bulk nanofiltration using NF245
- Acid neutralisation of both eluate and nanofiltration concentrate using lime
- Iron precipitation from both eluate and nanofiltration concentrate using sodium hydroxide
- Uranium precipitation using hydrogen peroxide under pH control and sodium hydroxide.

13.15.2. Results

The feed PLS composition is listed in Table 13-13.

Table 13-13: IX feed PLS composition

Description	Al	Ca	Co	Cu	Fe	Mg	Mn
	ppm	ppm	ppm	ppm	ppm	ppm	ppm
Comp-1 PLS	193	293	5.26	53.4	910	405	740
Comp-2 PLS	758	290	5.78	84.1	3 340	810	897
Comp3-PLS	197	310	7.12	17.8	3 110	244	359
Comp4-PLS spiked	149	258	<2	33.7	2 250	203	513
Comp5-PLS	207	271	3.86	11.2	6 540	636	347
	Mo	Ni	S	Si	V	Zn	U
	ppm	ppm	ppm	ppm	ppm	ppm	ppm
Comp-1 PLS	<2	3.38	2 570	80	<2	12.7	193
Comp-2 PLS	<2	4.45	5 670	54.3	2.6	18.3	279
Comp3-PLS	<2	3.94	3 290	71	<2	10.6	291
Comp4-PLS spiked	<2	<2	3 410	61.4	<2	3.13	427
Comp5-PLS	<2	3.77	5 640	66.6	3.44	16	212

13.15.3. Bulk loading and elution

Breakthrough curves from each of the bulk loading tests for each composite are shown in Figure 13-44.

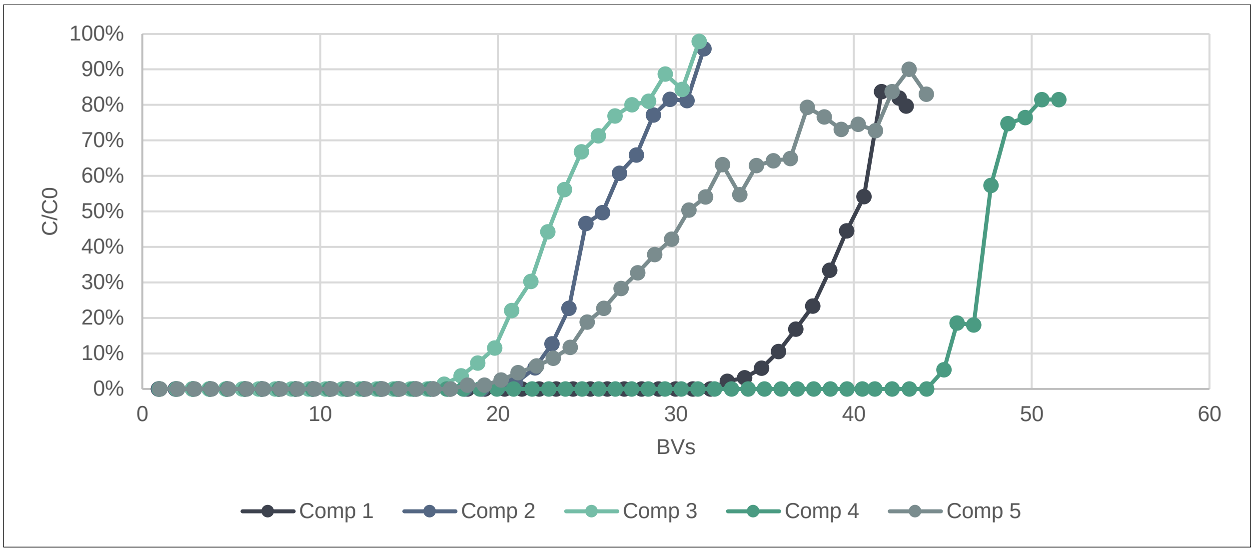

Figure 13-44: IX Break-through curves

Column 4 was started after the others and took significantly longer to achieve a breakthrough. Consequently, testwork continued using loaded resin from the other columns, while resin from composite 4 was retained for further work. Analyses of the various components of the bulk IX tests are summarised in Table 13-14. Full solution analyses are available in the Mintek report.

Table 13-14: IX Bulk loading solution analyses

	Comp 1		Comp 2		Comp 3		Comp 5	
	ppm Fe	ppm U	ppm Fe	ppm U	ppm Fe	ppm U	ppm Fe	ppm U
PLS	935	246	5 627	356	4 835	384	11 020	219
Loaded resin	706	48 229	2 847	46 049	1 585	45 116	5 119	35 204
Barren 1	1 336	4	5 801	22	4 935	28	272	7
Eluate	78	5 363	225	5 028	188	5 163	605	5 380

The tests show that uranium is effectively recovered from the PLS, and low barren concentrations can be achieved. The resin selectively loads uranium over iron and achieves high uranium loading, although some iron does co-load. An eluate solution containing around 5 g/L uranium can be produced using 100 g/L sulfuric acid solution.

13.15.4. Resin scrubbing

Because some iron is still present in the eluate, it was decided to use the loaded resin from column 4 to test the effectiveness of scrubbing co-loaded iron using a dilute sulfuric acid solution before uranium elution. Two different acid concentrations were tested, each at 1, 3 and 5 bed volumes of total solution, and the results are summarised in Figure 13-45 and Figure 13-46.

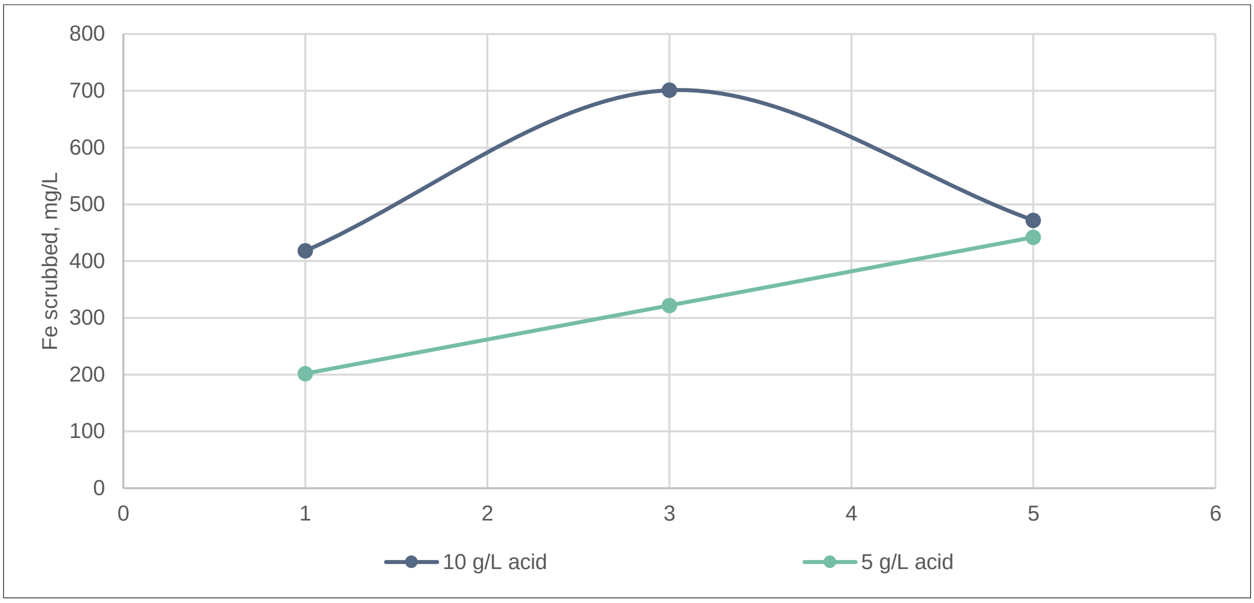

Figure 13-45: Resin scrubbing results; Iron concentration vs solution volume

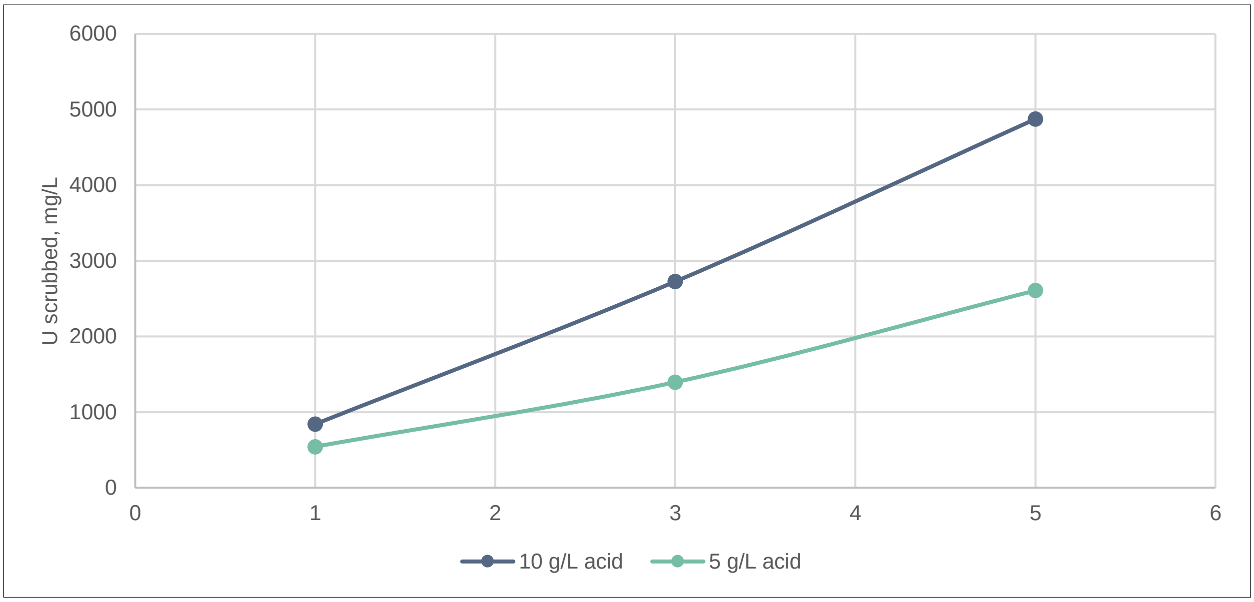

Figure 13-46: Resin scrubbing results; Uranium concentration vs solution volume

Iron is effectively scrubbed from the resin using a solution of 10 g/L sulfuric acid, along with a small portion of the uranium, with peck performance obtained using three-bed volumes of solution. Note that the used scrub solution will be recycled to the PLS so that no uranium is lost.

13.15.5. Adsorption and elution

Results from the adsorption and elution tests using the composite 4 solution and summarised in Figure 13-47 and Figure 13-48.

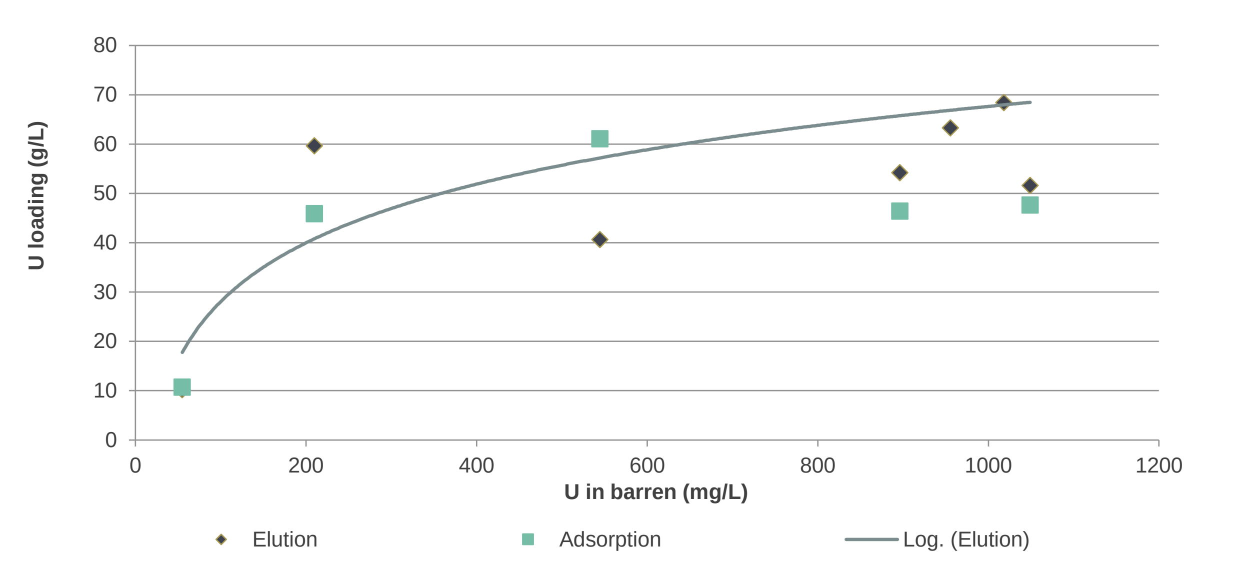

Figure 13-47: Uranium loading isotherm

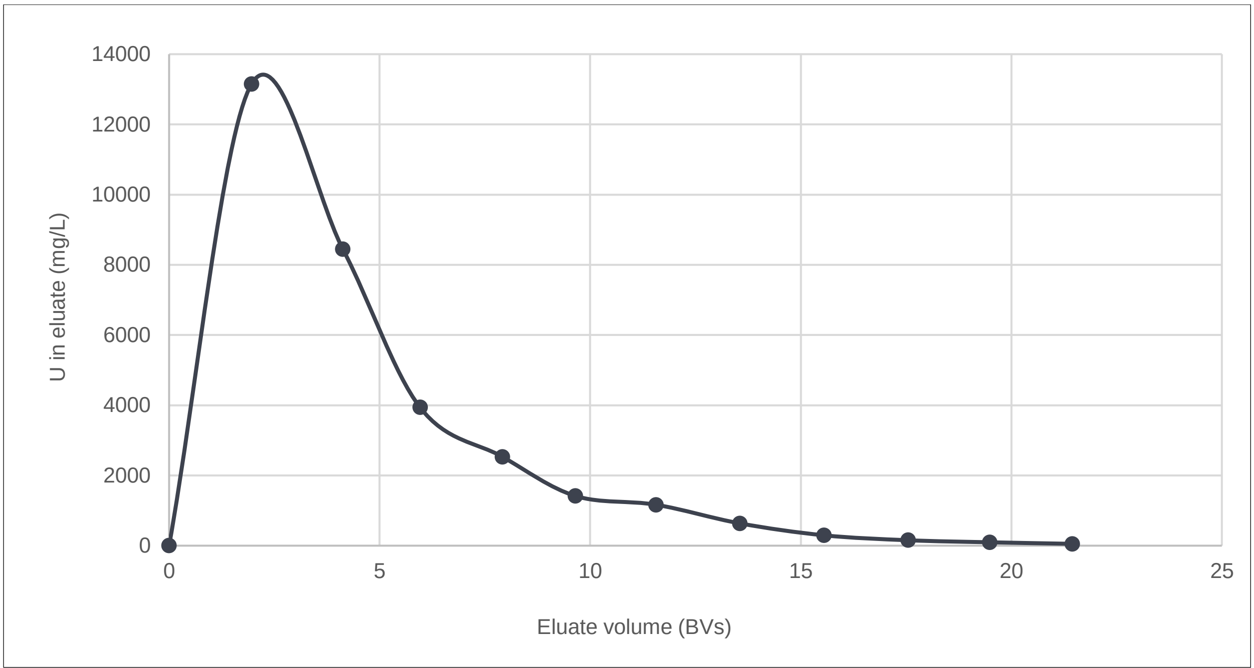

Figure 13-48: IX elution performance

13.15.6. Nanofiltration

Nanofiltration tests were conducted using the eluate produced from all of the composites and indicated that the process could consistently recover 80 % of the sulfuric acid into 20 % of the volume while maintaining a high flux. Data from the composite 4 tests using two different membranes are presented in Figure 13-49 to Figure 13-51. The membrane used in Test 2 (NF245) produced equivalent acid recovery with lower uranium loss at higher flux. The tests produced a concentrate solution containing 22.5 g/L uranium and 104 g/L sulfuric acid which was used in the subsequent neutralisation and precipitation tests. The permeate solution of around 90 g/L sulfuric acid contained 11 % of the uranium, which will be recycled in the plant.

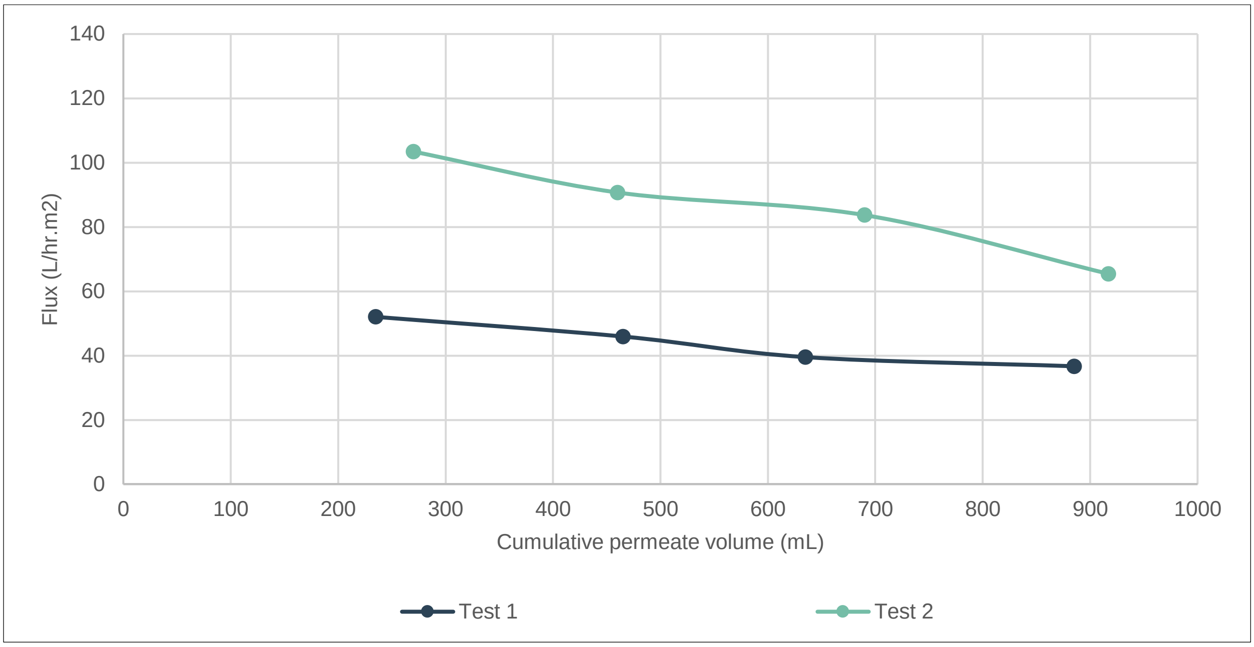

Figure 13-49: Nanofiltration flux vs permeate volume

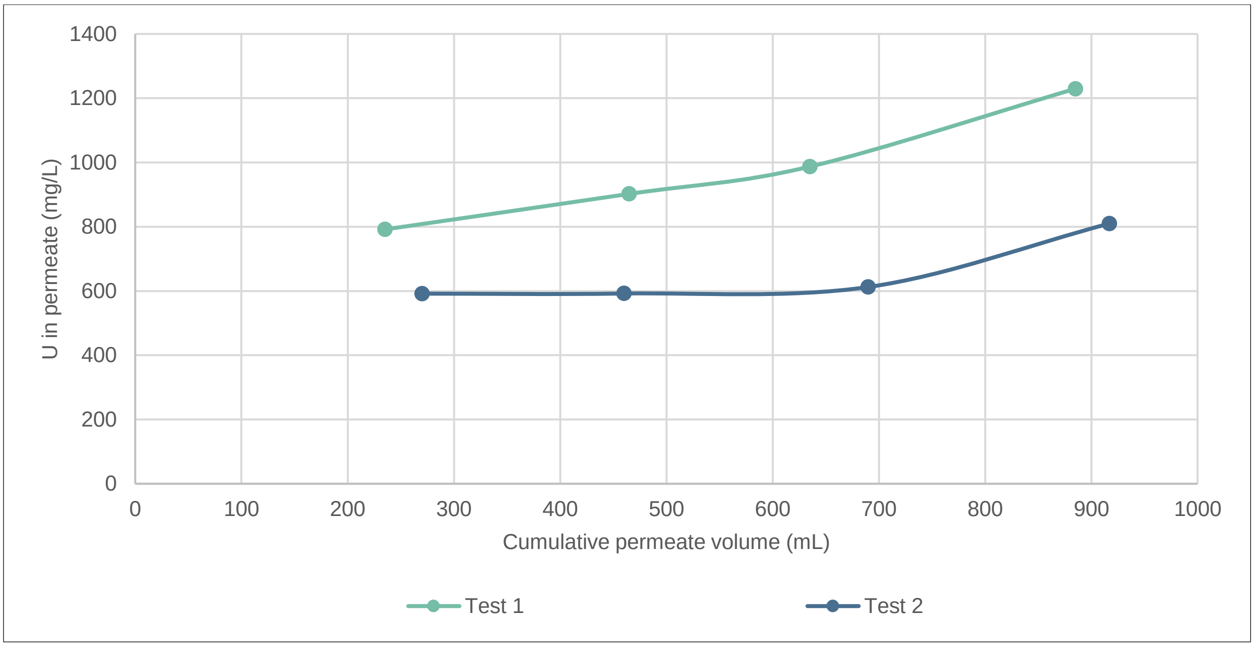

Figure 13-50: Nanofiltration Uranium concentration recovery

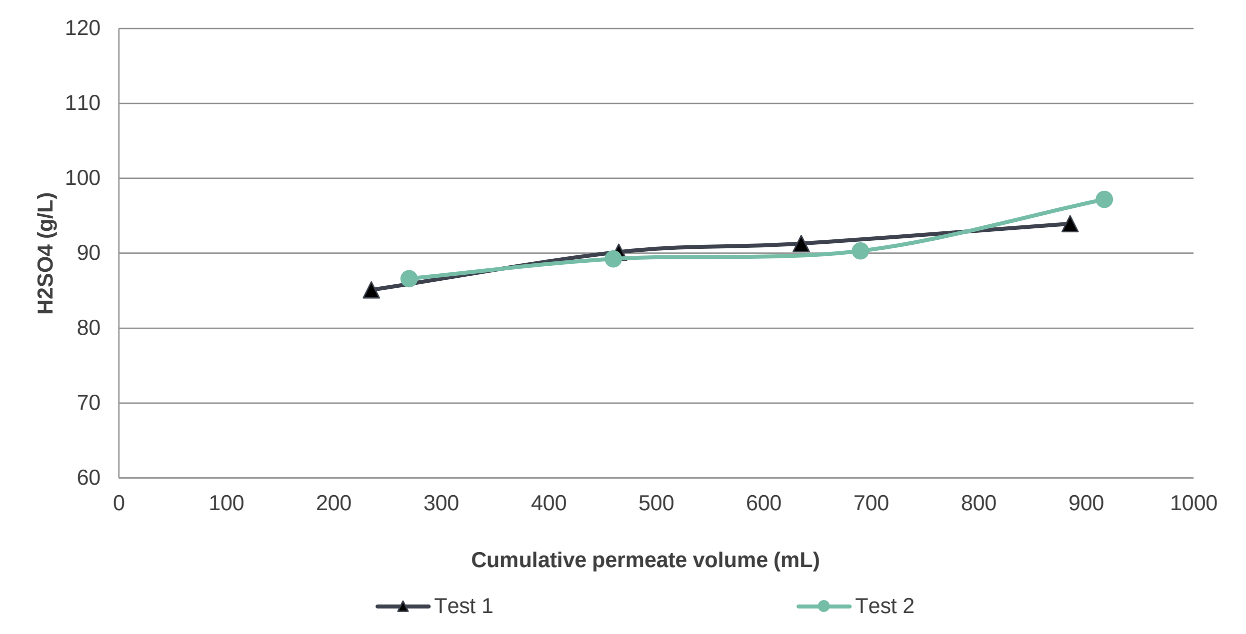

Figure 13-51: Acidity vs change in permeate volume

13.15.7. Acid neutralisation/ Gypsum precipitation

Excess acid must be neutralised before uranium can be precipitated from the solution. Lime is the preferred industrial reagent for this duty due to its low cost, although it carries the risk of introducing iron into the solution. Neutralisation tests were done at several pH endpoints to monitor the iron deportment, using both IX eluate and NF concentrate solutions to understand the effect of varying ionic strength. It was observed that a significant concentration of iron remains in solution during the acid neutralisation step, as illustrated in Figure 13-52.

The chemical analysis of the lime used in these tests is given in Table 13-15, and the final solutions produced in Table 13-16.

Table 13-15: Lime analysis

Element	Unit	Value
Al	%	0.21
Ca	%	47.8
Co	%	<0.05
Cr	%	<0.05
Cu	%	<0.05
Fe	%	0.4
Mg	%	0.61
Mn	%	<0.05
Ni	%	<0.05
Pb	%	<0.05
Si	%	1.86
Ti	%	<0.05
V	%	<0.05
Zn	%	<0.05
Ag	ppm	1.7

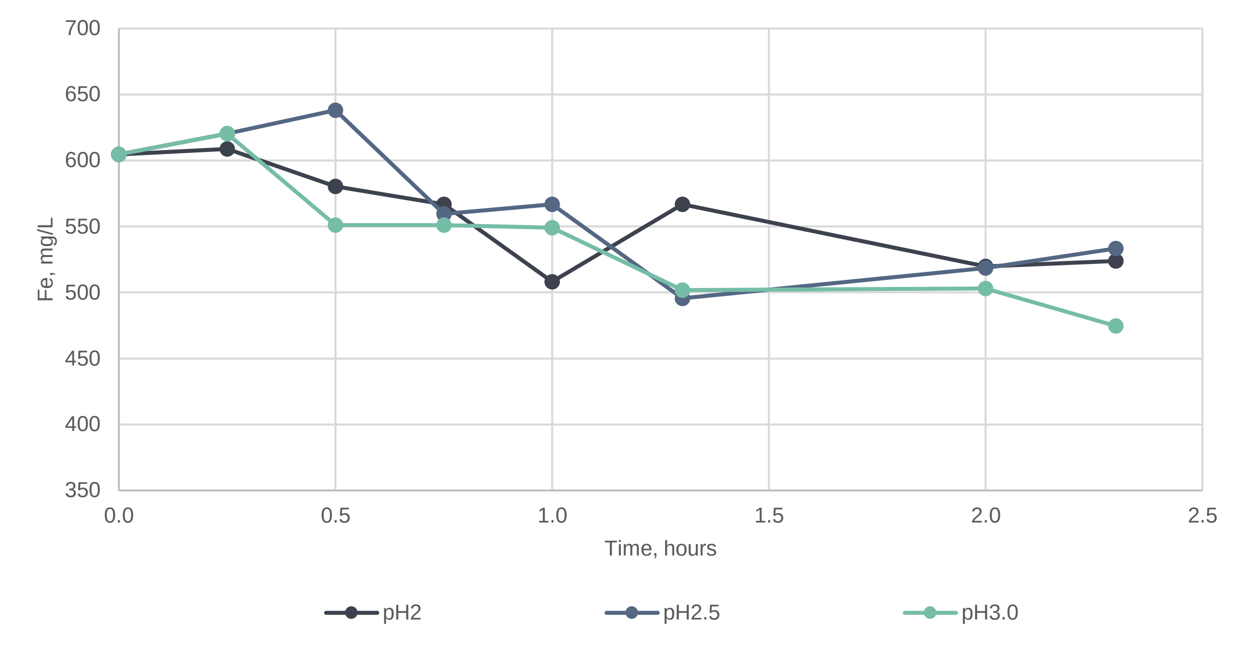

Figure 13-52: Iron concentration during Gypsum precipitation

Table 13-16: Gypsum precipitation solution analysis

U and Fe [ICP-OES] results						
Sample name	pH	Eh	Fe, mg/L	U, mg/L	Volume solution after test	$Ca[OH]_2$ added, g/L
Comp 4 IX eluate	2.50	513	145	6 101	360	49
Comp 4 NF conc	2.52	523	303	21 500	220	85

13.15.8. Iron precipitation

Iron remaining in solution must be removed before uranium precipitation to produce a clean uranium product, which is done by precipitation using sodium hydroxide. Chemical analyses of the feed and neutralised solutions (to pH 3.5) are presented in Table 13-7, where the iron is effectively precipitated from the solution.

Table 13-17: Iron precipitation results

Element	Unit	IX-eluate Feed	IX eluate - neutralised	NF-conc Feed	NF-conc neutralised
Al	ppm	149	3	178.5	14
As	ppm	<2	<2	<2	<2
Ca	ppm	561	10	668	67
Cd	ppm	<2	<2	<2	<2
Co	ppm	<2	<2	2.34	2
Cr	ppm	<2	<2	5.485	5
Cu	ppm	6.26	15	21.85	23
Fe	ppm	145	23	303	100
Li	ppm	<2	<2	<2	<2
Mg	ppm	352	<2	379	9
Mn	ppm	2.86	<2	3.52	2
Mo	ppm	<2	<2	<2	<2
Ni	ppm	<2	3	4.03	5
Pb	ppm	4.85	5	18.45	18
S	ppm	2 200	32 000	4 635	34 000
Si	ppm	301	<2	379	6
Ti	ppm	3.38	<2	12	7
V	ppm	17.6	19	65.3	82
Zn	ppm	<2	<2	7.02	9
U	ppm	6 101	5 752	21 500	23 300

13.15.9. Uranium precipitation

A single uranium precipitation test was conducted from each of the eluate and nanofiltration concentrate samples from the composite 4 material. The test was done at 50ºC, with pH controlled at 3.5 using sodium hydroxide. Analyses of the yellowcake produced are given in Table 13-18.

Table 13-18: Final product analysis

Unit	%
U	70
U_3O_8	83
Al	0.12
Ca	0.19
Mg	0.05
Pb	0.04
Si	0.05
V	0.14
K	0.21
Na	0.02
Fe	0.20

Figure 13-53: Uranium precipitation test

13.16.Miscellaneous testing summary

The application of curing acid optimisation test protocols provided for a substantial reduction in the leach cycle in the columns to between 20 and 60 days. Most of the uranium dissolves in the curing stage and washes out during initial irrigation. The application of curing acid provides a means to introduce the acid rapidly and evenly throughout the bed and avoid acid limitation within zones of poor solution contact.

High uranium dissolutions of 90 % and above were achieved in the columns which contained Muntanga and Dibbwi East ores, which account for 80 % of the M&I Mineral Resource. Only the Gwabi ore yielded lower dissolutions of 75 %.

Slumping was minimal (<1.7 %) as measured by a decrease in the height of the bed. Slumping was also reduced by compacting the agglomerates during loading to between 1.46 t/m^3 to 1.49 t/m^3 by hammering on the sides of the column. This prevents further compaction during leaching on account of agglomerates “settling” to a higher bulk density during wetting. Moreover, no permeability restraints were observed during irrigation in the columns at 10 $L/m^2/h$. Compared with test heaps; columns are known to provide “wall support” which is absent in heaps.

The good permeability may be attributed to a low percentage of silt plus clay (-75 µm) material in the feed solids. The silt plus clay fraction is known to block pores if it is present in amounts greater than 10 % to 14 %. Even though the samples contained a large percentage of sand, normally classified as -2 mm or -4 mm, this is not associated with poor permeability, although it is associated with increased surface area and moisture content. During the leach, however, decrepitation increased the fines (-75 µm) content to 15 %, close to the recommended limit, although this did not translate into permeability restraints in the columns. Hydrodynamic column tests on the residues indicated that the columns operated close to saturation on account of the high fine content generated during the leach.

Uranium dissolutions by 1 m section and by size class were fairly uniform, suggesting that there was no reagent limitation down the height of the column, nor is uranium finely disseminated/ locked in coarser rocks. In other words, the crush size has little effect on the uranium dissolution, as the uranium is liberated, even in the coarser rocks.

13.17.Summary of approach and recoveries

The resulting flowsheet will comprise the following steps:

- Primary crushing and ore sorting at satellites, sorted ore trucked to Central plant using the mining fleet
- Primary crushing of ore from Central pits
- Secondary and tertiary crushing of combined ore
- Agglomeration with sulfuric acid and stacking
- Heap leaching to produce a PLS containing uranium, iron and other impurities followed by ripeos reclamation and disposal
- Recovery of uranium from PLS using IX, with barren solution recycled to the heap
- The concentration of IX eluate by nanofiltration, with recycling of recovered acid to IX elution
- Neutralisation of excess acid in NF concentrate using lime
- Precipitation of iron with sodium hydroxide
- Precipitation of uranium using hydrogen peroxide
- Calcining and packaging to produce U_3O_8.

13.18.Future testwork

There are several components of the process design that can be optimised by future testwork.

1. The control of iron leaching in the heap, and hence peroxide consumption, can be optimised by recirculating solutions continuously through a number of cycles using small lab columns
2. The final product precipitation process can be optimised with respect to impurity deportment, particularly iron. These tests can be done using PLS produced by the small column tests described above, using lime sourced from Zambia
3. Finally, rheology work can be done using the gypsum and uranium slurries produced above, to finalise parameters for sizing the various thickening and filtration equipment.

14. Mineral Resource estimates

14.1. Introduction

The Mineral Resource statement presented herein represents an updated MRE prepared for the Project in accordance with the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101"). The project comprises the Muntanga, Dibbwi, Dibbwi East, Gwabi and Njame uranium deposits as depicted in Figure 14-1.

The Mineral Resource model prepared by SRK considers 2 366 historical drill holes drilled between 2006 and 2012, and 468 drill holes drilled by GoviEx from 2021 to 2023. The MRE work was completed by André Deiss, Pr.Sci.Nat. P.Geo., an "independent qualified person" as this term is defined in NI 43-101. The effective date of the Mineral Resource statement is January 31, 2024.

This section describes the MRE methodology and summarises the key assumptions considered by SRK. In the opinion of SRK, the MRE reported herein are reasonable representations of the global uranium Mineral Resources found in the Project at the current level of sampling. The Mineral Resources have been estimated in conformity with the generally accepted Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines" dated November 29, 2019, and "Definition Standards for Mineral Resources and Mineral Reserves" published May 10, 2014, and are reported in accordance with the Canadian Securities Administrators' NI 43-101 standards of disclosure for mineral projects. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resource will be converted into Mineral Reserves.

The database used to estimate the Project MRE was audited by SRK. SRK believes that the current drilling information is sufficiently reliable to interpret with confidence the boundaries for uranium mineralisation and that the sample data are sufficiently reliable to support Mineral Resource estimation.

Seequent's Leapfrog Geo™ ("Leapfrog") and Edge™ ("Edge") version 2024.1.1 software were used to review historical MREs and conduct sensitivity analyses, construct updated geological solids, prepare sample data for geostatistical and variography analysis, construct the block models, estimate uranium grades, and tabulate Mineral Resources.

Figure 14-1: Location map of the Muntanga uranium deposits

14.2. Mineral Resource estimation procedures

The Mineral Resource evaluation methodology involved the following procedures:

- Database compilation and verification
- Review of Njame and Gwabi's historical MRE
- Construction of grade shell wireframe models for the boundaries of uranium mineralisation for the Muntanga, Dibbwi and Dibbwi East deposits
- Data conditioning (compositing and capping) for geostatistical analysis and variography
- Block modelling and grade interpolation
- Mineral Resource classification and validation
- Assessment of “reasonable prospects for economic extraction” (“RPEEE”) and selection of appropriate COGs
- Preparation of the Mineral Resource statement.

14.3. Mineral Resource database

The Mineral Resource drill hole database for the Project contains 2 834 drill holes totalling 191 711 m of drilling; 468 of these drill holes were drilled by GoviEx between 2021 and 2023 totalling 52 924 m of drilling. The database contains 33 280 uranium (U_3O_8) assays and 114 364 m of down-hole radiometric probe data converted in equivalent U_3O_8 (eU_3O_8) grade data for MRE purposes.

Table 14-1: Mineral Resource drill hole database summary

Deposit	Year	Number of AC holes	Total metres AC holes	Number of Diamond core holes	Total metres of diamond core holes	Number of RC holes	Total metres RC holes	Total number of Assay samples collected	Total Assay sample length [m]	Total eU_3O_8 length [m]
Gwabi	2007	0	0	5	200	226	10 905	3 359	3 340	0
	2008	0	0	34	1 168	54	1 628	2 028	1 813	0
	2009	0	0	0	0	6	221	90	90	0
Njame	2006	63	2 794	0	0	0	0	1 650	1 650	0
	2007	0	0	28	1 412	255	14 617	6 202	6 095	0
	2008	0	0	126	6 113	258	14 822	8 344	7 627	0
	2009	0	0	0	0	80	3 540	1 660	1 660	0
Muntanga	2005	0	0	7	332	0	0	456	331	298
	2006	0	0	32	1 788	70	2 052	2 720	2 646	2 677
	2007	0	0	32	1 897	9	540	0	0	2 112
	2008	0	0	207	11 391	263	14 168	851	852	22 058
	2010	0	0	6	313	0	0	0	0	297
	2012	0	0	1	293	2	300	62	31	291
	2021/2022	0	0	11	610	0	0	0	0	569
	2023	0	0	10	490	44	2 830	261	152	3 234
Dibbwi	2006	0	0	0	0	25	1 362	679	679	679
	2007	0	0	27	1682	1	110	36	37	1 569
	2008	0	0	140	12 914	114	7 343	297	297	15 009
	2010	0	0	9	495	0	0	0	0	454
	2012	0	0	6	1 101	14	1 681	337	244	2 344
	2021/2022	0	0	3	300	0	0	0	0	297
Dibbwi East	2023	0	0	12	601	3	360	148	92	832
	2008	0	0	49	3 602	27	854	0	0	3 505
	2011	0	0	34	3 842	98	8 855	2 103	1 361	11 447
	2012	0	0	29	4 151	29	300	0	0	290
	2021/2022	0	0	35	4 699	207	27 705	1 032	594	31 414
	2023	0	0	28	2 075	115	13 254	965	590	14 987
Total		**63**	**2 794**	**871**	**61 469**	**1 900**	**127 448**	**33 280**	**30 181**	**114 363**

14.4. Njame and Gwabi Mineral Resource estimate review

MREs for the Gwabi and Njame deposits were originally developed by AFR in February and December 2009, respectively. SRK reviewed the drill hole databases, geological models, and MREs for the Gwabi and Njame deposits and considers these MREs to be reasonable representations of the global U_3O_8 mineral resources in these deposits at the current level of sampling and geological understanding. It is the opinion of the QP that the Mineral Resources have been estimated in conformity with the generally accepted CIM, "Estimation of Mineral Resource and Mineral Reserve Best Practise Guidelines" and are reported in accordance with the NI 43-101 standards of disclosure for mineral projects.

The following sections describing the geological model, data used for MRE purposes, and MRE parameters have in part been summarised from the following reports:

- AFR Mineral Resource report entitled: Mineral Resource Report for the Njame and Gwabi Uranium Deposits, Chirundu Project, Zambia (2009) and
- SRK Preliminary Ecological Appraisal ("PEA") report entitled: NI 43-101 Technical Report on a Preliminary Economic Assessment of the Muntanga Uranium Project in Zambia (2017).

14.4.1. Mineralisation domain modeling

Mineralisation domains for the Gwabi and Njame deposits were generated using the three-dimensional ("3D") software package Gemcom Surpac® ("Surpac"). Uranium mineralisation occurs in fine to coarse-grained sedimentary units consisting of siltstone, sandstones, pebbly/gritty sandstones, and grits-to-pebble conglomerates. Mineralised lenses occur as sub-parallel layers with shallow dips of 2° to 5° to the southeast at Njame (Figure 14-2), and to the east-northeast at Gwabi (Figure 14-3), and were defined using a 100 ppm U_3O_8 COG.

At Njame, the main concentration of uranium mineralisation occurs at the contact between sedimentary sequences where there is rapid change from fine to coarse sediments. At Gwabi, the main concentration of uranium mineralisation is hosted in a 10 m to 20 m thick coarse-grained sandstone located above a thick siltstone/ mudstone unit.

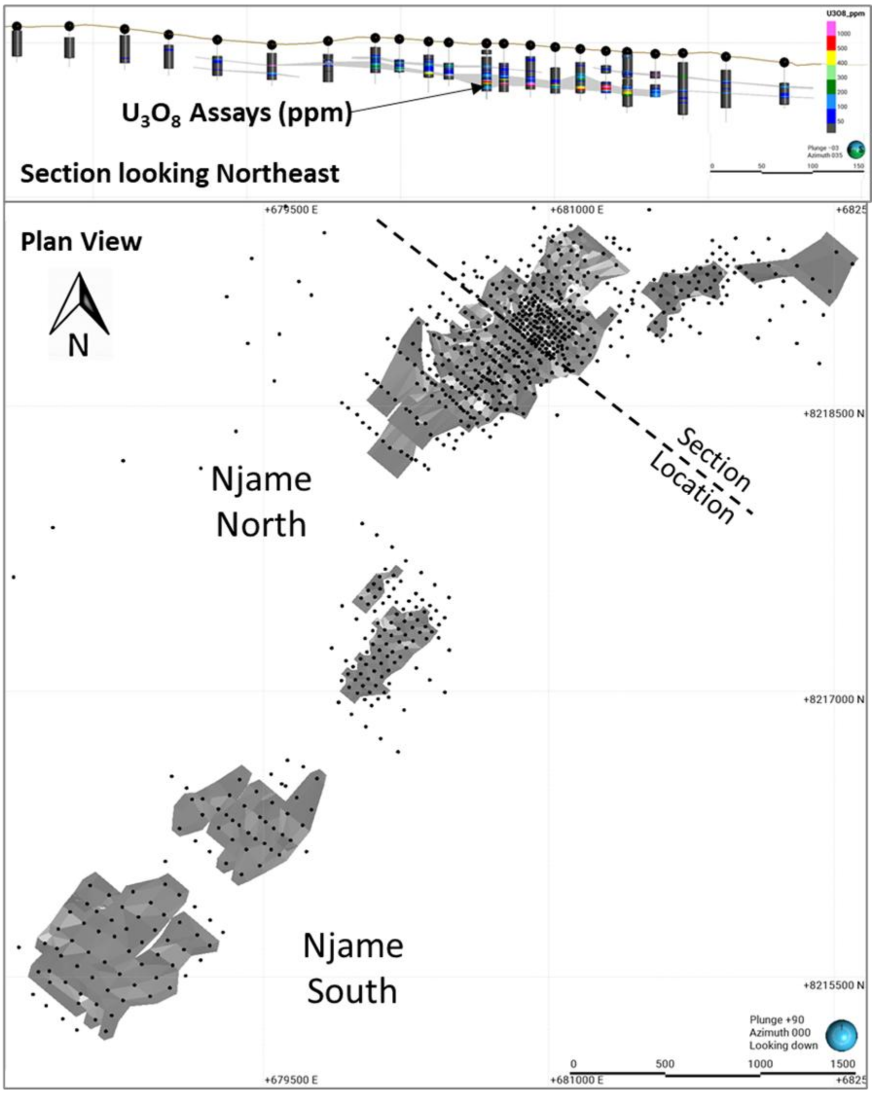

Figure 14-2: Njame deposit mineralisation domain model

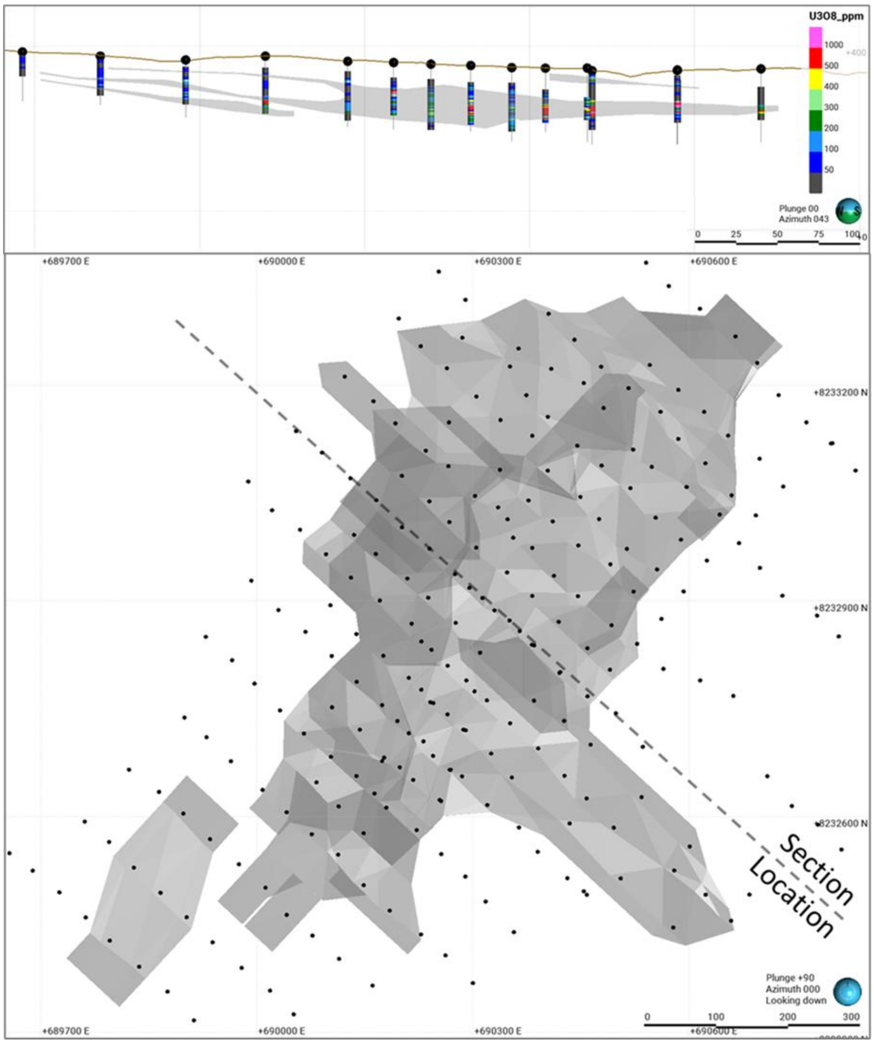

Figure 14-3: Gwabi deposit mineralisation domain model

14.4.2. Bulk density

Bulk density analysis conducted on drill core samples between 2007 and 2008 utilised the water submersion method, where samples were dried and weighed in air followed by plastic wrapping and weighing in water. The bulk density was then determined as a ratio of the weight in air over the weight in water.

A summary of density samples collected a Njame and Gwabi by sedimentary lithology type are summarised in Table 14-2 and Table 14-3, respectively.

Table 14-2: Njame density sample summary statistics

Lithology type	Number of samples	Mean [t/m^3]	Std Dev	Minimum [t/m^3]	Maximum [t/m^3)
Grit	4	1.91	0.06	1.82	1.96
Gritty sandstone	22	1.99	0.06	1.89	2.13
Pebbly grit	32	2.05	0.09	1.89	2.26
Pebbly sandstone	6	2.11	0.10	1.99	2.27
Siltstone	22	2.09	0.15	1.84	2.31
Sandstone	78	1.98	0.09	1.81	2.18

Table 14-3: Gwabi density sample summary statistics

Lithology type	Number of samples	Mean [t/m^3]	Std Dev	Minimum [t/m^3]	Maximum [t/m^3]
Grit	14	2.08	0.12	1.98	2.42
Gritty sandstone	21	2.04	0.14	1.86	2.36
Pebbly grit	22	2.12	0.16	1.85	2.50
Pebbly sandstone	17	2.17	0.19	1.73	2.46
Sandstone	26	2.07	0.15	1.71	2.44

Based on the above sample data, mineralised lenses at Njame were assigned uniform densities ranging from 1.98 t/m^3 to 2.08 t/m^3 dependent on the dominant sedimentary lithology type hosting the mineralisation. At Gwabi, a global density of 2.09 t/m^3 was used for Mineral Resource reporting.

14.4.3.Njame compositing and variography

The drill hole database was composited to 1 m down-hole composite intervals, within the modelled Mineral Resource wireframes; 1 m was chosen as an appropriate composite length as more than 90 % of samples, within the modelled mineralisation, were 1 m length or less and the mining approach is assumed to be reasonably selective. Residual (partial) composites less than 40 % of the 1 m interval were rejected from further study.

Basic statistics of the U_3O_8 composites within all the modelled mineralisation lenses are presented in Table 14-4. The composites have been grouped into two main modelled zones, Njame North and Njame South as many of the individual modelled lenses are small and contain statistically insignificant numbers of samples.

As presented in Figure 14-4 and Figure 14-5, the U_3O_8 grade distribution displays a positive skew with a moderate coefficient of variation. An assessment of the high-grade composites was completed to determine the requirement for high-grade capping. Upon review of the basic statistics and histogram charts, a grade cap of 2 500 ppm U_3O_8 for Njame was selected as highlighted in Figure 14-4 and was applied before estimation.

Grade continuity was modelled using variography calculated and modelled within the geostatistical software Isatis and in the mining package Surpac. Variography was generated for the U_3O_8 variable, based on the 1 m capped down-hole composites (Figure 14-5).

In summary, the key aspects of the variography are:

- The relative nugget has been modelled at approximately 35 %
- 40 % relative variance is modelled to a range of 40 m and
- The overall range of 120 m major, 90 m semi-major, and 8 m minor is noted to be more than the current drill spacing.

The variography indicates that moderate levels of short-range variability exist, which is consistent with this mineralisation style.

Table 14-4: Njame composite summary statistics

Deposit	Samples	Mean U_3O_8 [ppm]	Std Dev	CV	Min U_3O_8 [ppm]	Max U_3O_8 [ppm]
Njame North	2 451	310	389	1.26	0	9 650
Njame South	257	263	197	0.75	0	1 090

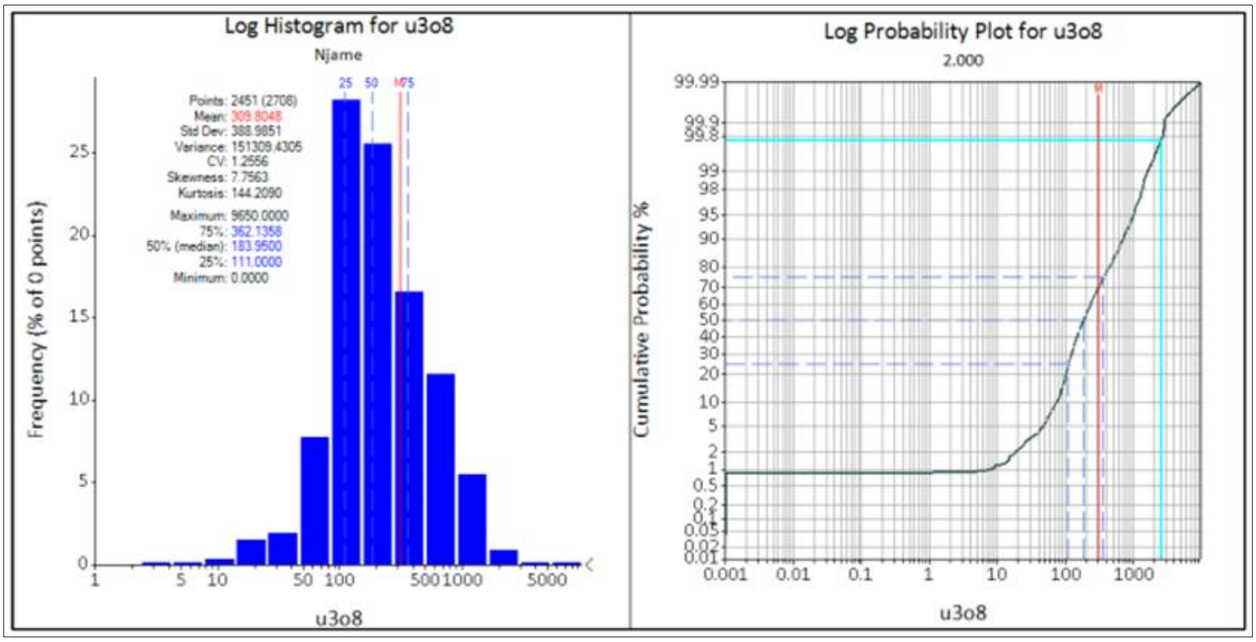

Figure 14-4: Histogram and log-probability plot of Njame North U_3O_8 (ppm) composites (Source: SRK 2007)

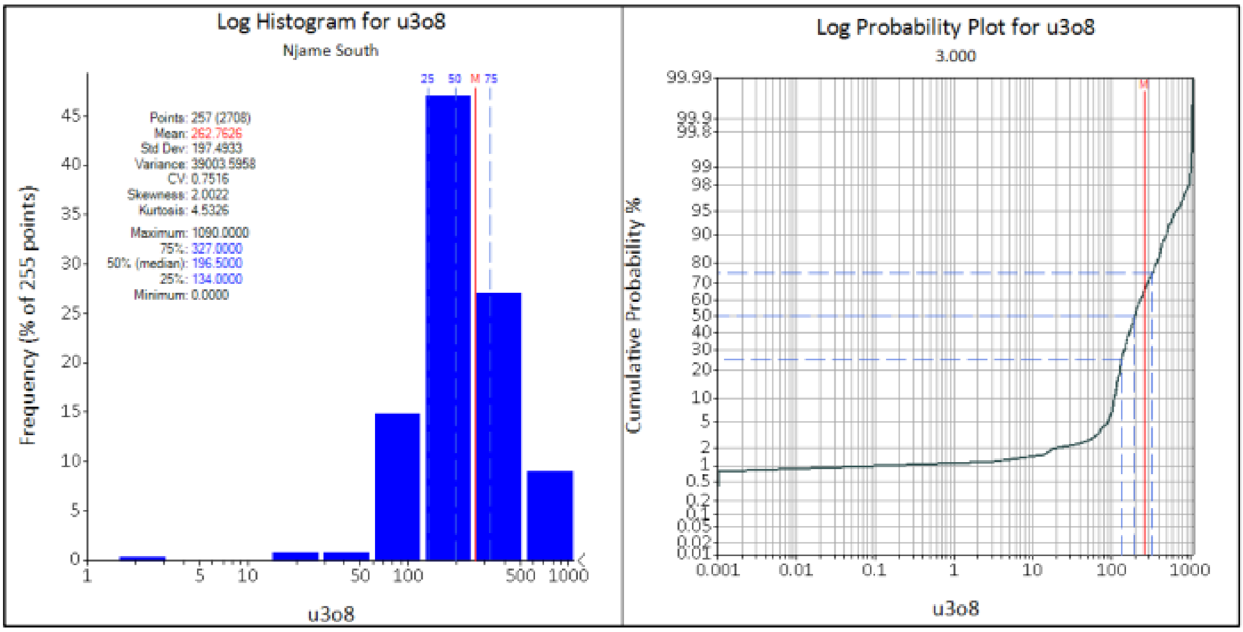

Figure 14-5: Histogram and log-probability plot of Njame South U_3O_8 (ppm) composites (Source: SRK 2007)

Table 14-5: Njame variogram parameters

Deposit	Directions			Normalized Nugget	Structure 1					Structure 2				
					Normalized Sill	Structure	Range [m]			Normalized Sill	Structure	Range [m]		
	Strike	Pitch	Dip				Major	Semi-major	Minor			Major	Semi-major	Minor
Njame	70	0	0	0.35	0.4	Spherical	40	40	3	0.25	Spherical	120	90	8

14.4.4.Gwabi compositing and variography

The drill hole database was composited to 1 m down-hole composite intervals, within the modelled Mineral Resource wireframes. 1 m was chosen as an appropriate composite length as more than 90 % of samples, within the modelled mineralisation, were 1 m in length or less and the mining approach is assumed to be reasonably selective. Residual (partial) composites less than 40 % of the 1 m interval were rejected from further study.

Basic statistics of the U_3O_8 composites within all the modelled mineralisation lenses are presented in Table 14-6. The composites have been grouped as the main modelled lens comprises more than 95 % of the total model volume and the smaller lenses contain a statistically insignificant number of samples (<30 samples each).

As presented in Table 14-6, the U_3O_8 grade distribution displays a positive skew with a moderate coefficient of variation. An assessment of the high-grade composites was completed to determine the requirement for high-grade capping. Upon review of the basic statistics and histogram charts, a grade cap of 1 700 ppm U_3O_8 was selected as highlighted in Figure 14-6 and applied before estimation.

Grade continuity was modelled using the geostatistical software Isatis and the mining package Surpac. Variography was generated for the variable U_3O_8 based on the 1 m capped down-hole composites (Table 14-7).

In summary, the key aspects of the variography analysis are:

- The relative nugget has been modelled from a down-hole variogram at approximately 25 %
- 30 % relative variance is modelled to a range of 110 m and
- The overall range of 350 m major, 170 m semi-major, and 8 m minor is noted to be more than the current drill spacing.

The variography indicates that moderate levels of short-range variability exist, which is consistent with this mineralisation style.

Table 14-6: Gwabi composite summary statistics

Samples	Mean U_3O_8 [ppm]	Std Dev	CV	Min U_3O_8 [ppm]	Max U_3O_8 [ppm]
1 270	273	373	1.36	0	4 920

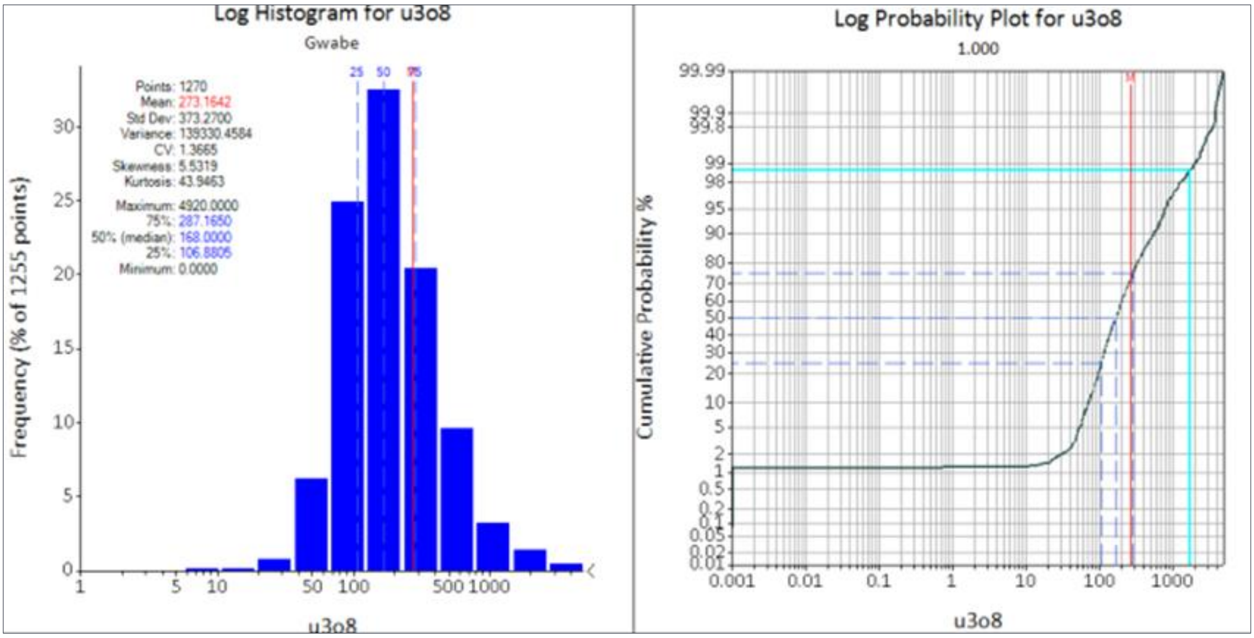

Figure 14-6: Histogram and log-probability plot of Gwabi U_3O_8 (ppm) composites (Source: SRK 2007)

Table 14-7: Gwabi variogram parameters

Deposit	Directions			Normalised Nugget	Structure 1					Structure 2				
					Normalised Sill	Structure	Range [m]			Normalised Sill	Structure	Range [m]		
	Strike	Pitch	Dip				Major	Semi-major	Minor			Major	Semi-major	Minor
Gwabi	17	0	0	0.25	0.3	Spherical	110	60	2	0.45	Spherical	350	170	8

14.4.5.Block model configuration

Block model configuration details for Gwabi and Njame are summarised in Table 14-8. A parent block size of 25 x 25 x 2.0 m was sub-blocked for volumetric reporting. Grade interpolation was conducted at the parent block size of 25 x 25 x 2.0 m.

Table 14-8: Block model configuration details for Gwabi and Njame

Deposit	Parameters	X [m]	Y [m]	Z m]
Gwabi	Parent block size	25	25	2
	Sub-block size	6.25	6.25	0.5
	Basepoint*	689 804	8 230 494	594
	Boundary size	4,025	2 525	400
	Rotation			312
Njame	Parent block size	25	25	2
	Sub-block size	6.25	6.25	0.5
	Basepoint*	676 997	8 215 700	600
	Boundary size	2 525	6 750	250
	Rotation			40

*Coordinates specified in UTM WGS84 Zone 35S reference datum

14.4.6. Grade estimation

Grade estimation was completed within an area encompassing all the modelled Njame and Gwabi mineralised zones with block model geometry and extents as presented in Table 14-8. A parent block size of 25 x 25 x 2 m, sub-blocked to 6.25 x 6.25 x 0.5 m, representing the approximate drill spacing of the tightly infilled drilling area, was chosen for the model.

The resource estimation methodology was based on the following:

- 1 m capped composite data were used for the estimation
- Hard boundary conditions were employed in the estimation
- Only samples from within individual mineralisation model domains were used to estimate blocks within those domains
- U_3O_8 (ppm) was estimated by Ordinary kriging ("OK"), using the variogram parameters presented in Figure 14-5 and Table 14-7 respectively
- Estimation of U_3O_8 (ppm) grade was completed in multiple passes using search criteria and sample numbers as summarised in Table 14-9
- Sub-block grades were assigned the grade of the parent block.

Table 14-9: Gwabi and Njame estimation parameters

Deposit	Variable	Interpolant	Estimation Pass	Ellipsoid Ranges			Number of Samples		
				Maximum	Intermediate	Minimum	Min	Max	Max per Hole
Gwabi	U_3O_8	OK	1	75	50	25	8	24	5
			2	150	120	50	8	24	5
			3	500	400	50	8	24	5
Njame	U_3O_8	OK	1	37.5	37.5	9.375	8	24	5
			2	75	60	18.75	8	24	5
			3	150	120	37.5	8	24	5
			4	500	400	50	8	24	5

14.4.7. Model validation

Block model validation conducted as part of the original estimation process included:

- Review of the block estimate and the composite data in cross-section, long-section and plan views
- Comparison of the mean grade of the estimate versus the mean grade, subdivided by estimation domain
- Comparison of composite grades and block model grades broken down into nothing and reduced level ("RL") zones.

AFRs validation indicates that the Mineral Resource model replicates the source input data well in regions of higher-density drilling. In the regions where the data density is lower, smoothing is evident, however, the estimates are considered appropriate.

SRK validated the grade estimates for Gwabi and Njame by conducting independent estimates using alternative estimation parameters and found that the results agreed very closely with those achieved in the AFR models. In the opinion of SRK, the AFR Mineral Resource models for the Gwabi and Njame deposits are reasonable representations of the global U_3O_8 Mineral Resources at the current level of sampling.

14.5. Muntanga, Dibbwi and Dibbwi East Mineral Resource Estimate updates

14.5.1. Mineralisation domain modelling

Mineralisation domains used for MRE within the Muntanga, Dibbwi and Dibbwi East deposits have been defined based on grade shells generated using a 100 ppm eU_3O_8 cut-off with an 80 ppm eU_3O_8 cut-off low-grade halo. The updated mineralisation domain models incorporate additional drill hole information and database QAQC conducted since the previous MREs were completed in 2023 for Muntanga, Dibbwi East and Dibbwi (SRK, 2023). 3D grade shells were generated using Leapfrog software predicated on equivalent uranium (eU_3O_8) grade data obtained from down-hole radiometric probing.

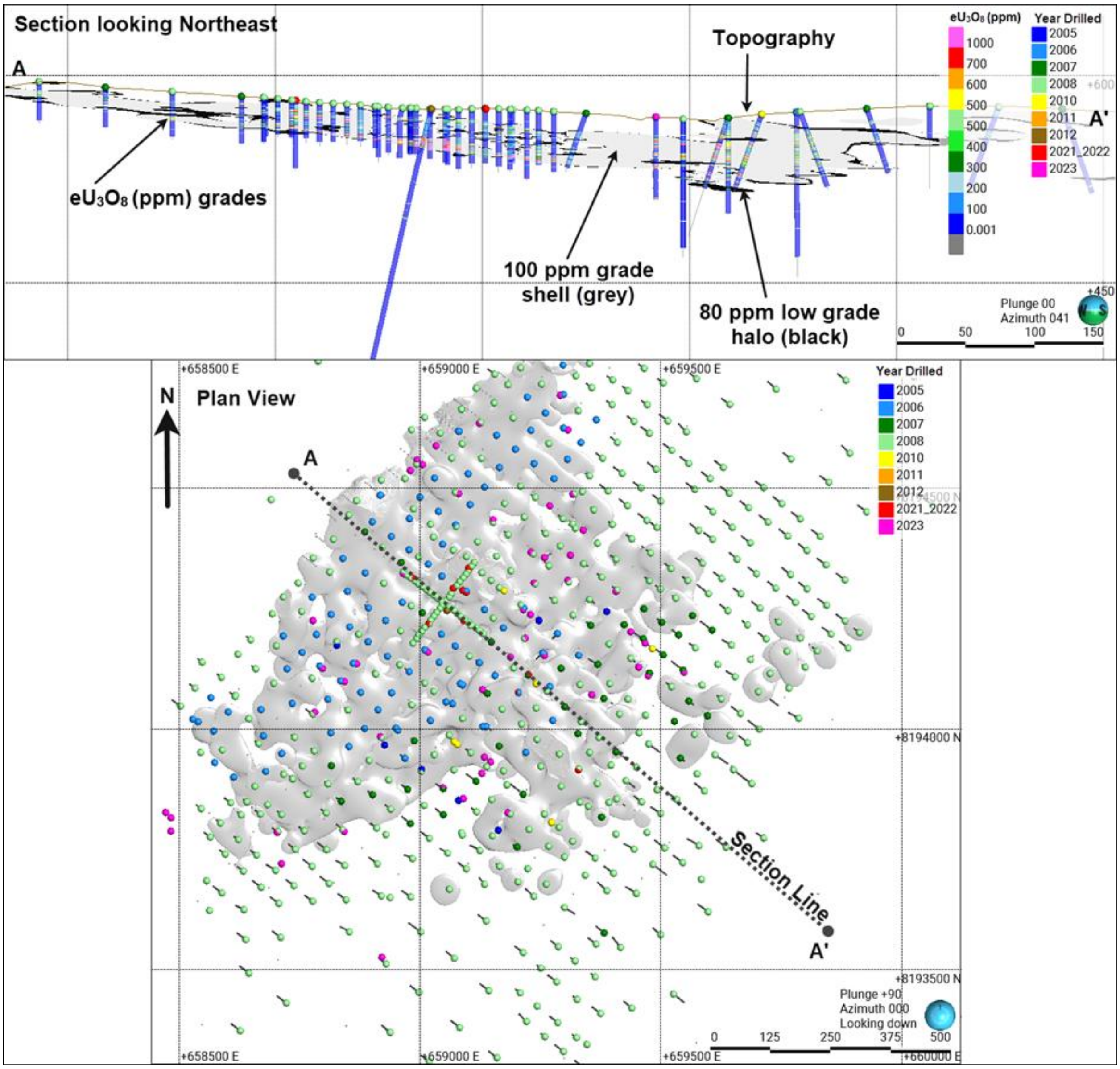

Figure 14-7: Muntanga deposit mineralisation domain model (Note: Drill hole collars are colour coded by drilling campaign year)

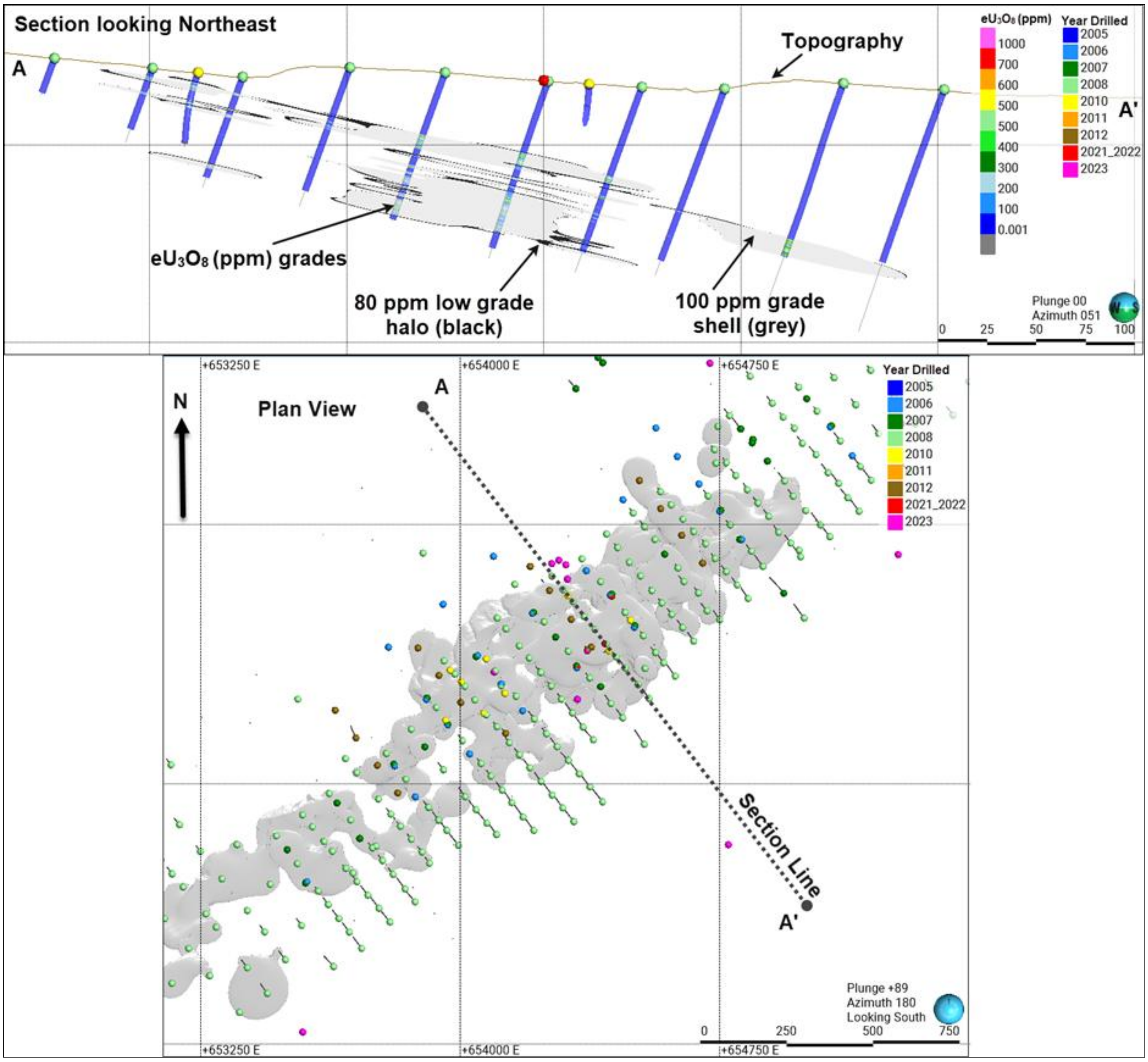

Figure 14-8: Dibbwi deposit mineralisation domain (Note: Drill hole collars are colour coded by drilling campaign year)

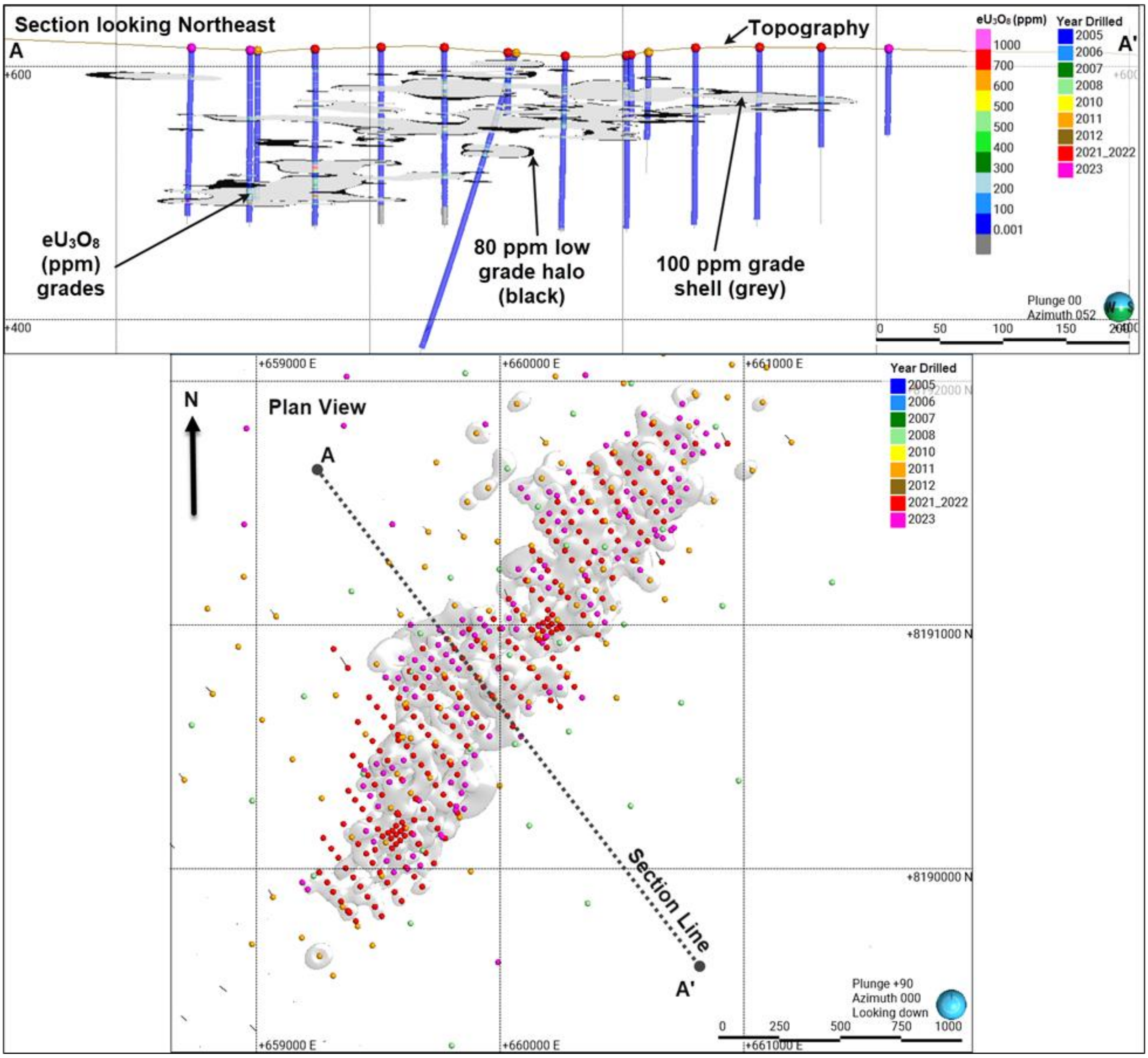

Figure 14-9: Dibbwi East deposit mineralisation domain (Note: Drill hole collars are colour-coded by drilling campaign year)

14.5.2.Radiometric-grade correlation

To facilitate a reliable conversion of down-hole radiometric probe data into equivalent uranium eU_3O_8, a deposit/probe-specific Radiometric-Grade ("Ra-Grade") correlation must be established. However, prior to developing a Ra-Grade correlation raw probe data must be adjusted to account for gamma signature attenuation associated with the logging environment, such as the size of the drill hole, fluid presence within the drill hole, casing/steel parameters and probe correction factors.

The Ra-Grade correlation was conducted by comparing geochemical sample assays to their corresponding probe data. Data was segregated into historical data comprised of down-hole gamma data predominately acquired by Denison from 2007 to 2012, and data collected by GoviEx during the 2021 to 2023 drilling campaigns.

Figure 14-10 to Figure 14-12 provide examples of drill hole profiles comparing assay results and radiometric probe profiles (preliminary eU_3O_8 ppm values) for intervals included within the correlation study.

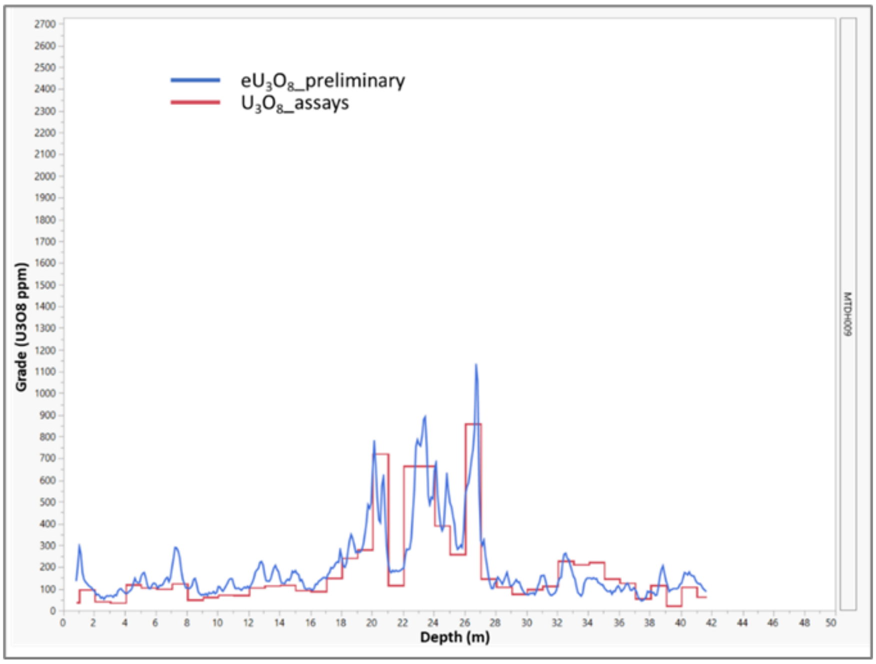

Figure 14-10: Muntanga deposit, drill hole MTDH009: Comparison of assay results and preliminary eU_3O_8 profiles

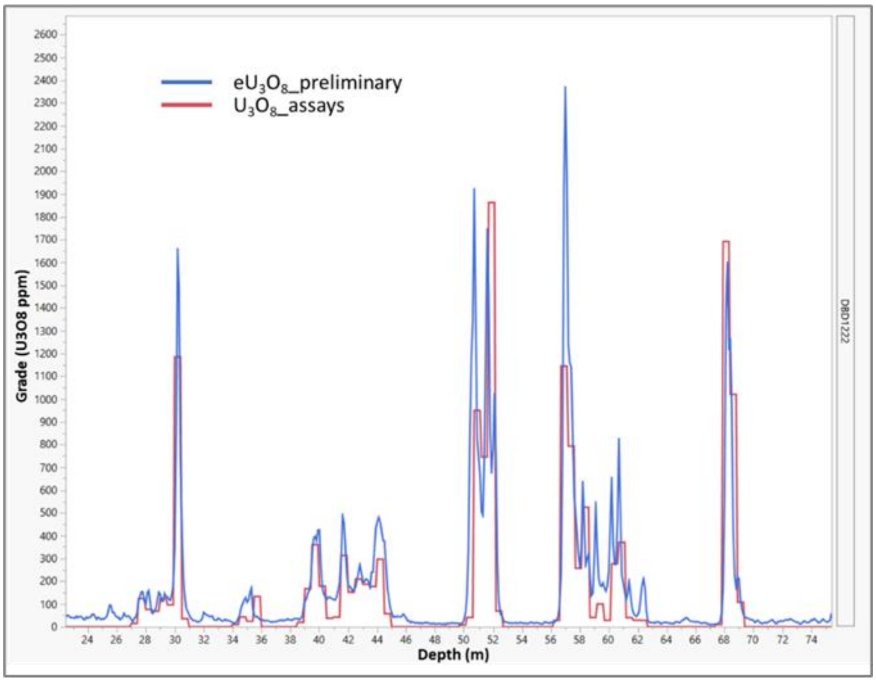

Figure 14-11: Dibbwi deposit, drill hole DBD1222: Comparison of assay results and preliminary eU_3O_8 profiles

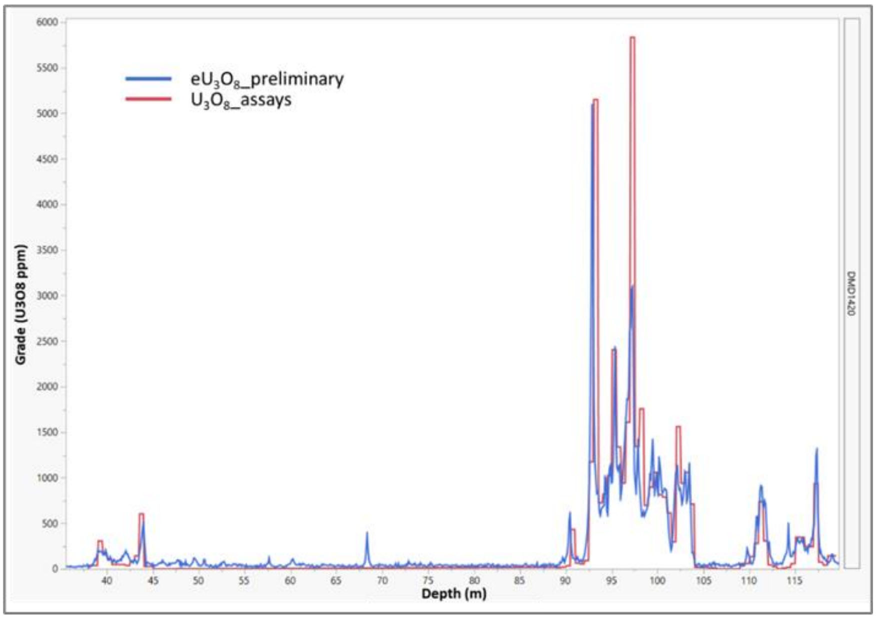

Figure 14-12: Dibbwi East deposit, drill hole DMD1420: Comparison of assay results and preliminary eU_3O_8 profiles

In the initial study, 76 mineralised intervals (Grade * thickness or "GT" intervals, expressed in units of ppm * m) from Muntanga-Dibbwi historical drill holes, 119 mineralised intervals from Dibbwi East historical drill holes, and 49 mineralised intervals from Dibbwi East 2021-2022 drill holes were selected for the study. The initial study results in Ra-Grade correlations established for the above data sets are provided in Figure 14-13 to Figure 14-15.

In 2024 the study was expanded to 254 mineralised intervals from 2023 drilling with results from all the Mineral Resource areas. Seven outliers were removed to improve the regression results. When analysing Muntanga (69 GTs), Dibbwi (20 GTs) and Dibbwi East (144 GTs) results in the impact on low Ra-grades (<100 ppm) tend to bias low by 7 % and at high Ra-grades (>5 000 ppm) tend to bias low by 10 %. Therefore, based on the 2023 analysis as shown in Figure 14-16, the Ra-grades generally seem to be reporting lower than analytical results in the order of 7 % to 10 %.

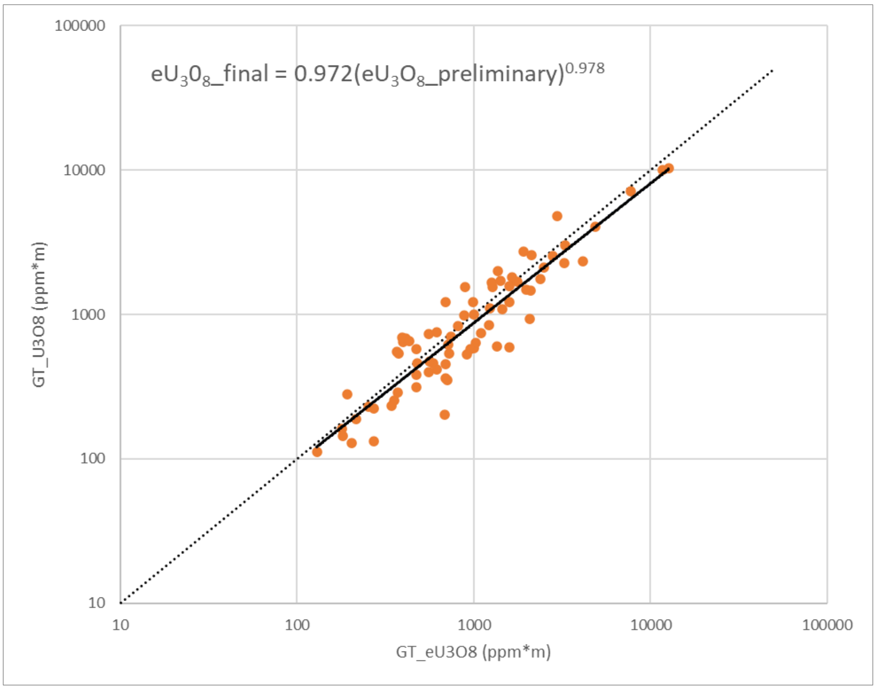

Figure 14-13: Muntanga-Dibbwi Ra-grade correlation (historical drill holes)

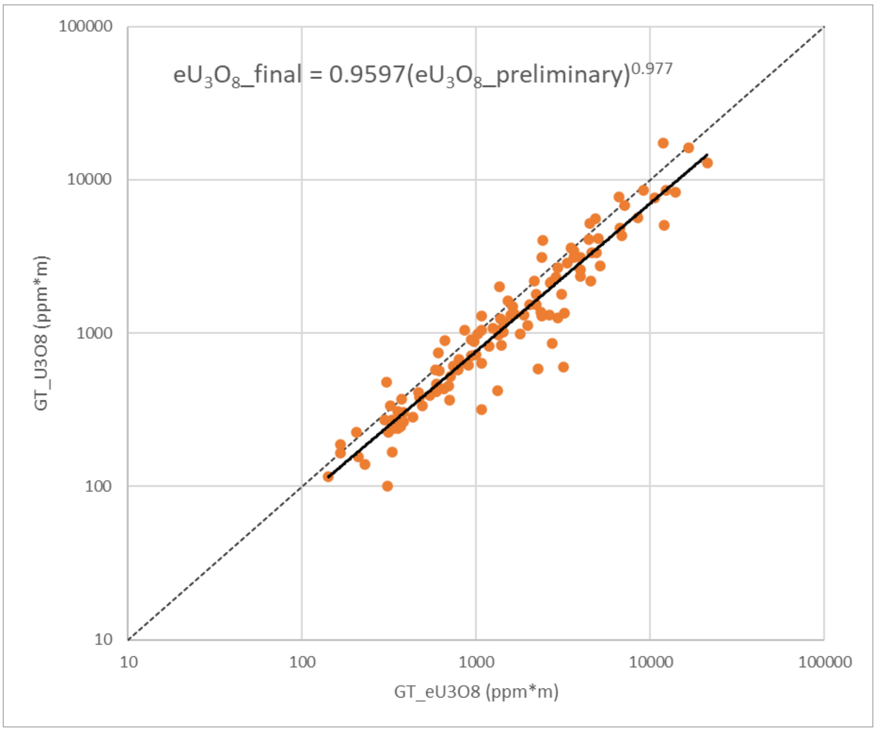

Figure 14-14: Dibbwi East Ra-grade correlation (historical drill hole)

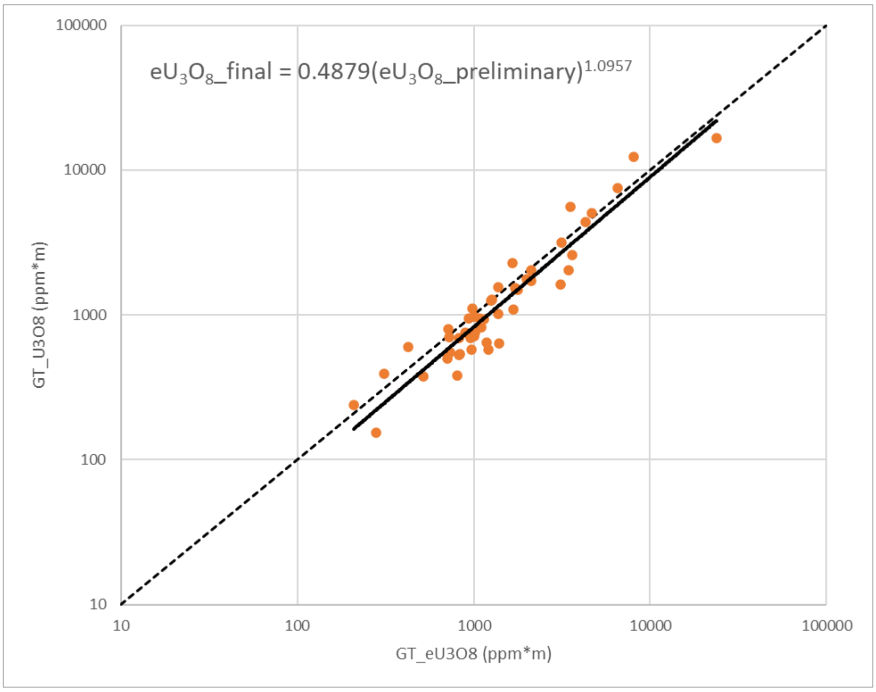

Figure 14-15: Dibbwi East Ra-grade correlation (2021 to 2022 drill holes)

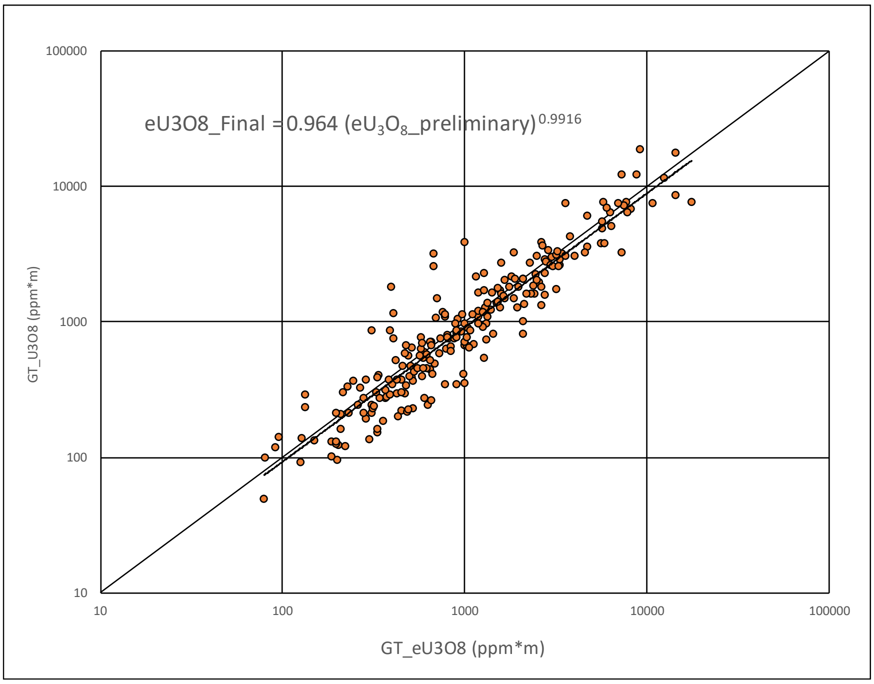

Figure 14-16: Muntanga, Dibbwi and Dibbwi East Ra-grade correlation (2023 drill holes)

14.5.3.Bulk density

A total of 450 bulk density measurements were being collected across the Muntanga, Dibbwi and Dibbwi East deposits. Summary statistics for these samples are provided in Figure 14-17. A global dry bulk density of 2.1 t/m^3 has been assigned for tonnage reporting for all three deposits. SRK noticed variations related to lithology and redox state. However, the individual sample populations are not significant and therefore SRK recommends that more density values be collected. A wax coating was used in 88 % of the volume displacement density determinations, taking the rock's porosity into account to prevent overstating the density. The CV of the density values is in the order of < 0.06. Therefore, the use of a mean density value is suitable for the current MRE.

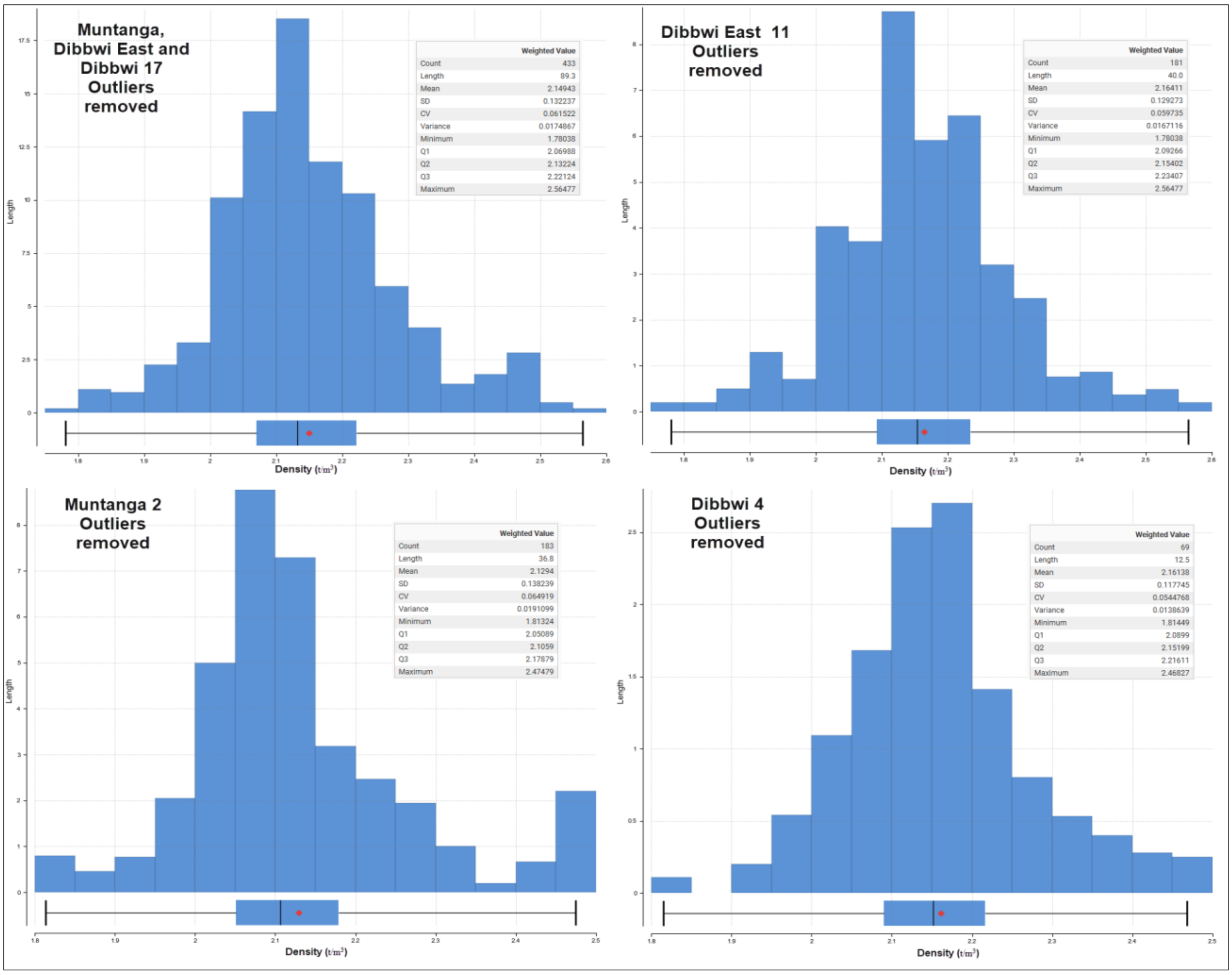

Figure 14-17: Dry bulk density

14.5.4.Compositing

Uranium grade data was composited to 1.0 m lengths within the grade shell boundaries, with all residual composites smaller than 0.5 m in length added to the adjacent composite interval. Assay samples were predominately collected using a 1.0 m sample length and eU_3O_8 data from down-hole radiometric probing is collected at 0.1 m intervals.

Summary statistics of drill hole uranium grade data by deposit, for both raw sample intervals and composited samples are provided in Table 14-10 and Table 14-11. Total proportions of uranium grade data based on down-hole radiometric data vary within each deposit but typically comprise the majority of the total grade data set (by drill hole mineralised length) for each deposit.

Table 14-10: Summary statistics (length-weighted) for raw sample interval uranium grade data (U_3O_8 ppm) by deposit

Deposit	Proportion of probe data [by length] [%]	Mean	Std Dev	Min	25th	Median	75th	Max
Muntanga[1]	68	98	89	1	70	85	99	2 583
Muntanga[2]	66	409	839	1	118	182	364	41 255
Dibbwi[1]	81	100	80	2	70	87	104	1 205
Dibbwi[2]	73	257	253	2	120	181	310	4 921
Dibbwi East[1]	81	109	157	1	74	87	105	9 099
Dibbwi East[2]	88	417	819	1	126	199	395	18 529

[1] 80 ppm low-grade halo shell external to 100 ppm grade shell
[2] 100 ppm grade shell

Table 14-11: Summary statistics for composited (uncapped) uranium grade data (U_3O_8 ppm) by deposit

Deposit	# of Composites	Mean	Std Dev	CV	Min	25th	Median	75th	Max
Muntanga[1]	1 968	98	74	0.75	11	77	85	96	1 390
Muntanga[2]	2 940	434	666	1.53	30	143	218	437	10 685
Dibbwi[1]	502	103	71	0.70	9	76	89	107	937
Dibbwi[2]	1 233	259	177	0.68	10	147	206	317	1 837
Dibbwi East[1]	1 732	115	121	1.05	17	82	90	109	2 920
Dibbwi East[2]	7 151	419	628	1.50	26	149	228	428	12 647

[1] 80 ppm low-grade halo shell external to 100 ppm grade shell
[2] 100 ppm grade shell

A sensitivity study was run to determine the effect of the inclusion or exclusion of minor intervals during the compositing process for the Muntanga deposit. The minor intervals affected reduced the U_3O_8 composites mean grade by 16 %. On investigation the majority of these minor intervals are associated with very thin mineralized horizons. The QP decided to exclude these minor intervals to prevent them negatively biasing the resource estimates. This was dealt with by adjusting the minimum coverage parameter in Leapfrog™ to 100 %.

14.5.5. Evaluation of outliers

Grade capping is a technique used to mitigate the potential effect that a small population of high-grade sample outliers can have during grade estimation. These high-grade samples are not considered to be representative of the general sample population and are therefore “capped” to a level that is more representative of the general data population. Although subjective, grade capping is a common industry practice when performing grade estimation for deposits that have significant grade variability.

Outlier analysis was conducted on the 1.0 m composited data for all deposits. Histograms and normal quantile plots were generated for each data population and used to assess appropriate grade capping thresholds. Composites were capped before grade estimation. A summary of grade capping thresholds and capped summary statistics are provided in Table 14-12.

Table 14-12: Grade capping thresholds and capped uranium grade (U_3O_8 ppm) summary statistics (by deposit)

Deposit	Cap value	Mean [uncapped]	Mean [capped]	Std Dev [capped]	CV [capped]
Muntanga[1]	240	98	91	37	0.41
Muntanga[2]	5 350	434	431	631	1.46
Dibbwi[1]	280	103	98	44	0.45
Dibbwi[2]	725	259	250	147	0.59
Dibbwi East[1]	340	115	107	55	0.52
Dibbwi East[2]	5 000	419	404	555	1.37

[1] 80 ppm low-grade halo shell external to 100 ppm grade shell
[2] 100 ppm grade shell

14.5.6. Variography

Grade continuity analysis of uranium mineralisation was conducted on capped composites for each deposit. Variogram analysis was conducted using Seequent’s Edge software. Variogram parameters used for grade interpolation are provided in Table 14-13.

Table 14-13: Variogram parameters

Deposit	LF Directions			Normalised Nugget	Structure 1					Structure 2				
					Normalised Sill	Structure	Range [m]			Normalised Sill	Structure	Range [m]		
	Dip	Dip Azimuth	Pitch				Major	Semi-major	Minor			Major	Semi-major	Minor
Muntanga	5	160	160	0.2	0.52	Spherical	18	15	3	0.28	Spherical	60	40	12
Dibbwi	13	137	72	0.3	0.41	Spherical	23	58	4	0.29	Spherical	90	85	6
Dibbwi East	4	181	163	0.2	0.54	Spherical	18	14	3	0.26	Spherical	100	85	5

14.5.7.Block model configuration

Block model configuration details are summarised in Table 14-14. A parent block size of 20 x 10 x 2.5 m was sub-blocked for volumetric reporting. Grade interpolation was conducted at the parent block size of 20 x 10 x 2.5 m.

Table 14-14: Block model configuration parameters

Deposit	Parameters	X [m]	Y [m]	Z [m]
Muntanga	Parent block size	20	10	2.5
	Sub-block size	1.25	1.25	0.3125
	Basepoint*	658 980	8 192 920	665
	Boundary size	1 700	1 610	255
	Rotation			323
Dibbwi	Parent block size	20	10	2.5
	Sub-block size	1.25	1.25	0.625
	Basepoint*	653 980	8 182 190	640
	Boundary size	4160	3 420	250
	Rotation			323
Dibbwi East	Parent block size	20	10	2.5
	Sub-block size	1.25	1.25	0.625
	Basepoint*	659 315	8 188 545	665
	Boundary size	3 560	2 760	255
	Rotation			323

*Coordinates specified in UTM WGS84 Zone 35S reference datum

14.5.8.Grade estimation

Estimates of uranium grade (U_3O_8 ppm) were interpolated into the block model using OK, and a multiple-pass estimation strategy with successively expanding search criteria in subsequent estimation passes. Outlier restrictions were used for the Muntanga and Dibbwi East deposits to mitigate the potential of over-estimation of grade due to the presence of a small number of high uranium-grade composites. A summary of the estimation parameters used for the Muntanga, Dibbwi and Dibbwi East deposits is provided in Table 14-15.

Table 14-15: MRE parameters

Deposit	Variable [ppm]	Interpolant	Estimation pass	Ellipsoid ranges			Number of samples			Outlier restriction	
				Maximum	Intermediate	Minimum	Min	Max	Max per Hole	Distance [% of Search]	Value threshold to Clamp
Muntanga	U_3O_8	OK	1	60	40	12	9	20	3	66	1 500
			2	90	60	12	9	20	3	44	1 500
			3	120	80	24	3	10	3	33	1 500
Dibbwi	U_3O_8	OK	1	90	85	10	9	20	3	N/A	N/A
			2	135	128	10	9	20	3		
			3	180	170	10	4	9	3		
			4	180	170	10	2	6	3		
Dibbwi East	U_3O_8	OK	1	100	85	10	9	20	3	60	1 000
			2	150	125	10	9	20	3	40	1 000
			3	200	170	10	4	9	3	30	1 000
			4	200	170	10	1	9	3	30	1 000

14.5.9.Model validation

Block model validation was conducted using multiple techniques including:

- Visual inspection of estimated block grades relative to composite grades
- Swath plot analysis of grade profiles between OK, inverse distance (“ID2’) and nearest-neighbour (“NN”) block estimates and
- Statistical comparison of global average MRE estimated block grades and declustered composite grades (NN).

Cross-sectional comparisons of interpolated block grades vs drill hole sample grade data for Muntanga, Dibbwi and Dibbwi East are provided in Figure 14-18 to Figure 14-20, respectively. A reasonable visual correlation between the block estimates and composite data can be observed.

Swath plot comparisons of interpolated U_3O_8 grades from the OK, ID2 and NN models for Muntanga, Dibbwi and Dibbwi East are provided in Figure 14-21 to Figure 14-22: Dibbwi deposit, swath plot comparison of U3O8 (ppm) grade for OK, ID2 and NN block model estimates
, respectively. A reasonable correlation between the OK, ID2 and NN estimates is observed on these plots, with the OK estimates showing slightly lower grade profiles for all three MREs. The lower grade profile seen in the OK estimate is associated with the secondary high-grade restrictions used in the estimation workflow (i.e., Muntanga and Dibbwi East) and the sample weighting scheme derived from the OK algorithm.

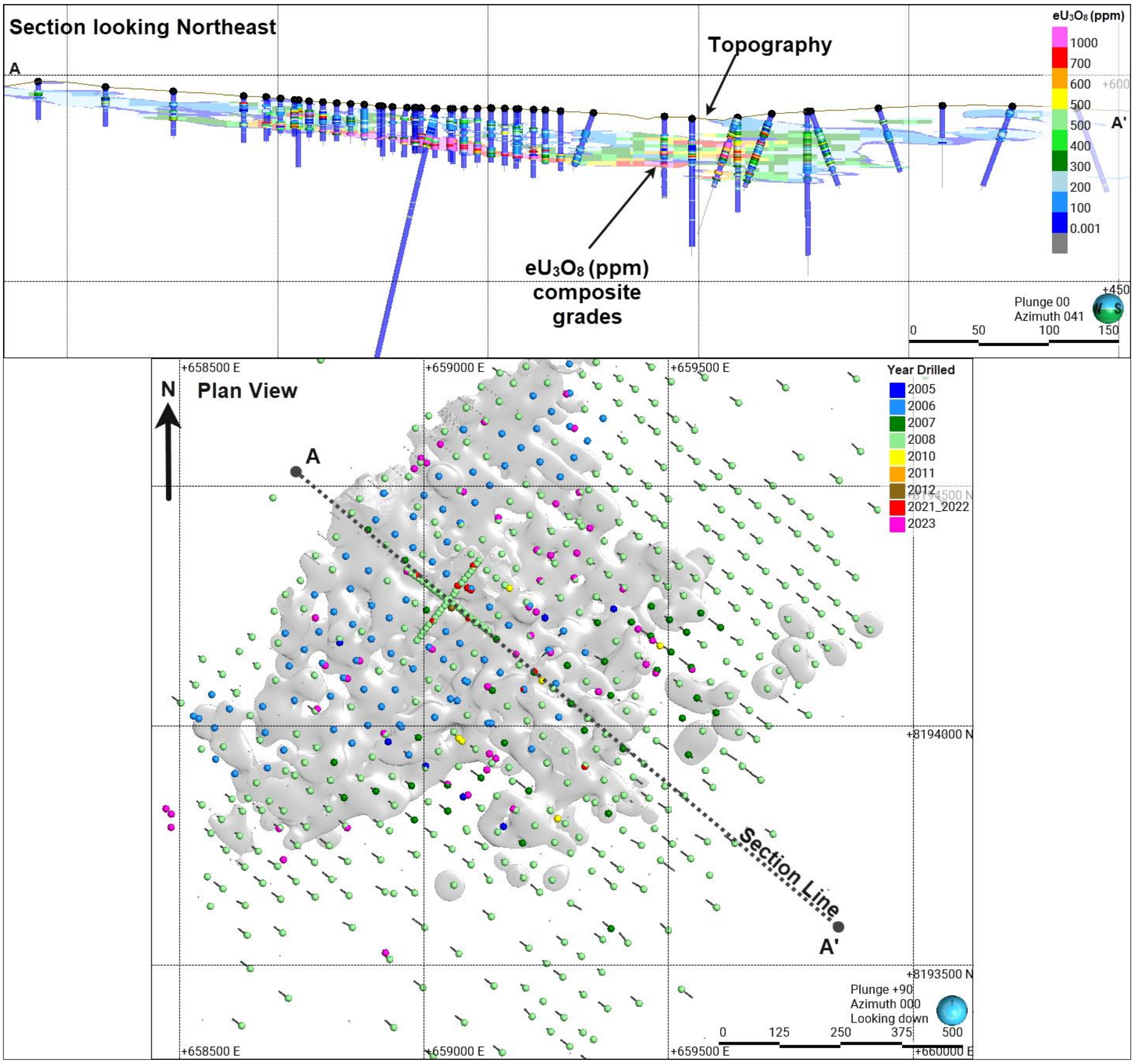

Figure 14-18: Muntanga deposit, cross-section comparison of interpolated U_3O_8 (ppm) grades vs eU_3O_8 (ppm) composites (looking Northeast)

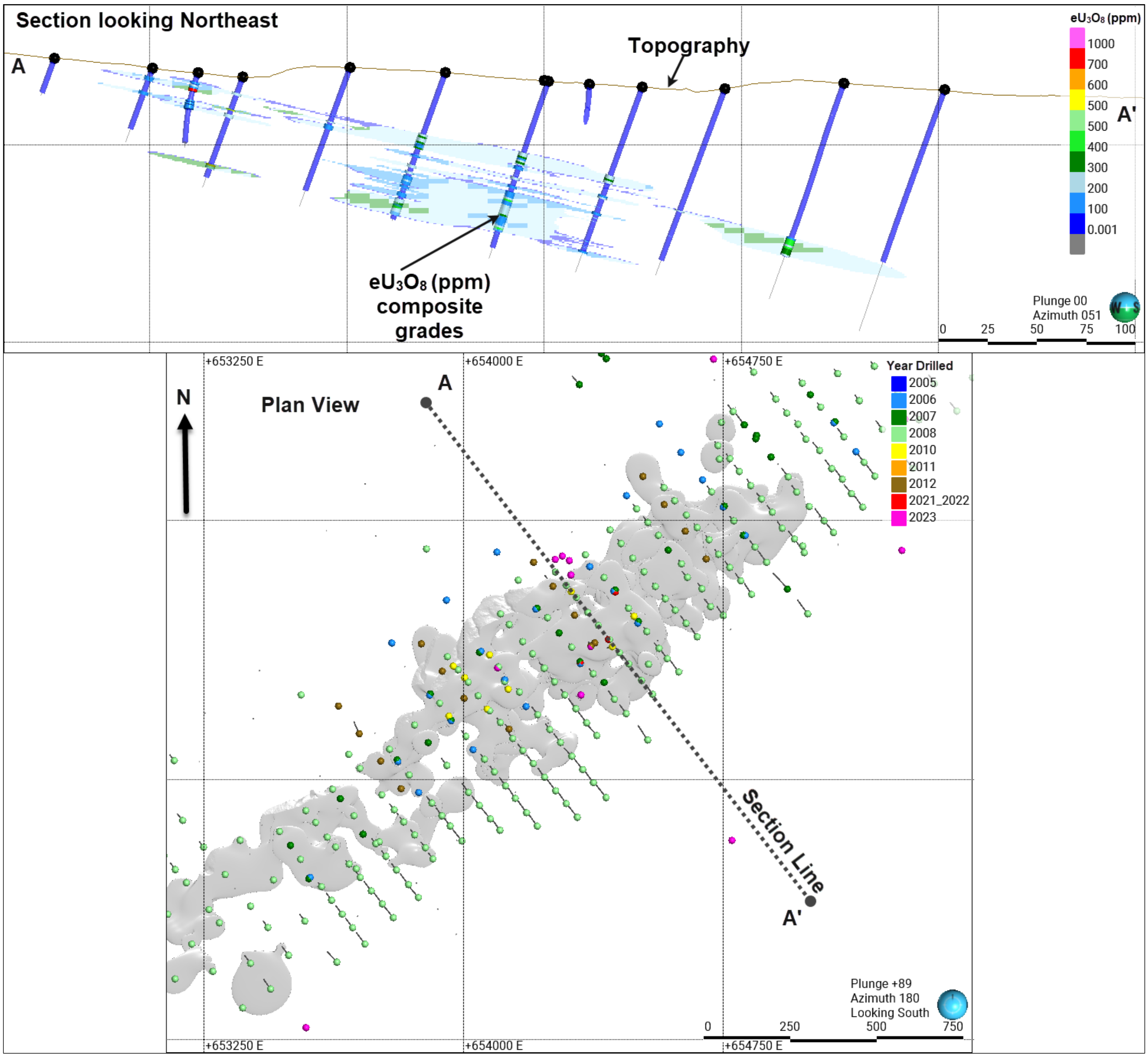

Figure 14-19: Dibbwi deposit, cross-section comparison of interpolated U_3O_8 (ppm) grades vs eU_3O_8 (ppm) composites (looking Northeast)

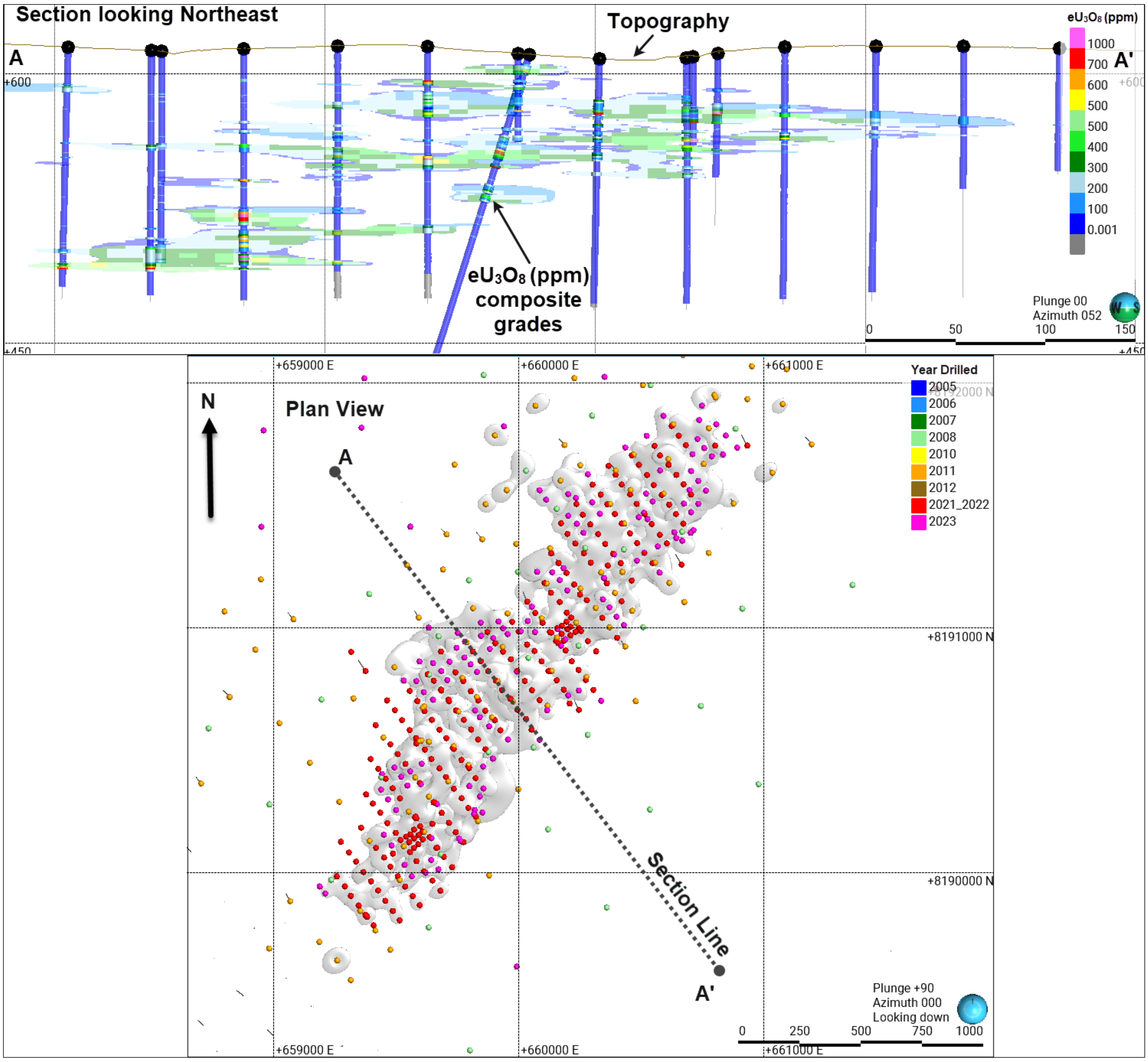

Figure 14-20: Dibbwi East deposit, cross-section comparison of interpolated U_3O_8 (ppm) grades vs eU_3O_8 (ppm) composites (looking Northeast)

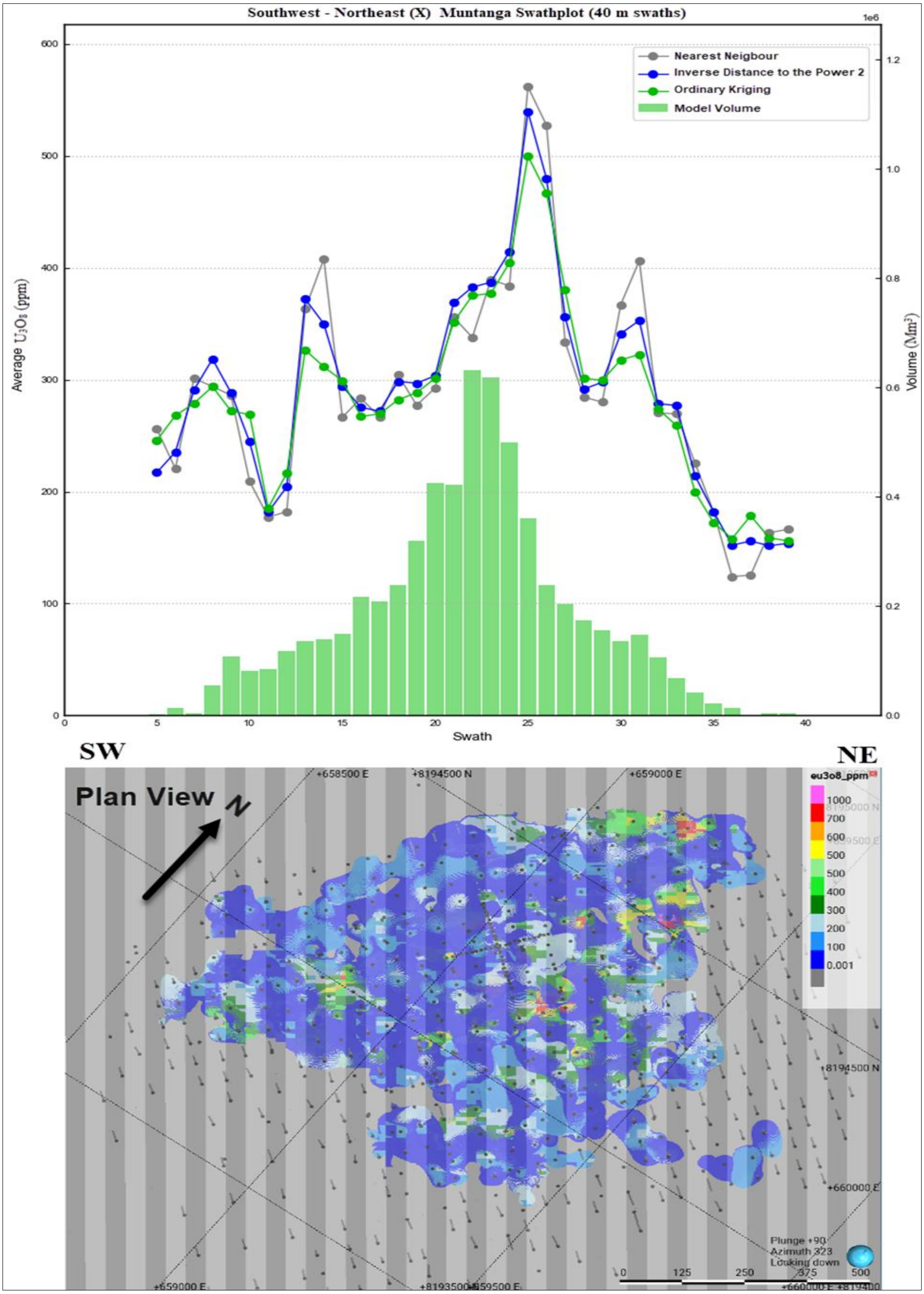

Figure 14-21: Muntanga deposit, swath plot comparison of U_3O_8 (ppm) grade for OK, ID2 and NN block model estimates

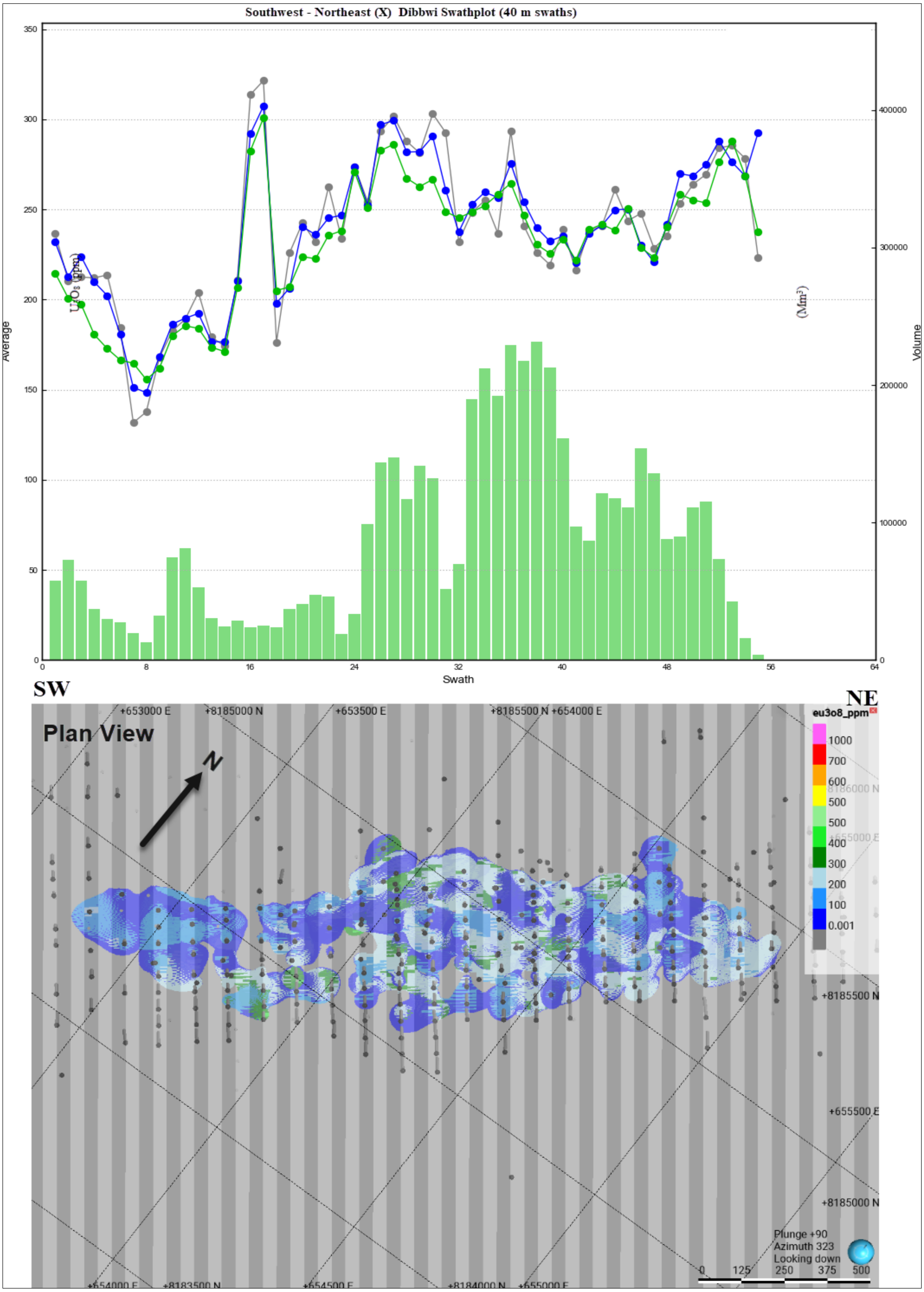

Figure 14-22: Dibbwi deposit, swath plot comparison of U_3O_8 (ppm) grade for OK, ID2 and NN block model estimates

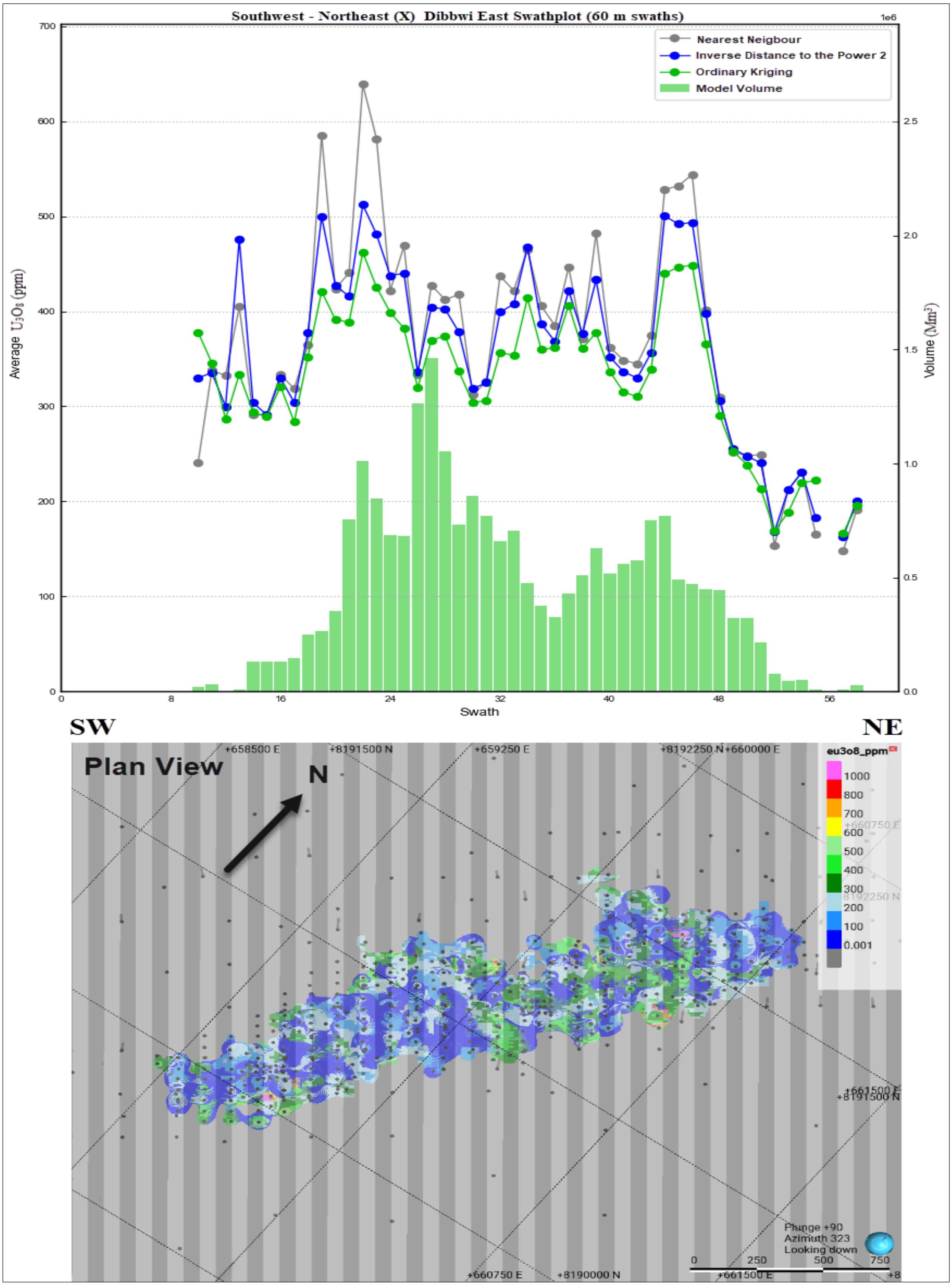

Figure 14-23: Dibbwi East deposit, swath plot comparison of U_3O_8 (ppm) grade for OK, ID2 and NN block model estimates

Figure 14-24 provides a comparison of the global average estimated U_3O_8 ppm grades between the OK, ID2 and NN models for each deposit. Generally, there is reasonable agreement between the estimates, however, as observed in the swath plot analysis the OK estimates produce slightly lower global average grades compared with the ID2 and NN models for all three deposits.

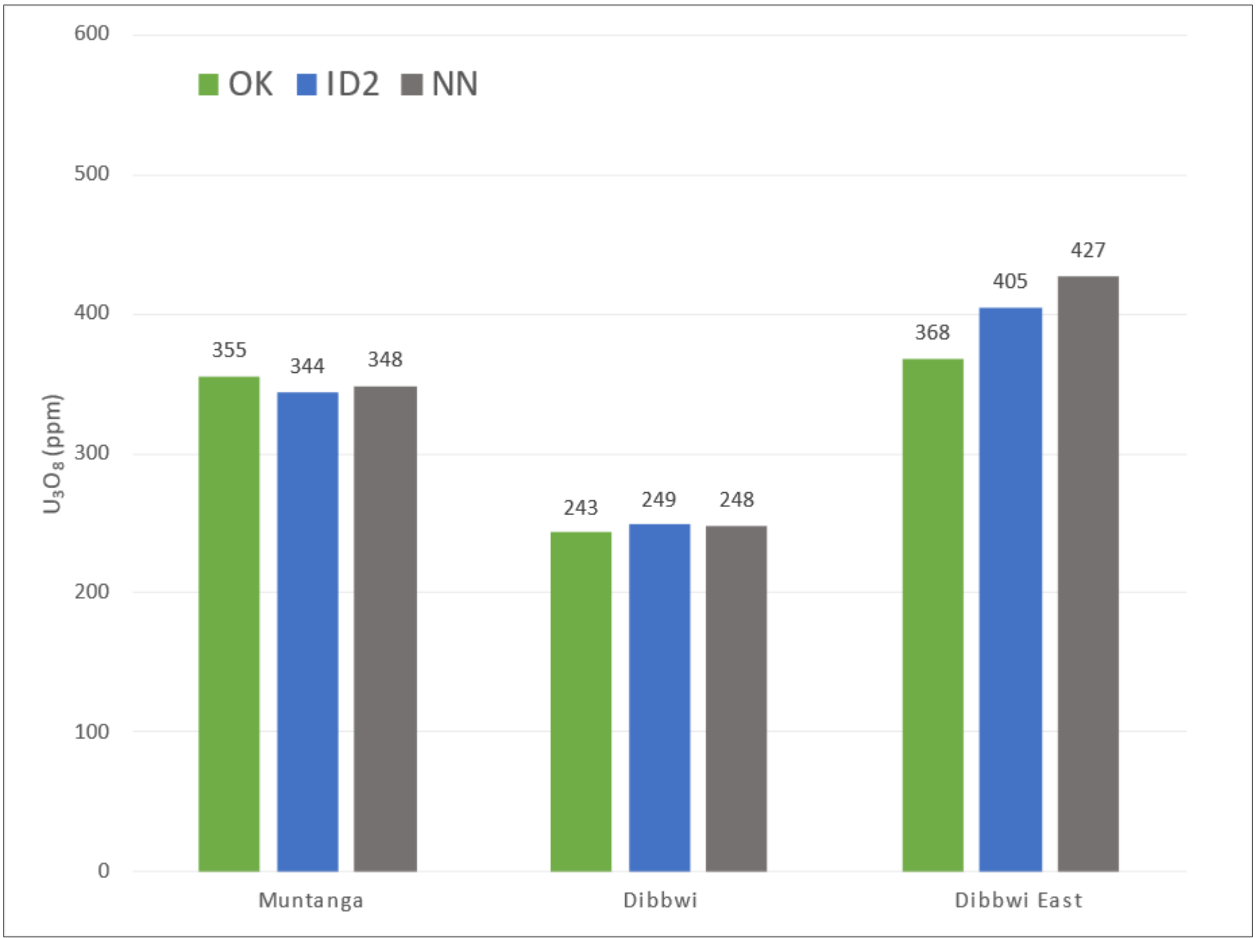

Figure 14-24: Global average grade (U_3O_8 ppm) comparison between OK, ID2 and NN estimates

14.6. Mineral Resource classification

Block model estimates for the Project were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) by André Marcel Deiss, Pr.Sci.Nat. P.Geo. an independent QP as defined in NI 43-101.

Mineral Resource classification is typically a subjective concept, and industry best practices suggest that Mineral Resource classification should consider both the confidence in the geological continuity of the mineralised structures, the quality and quantity of exploration data supporting the estimates and the geostatistical confidence in the tonnage and grade estimates. Appropriate classification criteria should aim at integrating these concepts to delineate semi-contiguous areas of similar Mineral Resource categories. Mr Deiss is satisfied that the mineralisation domain models honour the current geological understanding of the project area, and the location of the drill hole data and quality of uranium grade data are sufficiently reliable to support resource evaluation. Mineral Resource classification criteria considered the following components:

- Quality of the data used to support MRE
- Confidence in the interpretation of the mineralised zones
- Average drill hole spacing within the deposits and
- Estimation parameters including the number of drill holes and assay composites used to estimate a block.

The Gwabi and Njame deposits have been classified as Measured Mineral Resources where the drill hole spacing is less than 50 x 25 m. Indicated Mineral Resources have been classified where drill hole spacing is less than 50 x 50 m spacing, with all remaining Mineral Resources classified as Inferred Mineral Resources.

The Muntanga deposit has been classified as Indicated Mineral Resources where the average drill hole spacing is less than 50 m and blocks were estimated by pass 1 or pass 2 estimation parameters (Table 14-15). Inferred Mineral Resources were classified where the average drill hole spacing was less than 75 m. No Measured Mineral Resources were classified at the Muntanga deposit.

The Dibbwi and Dibbwi East deposits have been classified as Indicated Mineral Resources where the average drill hole spacing is less than 80 m and blocks were estimated by pass 1 estimation parameters (Table 14-15). Inferred Mineral Resources were classified where the average drill hole spacing was less than 150 m and blocks were estimated by pass 1 or pass 2 estimation parameters. No Measured Mineral Resources were classified at either the Dibbwi or Dibbwi East deposits.

Block model quantities and grade estimates were reviewed to determine the portions of the MRE having RPEEE from an open pit mine, based on parameters summarised in Table 14-16.

Table 14-16: Assumptions considered for conceptual open pit optimisation

Parameter	Value	Unit
U_3O_8 price	100	USD per pound
Mining cost	3.30	USD per tonne mined
Processing	9.00	USD per tonne of feed
General and administrative	1.50	USD per tonne of feed
Mining dilution	10	Percent (%)
Mining loss	5	%
Average pit slope	39	Degrees (°)
Process rate	3.5	Million tonnes feed per year
Royalty	5	% on U_3O_8 price
Recoveries		
Muntanga	93.0	%
Dibbwi	92.2	%
Dibbwi East	89.7	%
Njame	93.0	
Gwabi	73.1	%
In Situ COG	90*	Parts per million (ppm)

* A U_3O_8 90 ppm cut-off value was calculated for all pits, except for Gwabi where a 110 ppm cut-off was applied due to significantly lower demonstrated recoveries.

SRK considers that the blocks located within the conceptual pit envelopes show RPEEE and can be reported as a Mineral Resource.

Table 14-17: Mineral Resource statement*, the Project, Zambia, effective date, January 31, 2024.

Category	U_3O_8 cut-off [ppm]	Deposit	Tonnes [Mt]	U_3O_8 Grade [ppm]	U_3O_8 Metal [Mlb]
Measured	110	Gwabi	1.1	254	0.6
	90	Njame	2.5	358	2.0
Indicated	90	Muntanga	8.6	369	7.0
	90	Dibbwi	3.2	253	1.8
	90	Dibbwi East	31.3	372	25.7
	110	Gwabi	2.7	374	2.2
	90	Njame	1.0	306	0.7
Total M&I			**50.4**	**359**	**40.0**
Inferred	90	Muntanga	3.4	278	2.1
	90	Dibbwi	1.0	213	0.5
	90	Dibbwi East	7.1	252	3.9
	110	Gwabi	0.2	272	0.1
	90	Njame	1.1	329	0.8
Total inferred			**12.8**	**263**	**7.4**

*Notes

1. The effective date of the Mineral Resource statement is January 31, 2024. The QP for the estimate is André Marcel Deiss, Pr.Sci.Nat., P.Geo. an Associate Consultant (Resource Geology) of SRK Consulting (Canada) Inc.
2. Mineral Resources are prepared in accordance with CIM Definition Standards (CIM, 2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (CIM, 2019).
3. Mineral Resources are constrained within an optimised pit shell using a uranium price of USD100 /lb U_3O_8, mining costs of USD3.30 /t, processing costs of USD9.00 /t, additional mining costs of USD0.55 /t, G&A costs of USD1.50 /t, Transport costs of USD1.50 and a royalty of 5 %.
4. Mineral Resources are reported at a U_3O_8 COG within the optimised pit shell and are inclusive of Mineral Reserves.
5. Mineral Resources are inclusive of mineralisation in the 80 ppm halo but reported above the relevant cut-off and classed as Inferred Resources. This mineralisation represents approximately 5 % of the total Mineral Resources metal (Mlb).
6. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources will be converted into Mineral Reserves in the future.
7. All figures have been rounded to reflect the relative accuracy of the estimate.

14.7. Grade sensitivity analysis

The Mineral Resources of the Project are sensitive to the selection of the reporting COG. To illustrate this sensitivity, the block model quantities and grade estimates within the conceptual pit used to constrain the Mineral Resources are presented as grade tonnage curves in Figure 14-25 to Figure 14-29. Only classified Mineral Resources have been included in the grade tonnage curves.

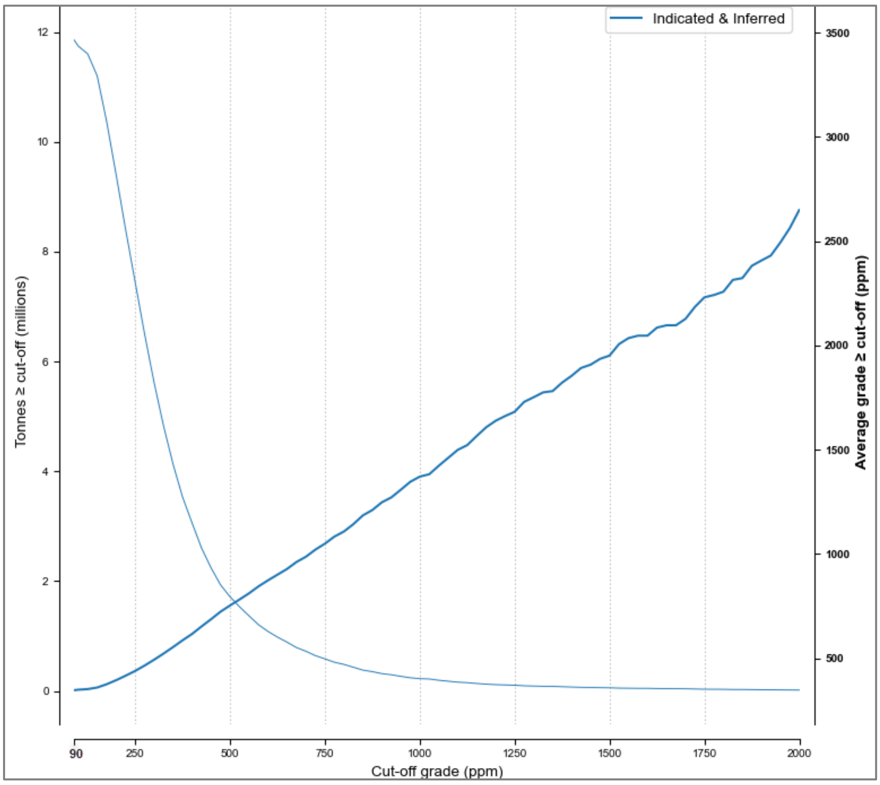

Figure 14-25: Grade (U_3O_8 ppm) tonnage curves for the deposit

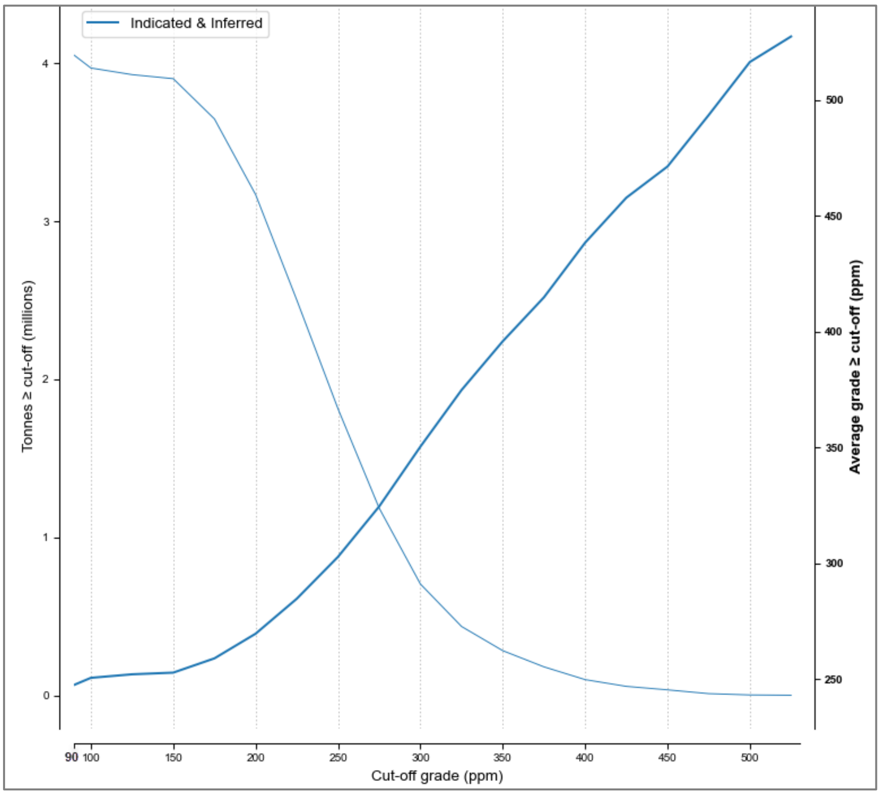

Figure 14-26: Grade (U_3O_8 ppm) tonnage curve for the Dibbwi deposit

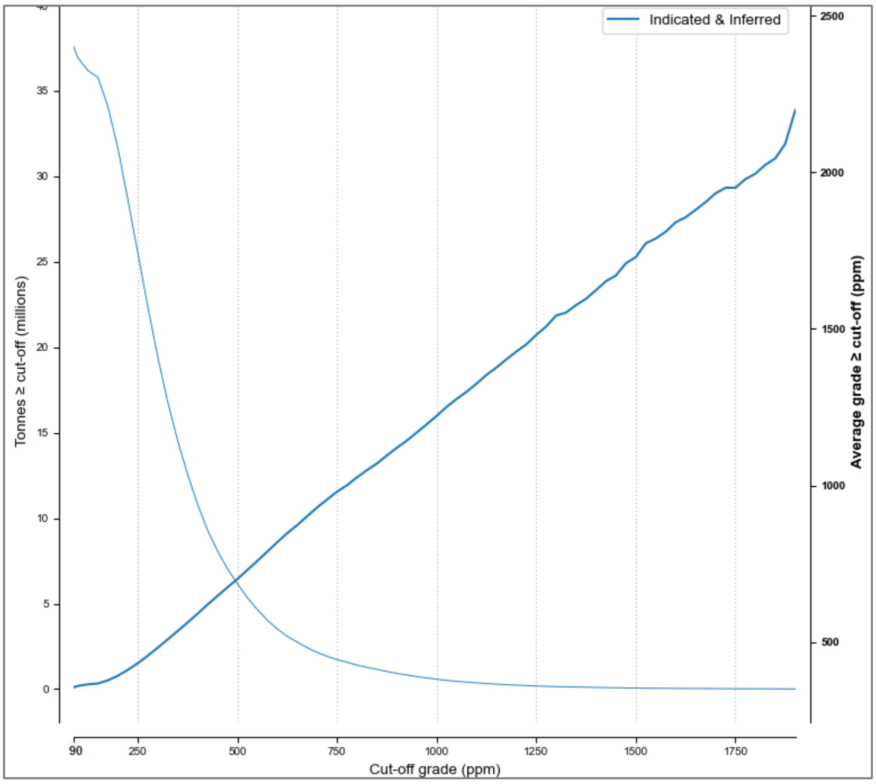

Figure 14-27: Grade (U_3O_8 ppm) tonnage curve for the Dibbwi East deposit

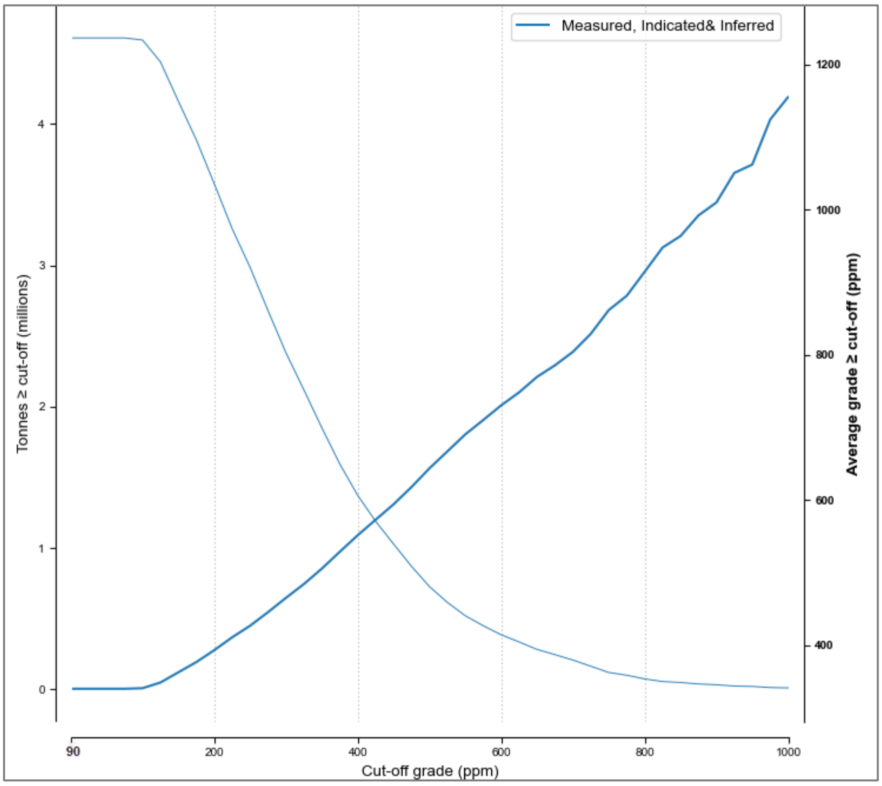

Figure 14-28: Grade (U_3O_8 ppm) tonnage curve for the Njame deposit

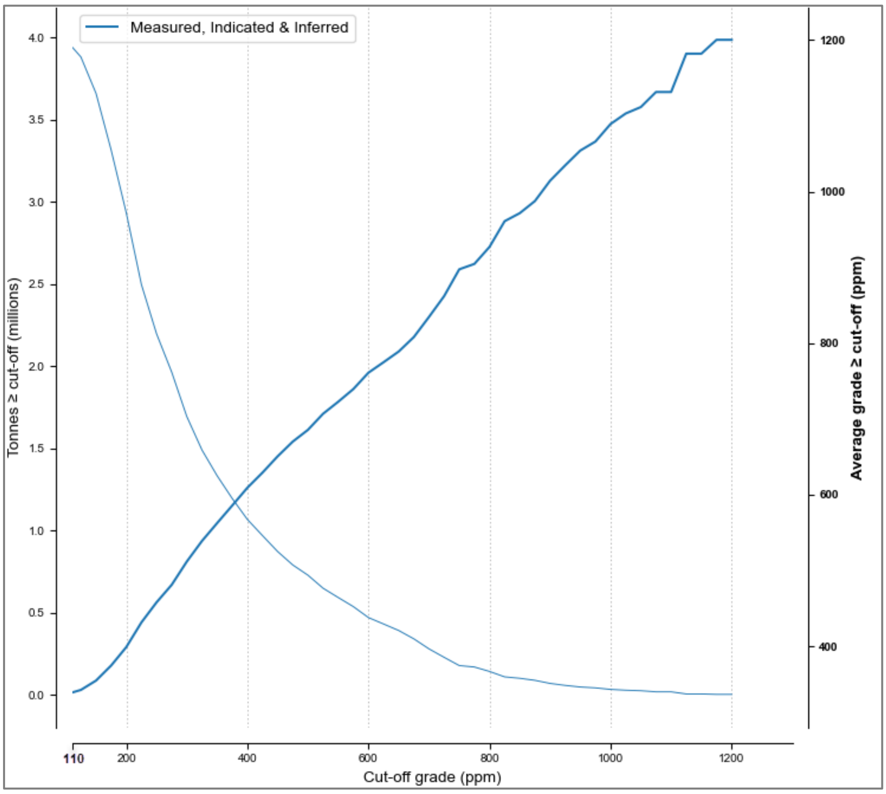

Figure 14-29: Grade (U_3O_8 ppm) tonnage curve for the Gwabi deposit

14.8. Previous Mineral Resource estimates

The previous MRE for the Project was reported by SRK with an effective date of March 31, 2023. A comparison of the current and previous MREs is provided in Table 14-18.

Table 14-18: Summary comparison of the current and previous Mineral Resource M&I estimate

M&I Mineral Resource	March 31, 2023 MRE	January 31, 2024 MRE
Tonnes Mt	42.6	50.4
U_3O_8 grade (ppm)	359	359
Contained U_3O_8 (Mlb)	33.7	40.0
COG (U_3O_8 ppm)	100	90
Inferred Mineral Resource	**March 31, 2023 MRE**	**January 31, 2024 MRE**
Tonnes Mt	15.0	12.8
U_3O_8 grade (ppm)	330	263
Contained U_3O_8 (Mlb)	10.9	7.4
COG (U_3O_8 ppm)	100	90

Comparison between the two MREs highlights the substantial conversion of previous Inferred resources to the Indicated category, particularly within the Dibbwi East deposit, based on the 2023 drill programme and analysis completed as part of the 2024 MRE update. The higher U_3O_8 commodity price and the improved recoveries have reduced the Mineral Resource cut-off values and consequently increased the total Mineral Resource tonnes and ounces by 10 % and 6 % respectively. The 2023 drilling has partially facilitated a 16 % increase in Indicated Mineral Resources relative to the previous MRE. The Gwabi higher cut-off results from a significant reduction in metallurgical recoveries. However, this higher cut-off does not impact the Mineral Resource results significantly once the appropriate level of relative accuracy has been applied.

14.9. Risks

The following risks were identified in the MRE work:

- The down-hole gamma contamination by radon conversion completed using Python scripting algorithms require continual refinement specifically at low radiometric value below eU_3O_8 100 ppm, as the contamination is not always clearly identifiable, and since the MRE cut-off is close to this limit material could be incorrectly classed as waste or resource. This risk has been mitigated by applying an applicable mineral resource category.
- There is a risk associated with the inclusion or exclusion of minor composite intervals related to very thin mineralized zones significantly thinner that the composite interval. For the Muntanga deposit there was a 6 % reduction in the mean composite grade by including these. Further sensitivity studies related to impact and dealing with these for mineralised domain modelling and / or estimation is recommended.

14.10. Recommendations

The following recommendations are provided to advance the understanding of the geology, mineralisation controls and Mineral Resources for the Project:

- Continue development of litho-structural models for the Project deposits, incorporating major fault interpretations within the vicinity of active mine areas or proposed future project infrastructure
- Continue infill drilling to support the conversion of Inferred to Indicated Mineral Resources
- Additional assay sampling to support further refinement of the Ra-grade correlation used to convert down-hole probe data into equivalent uranium grades
- Continue to assess for radon contamination within future drilling programmes and correct down-hole gamma signatures accordingly to mitigate the potential for over-estimation of grade due to radon
- Additional density analysis should be conducted on future drill programmes to refine tonnage estimates.

15. Mineral Reserve estimates

Detailed technical information provided under this item relates specifically to the Muntanga and Dibbwi East Mineral Reserve estimates completed for this Technical Report and based on the Mineral Resource models and estimates as reported in Item 14.

To conform with NI 43-101 standards, the Mineral Reserve estimate was derived from Measured and Indicated Mineral Resources only. The Measured and Indicated Mineral Resource estimates as listed in Table 14-3 and Table 14-29 are reported inclusive of the associated Mineral Reserve.

15.1. Methodology

The conversion of the Mineral Resource estimate to a Mineral Reserve estimate followed a conventional approach, commencing with open pit ("OP") optimisation techniques incorporating economic parameters and other modifying factors.

The ultimate (optimal) pit outlines (shells) were used to create practical and detailed OP designs accounting for the inclusion of batters, berms and haul roads.

These pit designs then provided the ore and waste mining inventories for a detailed production schedule that demonstrates viable OP mining. This schedule, which in turn provides the physical basis for cash flow modelling, is described in Section 22.

15.2. Mine modelling process and model unit sizes

The mine modelling process for OP projects includes three model types with specific block dimensions, normally increasing in physical size. The model types are outlined below in terms of the model basis, the purpose of the model and the use thereof. These included the following:

1. Geological model block dimensions
   - Based on: Exploration, sampling methodology, geostatistics, etc.
   - Purpose: MRE
   - Use: Basis for mining model
2. Mining block model or selective mining unit ("SMU") model units
   - Based on: SMU process and integration of modifying factors
   - Purpose: State selectivity and dilution (dilution applied where dilution will be incurred)
   - Use: Whittle, ore and waste definition, mining destination, input to schedule model (each SMU block can have only one material type, destination, grade, etc.)
3. Scheduling unit sizes
   - Based on: Physical layout and typical blast block dimensions (accumulation of SMUs)
   - Purpose: Simulate blast blocks and loader movement and interaction (once started, a schedule block must be completed to simulate loader movement during mining)
   - Use: Deswik,Xpac, Xpac Solutions or Studio OP, LOM schedule, and Mineral Reserve estimate.

The process progression is graphically represented in Figure 15-1.

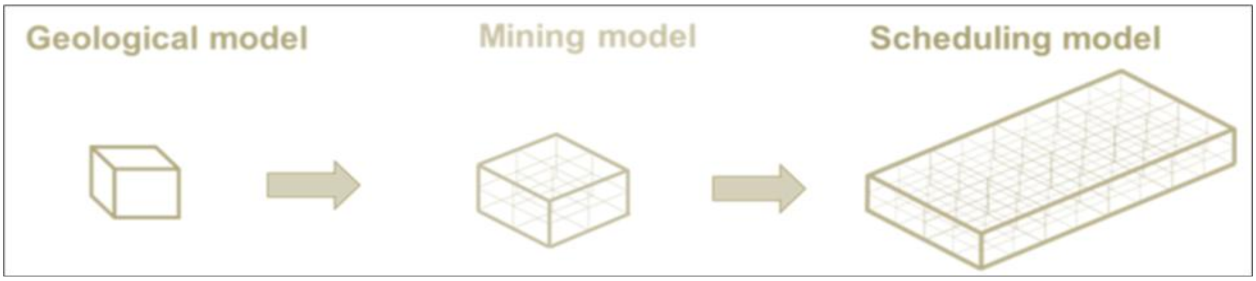

Figure 15-1: Mine modelling unit sizes

15.2.1. Mineral Resource models (Geological models)

Separate geological models for Muntanga and Dibbwi East were supplied by SRK on February 27, 2024. The supplied models shown in Table 15-1 formed the basis of the SMU selection.

Table 15-1: Mineral Resource models supplied

Description	Dibbwi East	Muntanga
Supplied Model name	*2023_Dibbwi East_20x10x2.5m_final_27022024*	*2023_muntanga_20x10x2.5m_final_27022024*
Parent block size	20(**X**) x 10(**Y**) x 2.5(**Z**)	20(**X**) x 10(**Y**) x 2.5(**Z**)
Minimum block size	1.25(**X**) x 1.25(**Y**) x 0.625(**Z**)	1.25(**X**) x 1.25(**Y**) x 0.3125(**Z**)
Rotation	-37°	-37°

No geological losses and background dilution were specified.

15.2.1.1. Muntanga Mineral Resource model

The Muntanga ore body has a gentle dip with reasonable thickness and is less structurally complex with multiple overlaying ore bodies and internal waste with a consistent economic grade continuity. It consists of Indicated material (Class 2) and Inferred material (Class 3) surrounded by waste and unclassified material. Figure 15-2 depicts a cross-section of the Mineral Resource model for Muntanga as received.

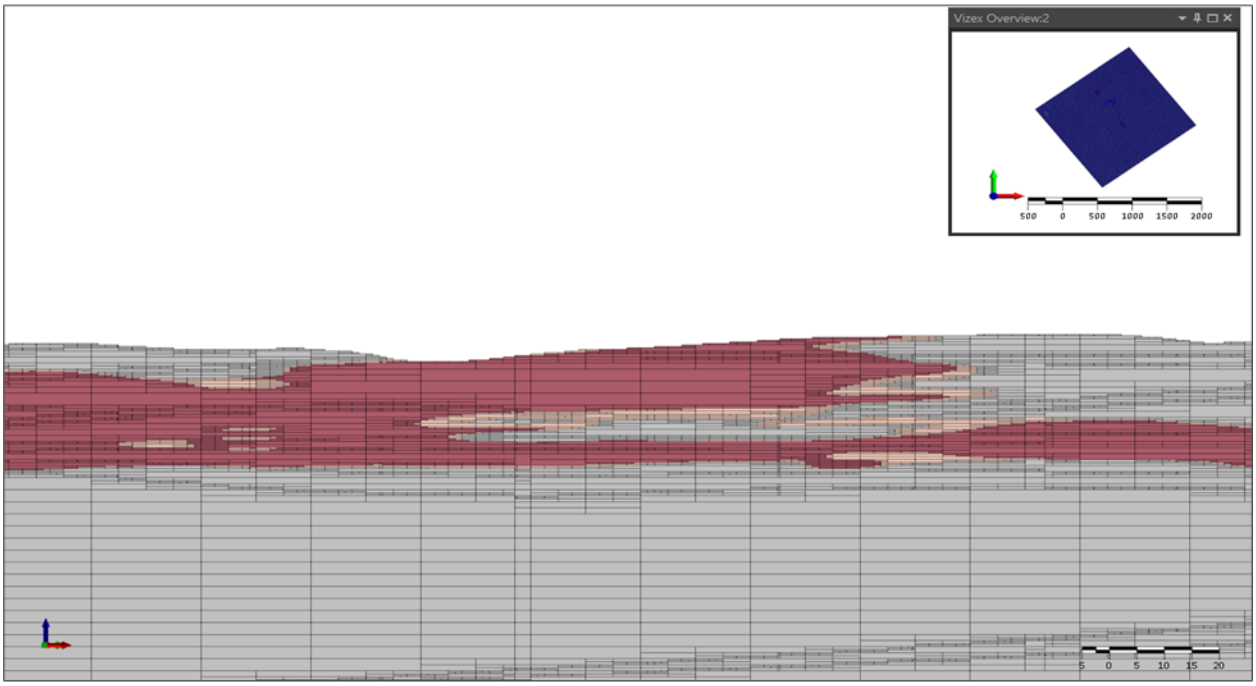

Figure 15-2: Muntanga Mineral Resource model cross section

15.2.1.2. Dibbwi East Mineral Resource model

Dibbwi East is flat with multiple overlaying ore bodies with a highly variable economic grade continuity. It consists of Measured material (Class 1), Indicated material (Class 2) and Inferred material (Class 3) surrounded by waste and unclassified material. Figure 15-3 depicts a cross-section of the Mineral Resource model for Dibbwi East as received.

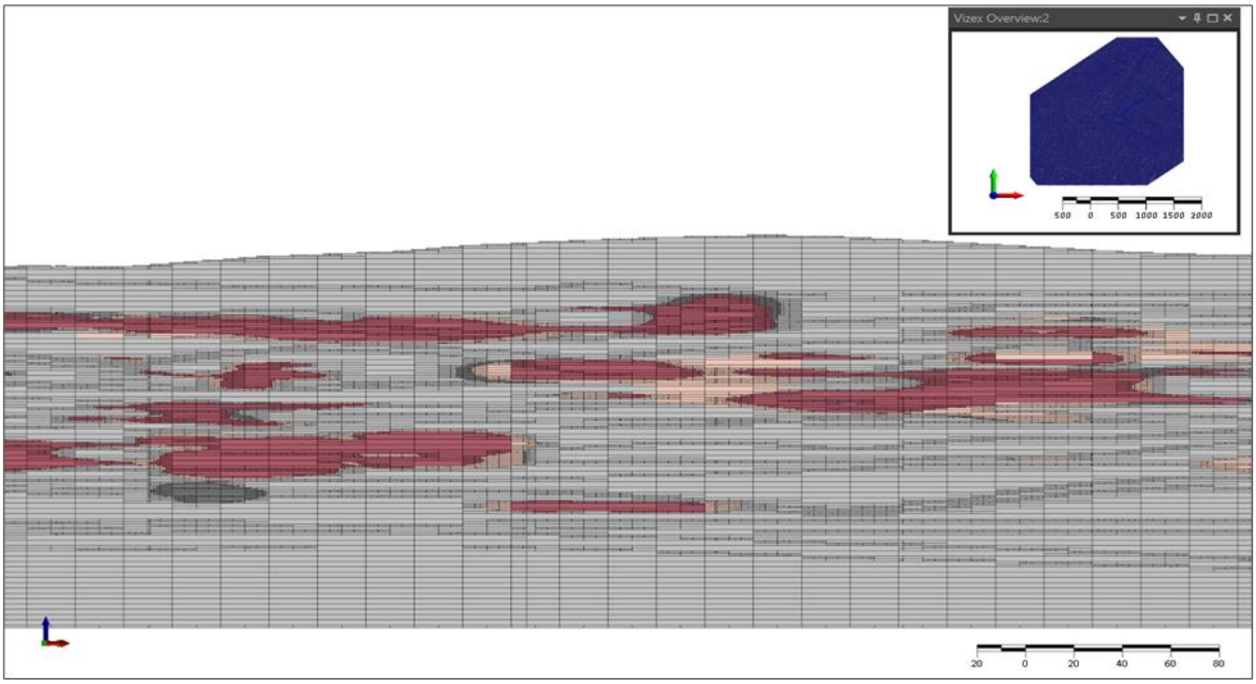

Figure 15-3: Dibbwi East Mineral Resource model cross section

15.3. Open pit mining-related modifying factors

15.3.1. Selective mining units

Dilution is defined as the waste material added during the mining process. Site-specific dilutions are added to the in situ Mineral Resources to define the quality of the practically mineable unit. Dilution reduces the overall grade of the practically mineable cut when compared to the original geology by adding material adjacent to the mineralised material. The tonnage containing mineralised material, therefore, increases relative to the Mineral Resources, but at a lower grade. COGs used as the basis for the ore classification process are applied to the SMU diluted grade. For the Project study, the dilution was estimated based on the SMU process.

The SMU is defined as the smallest unit that can be mined selectively. The outcome of the investigation aimed to determine an appropriate SMU for an informed decision on a modelling strategy that most accurately estimates actual practice and lowers the overall mining risk. Factors considered to determine a realistic SMU included:

- Size of the mining equipment selected and bucket size (not just applying digging reach)
- Loader configuration (FEL, excavator, or shovel)
- Selectivity in relation to other modifying factors such as mining losses
- The structural complexity of the ore body in terms of dip, thickness, and structural continuity
- Ore block continuity and the way it was modelled
- The mining production rate in relation to pit room and the opportunity to mine selectively
- Degree of continuity above the COG
- Mining strategy consisting of blending (in-pit blending versus stockpile blending, product grade, and contaminants)
- Actual and historic modifying factor performance
- Benchmarking to similar operations.

The selection of a large SMU in a structurally complex ore body will result in greater internal and external dilution. COGs must be placed on SMUs and not sample (or Mineral Resource) grades, as SMU grades represent fully diluted and practically mineable estimates.

The creation of a SMU model involves combining (or re-blocking) any number of geological blocks into a single SMU block for a single material/ rock type, quality, density and Mineral Resource class. This approach is appropriate in this context because the SMU is the smallest unit that can be mined selectively. Several permutations exist including diluting waste rock-type blocks with waste blocks of another rock type, waste blocks with ore blocks; and ore rock-type blocks with ore blocks of a different rock type.

The Mineral Resource classification of the SMU blocks was kept and carried over to the SMU model. A downgrade classification methodology was followed, in terms of which if Measured and Indicated Mineral Resources exist in a single SMU block, the block was re-allocated to a block containing Indicated Mineral Resources; and if Indicated and Inferred Mineral Resource existed in a single SMU block, the block was reallocated to a block containing Inferred Mineral Resource. This methodology does not aim to re-classify Mineral Resources but to ensure that the Indicated Mineral Resources are not converted into a Proven Mineral Reserve and that the Inferred Mineral Resource is not included as part of the Mineral Reserve.

Because selectivity beyond the SMU is not possible (as per the SMU definition), each SMU can only contribute to a single potential Mineral Reserve class.

On the completion of each SMU scenario, a reconciliation was done between the Mineral Resource model and the SMU model to ensure that the stated Mineral Resource contained in the geological and Mineral Resource model remains the same in the SMU model, apart from the addition of dilution material.

15.3.1.1. Mining loss

The estimation of mining loss requires an understanding of the MRE methodology, structural geology, blasting practice, and loading fleet. The dip, strike, width and length of the zones within the deposit are the most significant considerations for mining loss and mining dilution. In addition to the absolute values, the variability in geometry has a significant influence on the efficiency of ore mining. The major contributors to mining losses usually are loading losses (based on ore included in the SMU that is mistakenly loaded as waste), blasting losses, material handling losses during hauling, and ore incorrectly hauled to a waste destination.

Ore and waste blasting characteristics influence the decision to bulk blast ore and waste as a unit or blast and load on a selective basis. The selected loading equipment capabilities must match the blasted rockpile profile and digability.

The equipment bucket size and direction of mining relative to the deposit geometry and blast displacement influence the mining loss and dilution. Mining loss and dilution estimates influence revenue, costs, Mineral Reserve, and the project's NPV. A mining loss of 5 % was applied, based on benchmark information for similarly sized operations.

15.3.1.2. Geological loss

The geological loss is an indication of MRE error, modelling inaccuracies or structural complexity of the deposit. The confidence level of the project study, the complexity of the deposit and the rigidness of the topography normally influence the assumption of geological loss. No geological loss was applied, based on guidance from the Mineral Resource QP.

15.4. Open pit selective mining unit model scenarios

15.4.1. Approach

The final supplied Mineral Resource models shown in Table 15-1 were used as the basis for the SMU selection. Based on the general geology, production rate and loading equipment applied at the Muntanga and Dibbwi East OPs, and to identify potential selectivity alternatives, four SMU scenarios were identified and modelled for the Project:

- Option 1: 5 m x 5 m x 2.5 m
- Option 2: 10 m x 10 m x 2.5 m
- Option 3: 5 m x 5 m x 5 m
- Option 4: 10 m x 10 m x 5 m.

Based on the geometry of the pit and ore body, it must be noted that dilution and mining losses related to loading activities are interrelated and will be encountered in the ore body. Generally, and where practically possible, the loading direction will be perpendicular to the transition zone from ore to waste on each production bench.

15.4.2. Reconciliation

The purpose of the reconciliation process was to verify the SMU technical process and ensure that the Mineral Resource contained in the SMU was not altered relative to the Mineral Resource contained in the geological model. A global reconciliation was conducted on all the models stated in Table 15-1.

15.4.2.1. Muntanga reconciliation

Table 15-2 shows the reconciliation for the different SMU sizes considered. The reconciliation shows an acceptable variance in Mineral Resources in the geological model relative to the Mineral Resources contained in the different SMU models. The geological model applied is 2023_muntanga_20x10x2.5m_final_27022024.

Table 15-2: Muntanga SMU reconciliation

SMU size	Description	Unit	Mineral Resource class	Mineral Resource class contained in Geological model	Mineral Resource class contained in SMU model	Variance [%]
5 x 5 x 2.5	Tonnes	Mt	Measured	-	-	0
			Indicated	7.6	7.6	0
			Inferred	8.2	8.2	0
			Total	**15.8**	**15.8**	**0**
	U_3O_8 grade	ppm	Measured			0
			Indicated	358	358	0
			Inferred	219	219	0
			Total	**286**	**286**	**0**
10 x 10 x 2.5	Tonnes	Mt	Measured	-	-	0
			Indicated	7.6	7.6	0
			Inferred	8.2	8.2	0
			Total	**15.8**	**15.8**	**0**
	U_3O_8 grade	ppm	Measured			0
			Indicated	358	358	0
			Inferred	219	219	0
			Total	**286**	**286**	**0**
5 x 5 x 5	Tonnes	Mt	Measured	-	-	0
			Indicated	7.6	7.6	0
			Inferred	8.2	8.2	0
			Total	**15.8**	**15.8**	**0**
	U_3O_8 grade	ppm	Measured			0
			Indicated	358	358	0
			Inferred	219	219	0
			Total	**286**	**286**	**0**
10 x 10 x 5	Tonnes	Mt	Measured	-	-	0
			Indicated	7.6	7.6	0
			Inferred	8.2	8.2	0
			Total	**15.8**	**15.8**	**0**
	U_3O_8 grade	ppm	Measured			0
			Indicated	358	358	0
			Inferred	219	219	0
			Total	**286**	**286**	**0**

*Reconciliation did not consider a COG

15.4.2.2. Dibbwi East reconciliation

Table 15-3 shows the Dibbwi East reconciliation for the different SMU sizes considered. The reconciliation shows an acceptable variance in Mineral Resources in the geological model relative to the Mineral Resources contained in the different SMU models. The geological model applied is 2023_Dibbwi East_20x10x2.5m_final_27022024.

Table 15-3: Dibbwi East SMU reconciliation

SMU size	Description	Unit	Mineral Resource Class	Mineral Resource Class contained in Geological model	Mineral Resource Class contained in SMU model	Variance [%]
5 x 5 x 2.5	Tonnes	Mt	Measured	26.8	26.8	0
			Indicated	25.0	25.0	0
			Inferred	**51.8**	**51.8**	0
			Total	-	-	**0**
	U_3O_8 grade	ppm	Measured	372	372	0
			Indicated	222	222	0
			Inferred	**299**	**299**	0
			Total	-	-	**0**
10 x 10 x 2.5	Tonnes	Mt	Measured	26.8	26.8	0
			Indicated	25.0	25.0	0
			Inferred	**51.8**	**51.8**	0
			Total	-	-	**0**
	U_3O_8 grade	ppm	Measured	372	372	0
			Indicated	222	222	0
			Inferred	**299**	**299**	0
			Total	-	-	**0**
5 x 5 x 5	Tonnes	Mt	Measured	26.8	26.8	0
			Indicated	25.0	25.0	0
			Inferred	**51.8**	**51.8**	0
			Total	-	-	**0**
	U_3O_8 grade	ppm	Measured	372	372	0
			Indicated	222	222	0
			Inferred	**299**	**299**	0
			Total	-	-	**0**
10 x 10 x 5	Tonnes	Mt	Measured	26.8	26.8	0
			Indicated	25.0	25.0	0
			Inferred	**51.8**	**51.8**	0
			Total	-	-	**0**
	U_3O_8 grade	ppm	Measured	372	372	0
			Indicated	222	222	0
			Inferred	**299**	**299**	0
			Total	26.8	26.8	**0**

*Reconciliation did not consider a COG

15.5. Selective mining unit selection

Based on the envisaged production rates per area, shovel configuration and ore body characteristics, the required face length to support the production rates and level of production pressure that considers grade control allowance, structural complexity and economic grade continuity of the ore body was analysed and a suitable SMU size selected.

Ukwazi recommended the following SMU sizes for each area as shown in Table 15-4.

Table 15-4: SMU size recommendation

Description	Unit	Dibbwi East	Muntanga
ROM ore prod/year	Kt	2 400	1 000
Operational strip ratio	t:t	5.5	1.8
Total prod / month	Kt	1 298	233
Applied bench width	m	40	40
Applied bench height	m	10	10
Max pit depth	m	180	90
Pit length	m	2 400	780
Average pit width	m	565	1 000
Required face length at uniform ore distribution	m	**1 545**	**278**
Steady state nr of primary loaders	nr	4.4	0.8
Production pressure		**High**	**Low**
Grade control allowance		2-4 active loading faces High impact	1x active loading face Medium impact
Structural complexity		Flat with multiple overlaying ore bodies	Gentle dip with reasonable thickness
Economic grade continuity		Highly variable	Fairly consistent
SMU size recommendation	m	**10x10x2.5**	**5x5x2.5**
Dilution	%	21	18

15.6. Selective mining unit results

15.6.1. Muntanga

The global tonnage and grade curve for the selected SMU scenario at Muntanga is shown in Figure 15-4. This shows the full geological model, plus the dilution added through the SMU process, grouped by a diluted estimated economic COG within the USD 70 /lb U_3O_8 RPEEE pits.

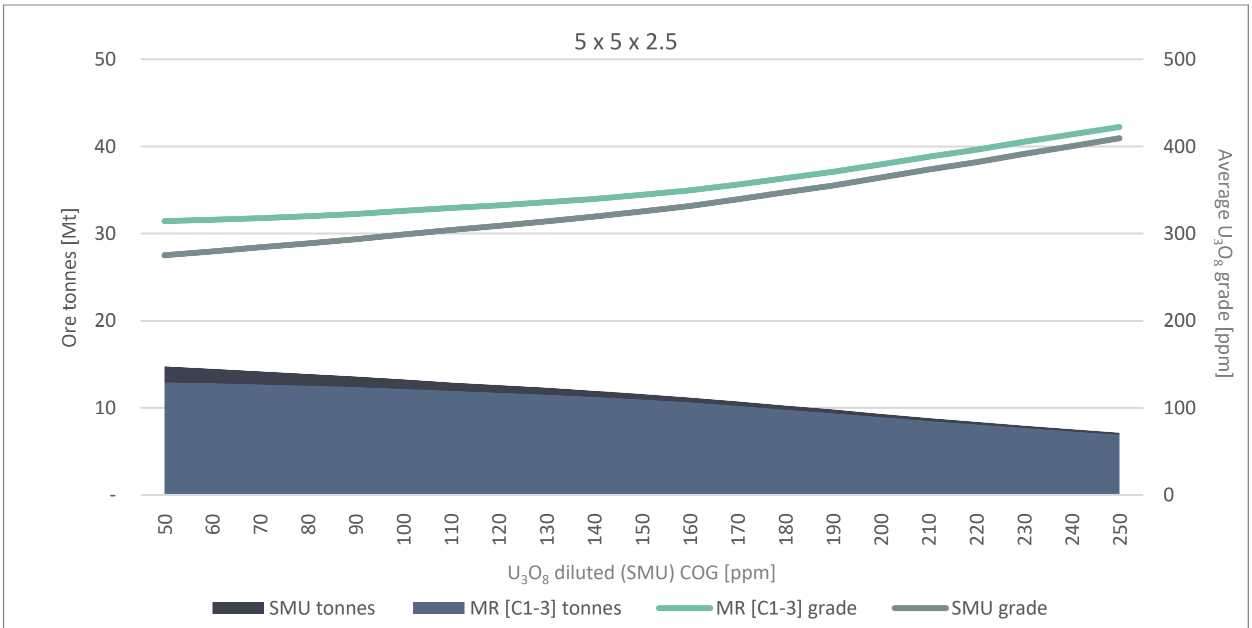

Figure 15-4: Muntanga TGP for selected SMU scenario within the USD 70 /lb U_3O_8 RPEEE pit

The average diluted U_3O_8 ore, grade, content and related tonnage-based dilution is shown in Table 15-5 for the selected SMU scenario.

Table 15-5: Muntanga SMU scenarios and related dilution

SMU size	COG =92ppm U_3O_8						
	Mineral Resource [Class 1-3]			SMU			
	Tonnes [Mt]	U_3O_8 grade [ppm]	U_3O_8 content [t]	Tonnes [Mt]	U_3O_8 grade [ppm]	U_3O_8 content [t]	Dilution [%]
5 x 5 x 2.5	12.7	317.7	4 032	14.2	284.1	4 032	11

15.6.2.Dibbwi East

The global tonnage and grade curve for the selected SMU scenario at Dibbwi East is shown in Figure 15-5. This shows the full geological model, plus the dilution added through the SMU process, grouped by a diluted estimated economic COG within the USD 70 /lb U_3O_8 RPEEE pits.

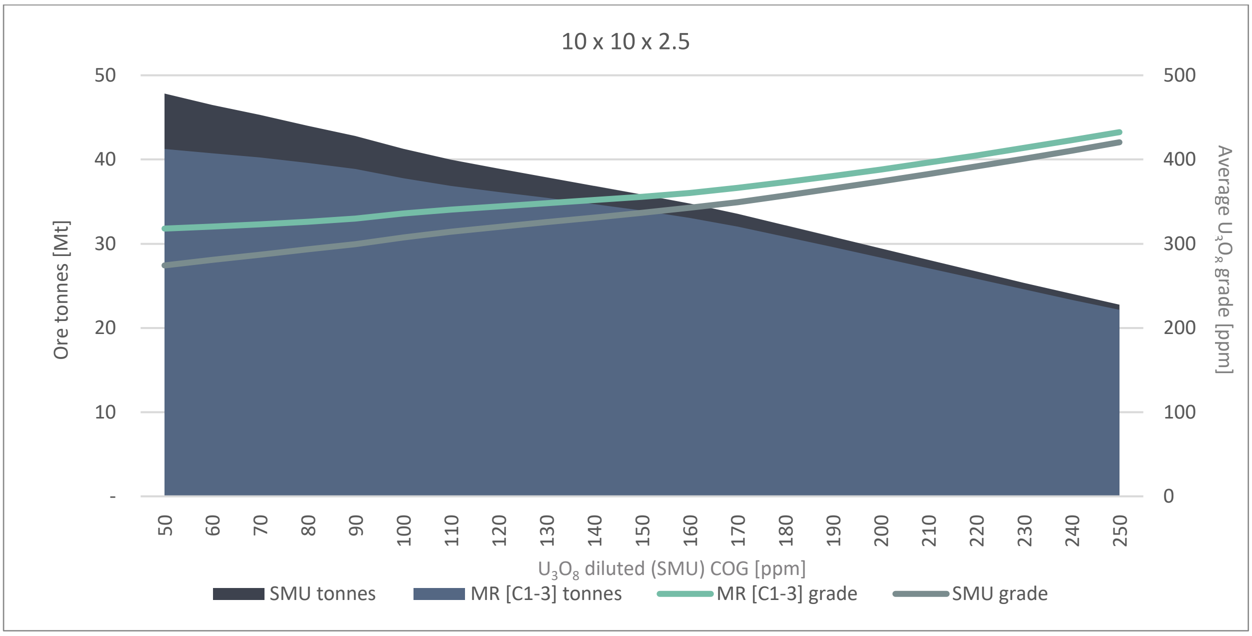

Figure 15-5: Dibbwi East TGP for selected SMU scenario within the USD 70 /lb U_3O_8 RPEEE pit

The average diluted U_3O_8 ore, grade, content and related tonnage-based dilution is shown in Table 15-6 for the selected SMU scenario.

Table 15-6: Dibbwi East SMU scenarios and related dilution

SMU size	COG =84ppm U_3O_8						
	Mineral Resource [Class 1-3]			SMU			
	Tonnes [Mt]	U_3O_8 grade [ppm]	U_3O_8 content [t]	Tonnes [Mt]	U_3O_8 grade [ppm]	U_3O_8 content [t]	Dilution [%]
10 x 10 x 2.5	39.6	326.0	12 908	44.0	293.3	12 908	10

15.7. Whittle open pit optimisation

15.7.1.Optimisation criteria

The Whittle process identifies which blocks must be mined and which ones must be left in the ground. To identify the blocks to be mined, an economic block model was created based on the mining (SMU) model. This was done by assuming production, mining, process, logistics and other costs and a long-term view of the commodity and consumables such as diesel prices.

Pit optimisation parameters considered were:

- Mining model with ore and waste definitions
- Pit slope angles
- Mining related modifying factors
- Physical characteristics
- Mining parameters
- Process parameters
- Logistical parameters
- Costs (fixed, variable, and ongoing capital)
- Mining
- Processing
- Logistics
- Selling
- Other
- Discount rate
- Revenue parameters.

15.7.2.Selective mining units imported into Whittle

Based on the SMU selection in Section 15.1 and the SMU results in Section 15.2, the SMU sizes per area as shown in Table 15-7 were imported into Whittle for optimisation.

Table 15-7: SMUs imported into Whittle

Domain	SMU dimensions [m]
Muntanga	5 x 5 x 2.5
Dibbwi - East	10 x 10 x 2.5

15.7.3.Overall pit slope angles

Overall pit slope angles as provided by SRK and used as input into the Whittle pit optimisation are shown in Table 15-8.

Table 15-8: Overall pit slope angles per area

Domain	Controlling mechanism	Description	Overall slope angle – Toe to crest [º]
Muntanga	Weathering of Mudstone	Sandstone/Siltstone/Mudstone	34
Dibbwi East	Weathering of Mudstone	Sandstone/Siltstone/Mudstone	39

15.7.4. General mining parameters

The modifying factors were applied as discussed in Section 15.3. The ore and waste densities as per the block models received where used, although these densities are very close to each other at 2.1 t/m^3, which indicates no distinct difference between the host material that contains U_3O_8 and the surrounding waste which mostly comprises cemented sandstone. Total U_3O_8 ore production (plant feed) for the OP optimisation process was limited to 3.4 Mtpa from both areas as shown in Table 15-9.

Table 15-9: Ore production limits for OP optimisation process

Description	Unit	Dibbwi East	Muntanga
ROM ore prod/year	Mtpa	2.4	1.0
Total	**Mtpa**		**3.4**

15.7.5.Cost

A summary of the mining cost parameters used in the optimisation is shown in Table 15-10. The mining cost was based on the following assumptions:

Table 15-10: Mining cost parameters used in optimisation

Description	Unit	Value	Comment
Reference mining cost (ore & waste)	USD/t rock	3.30	Calculated - Cost of mining a tonne of rock to a certain reference point. In this case, it was the pit exit.
Additional ore mining cost (ore only)	USD/t ore	0.55	Calculated - Additional ore of mining ore that includes grade control, selective loading cost for ore and rehandle.
Incremental depth mining cost per 10 m bench (ore & waste)	USD/t/10 m bench	0.05	Calculated (1.6 % of reference mining cost adjustment for depth)

15.7.6.Processing cost

The processing cost parameters applied in the optimisation process as confirmed by GoviEx are shown in Table 15-11. These were based on previous study results.

Table 15-11: Processing cost parameters used in optimisation

Description	Unit	Value	Comment
Additional ore hauling cost	USD/t ore-km	0.18	Calculated - Additional hauling cost further that waste material placed on WRD
Processing cost	USD/t ore	9.00	
Port cost	USD/lb U_3O_8	1.50	Total transport cost from project to convertors in Canada and USA
General & admin	USD/t ore	1.50	

15.7.7. Processing parameters

The processing recovery applied for the two areas in the optimisation process as confirmed by GoviEx is shown in Table 15-12. These were based on previous study results.

Table 15-12: Processing recovery per area

Area	Unit	Processing recovery
Gwabi	%	75.4
Njame	%	85.1
Muntanga	%	85.4
Dibbwi East	%	93.3
Dibbwi	%	74.6

15.7.8.Selling cost parameters

The product selling cost parameters are summarised in Table 15-13.

Table 15-13: Selling cost parameters

Description	Unit	Value	Comment
Royalty	%	5	% Government Royalty

15.7.9.Revenue and financial parameters

The financial parameters applied are shown in Table 15-14.

Table 15-14: Revenue and financial parameters

Description	Unit	Value
Product price	USD/lb U_3O_8	80
Discount rate	%	8
Tonne to pound conversion	lb/t	2 204.64

The product is sold per pound yellowcake containing a certain concentration of U_3O_8.

15.8. Open pit optimisation results

The OP optimisation process is defined as the techno-economic evaluation to determine the economically viable OP portion of a given Mineral Resource and/ or to determine the optimal transition point when going underground based on technoeconomic principles. The basis of this process is a mining block model and at a high level involves the determination of:

- Whether a given block in a model must be mined as waste or ore, or mined at all
- When it must be mined
- How it must be processed once it is mined.

15.8.1. Pit optimisation generic description

One of the outcomes of the pit optimisation process was the definition of the position and extent of the final techno-economically viable pit limit, using Whittle. Whittle uses the Lerchs-Grossman algorithm to analyse the optimal pit limit for the OP in three dimensions. The method is applied to the mining model and progressively constructs lists of related blocks that must or must not be mined. The final pit lists define a pit outline with the highest total relative value ("RV"), subject to the required pit slopes. This outline includes every block that "adds value" when waste stripping is considered and excludes every block that "destroys value". It considers all revenues and costs, mining and processing parameters. The optimisation process is divided into two processes:

1. Creation of a range of nested pit shells of increasing sizes, constructed by varying the product price and generating a pit shell at each price point
2. The selection of the optimal pit shell by generating various production schedules for each pit shell to estimate the RV for each schedule. The output of this process is a series of "pit-versus-value" curves.

Three pit-versus-value curves are generated:

1. **Best case:** Corresponds to minimum stripping in which mining follows the sequence of nested pit shells. Although this method gives the highest RV, it is not practical. It serves to provide the upper limit with regards to pit size, or the maximum theoretical value achievable within a certain pit
2. **Worst case:** Waste material is removed level-for-level to correspond with the maximum stripping scenario and therefore lowest RV. It serves to provide a lower limit with regards to pit size, or the minimum theoretical value achievable within a certain pit
3. **Specified case:** A case between the best and worst cases, which models the influence of pre-stripping to achieve a more practically executable production schedule than the best case, on the value curve.

The optimum specified Whittle shell is normally identified where the specified case value is maximised. A simplified illustration of the definitions is shown in Figure 15-6.

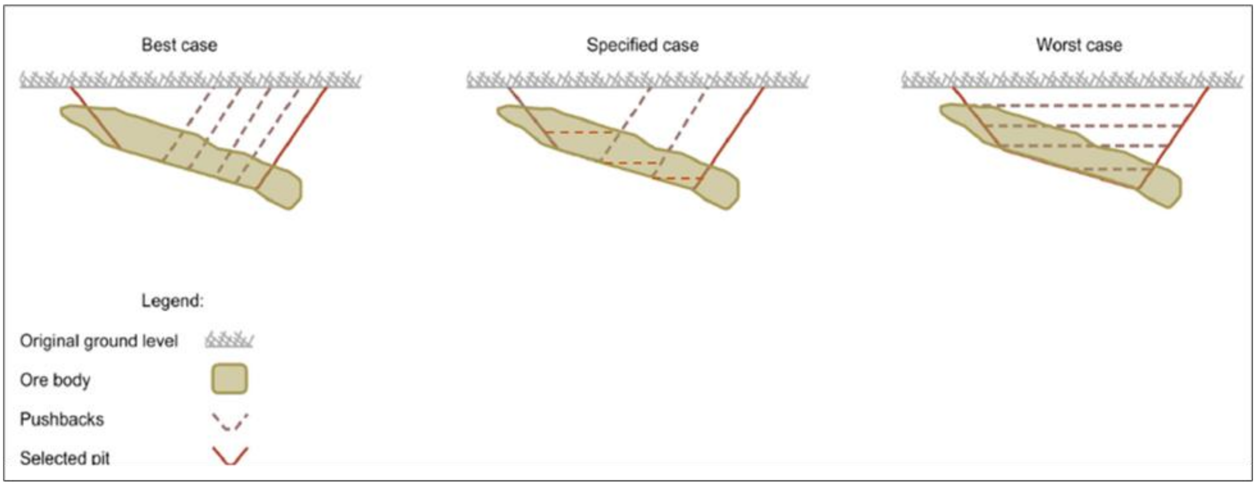

Figure 15-6: Best, worst and specified case definition

15.8.2. Pit-by-pit graph overview

The Whittle process identifies which blocks must be mined and which ones must be left in the ground. To identify the blocks to be mined, an economic block model was created based on the mining (SMU) model. This was done by assuming production, mining, process, logistics and other costs and a long-term view of the commodity and consumables such as diesel prices. Using the economic block values, each positive block was checked to determine whether its value exceeds the cost of removal of overlying waste blocks, or where the cost of mining, using OP mining methods, exceeds the UG mining cost. The analysis of pit limits which maximises the RV required that the time value of money be considered when deciding which blocks must be mined and which blocks must be left in the ground during the life of the OP mine. Pit optimisation parameters considered were:

- Mining model with ore and waste definitions
- Pit slope angles
- Mining related modifying factors
- Physical characteristics
- Mining parameters
- Process parameters
- Logistical parameters
- Costs (fixed, variable, and ongoing capital)
- Discount rate
- Revenue parameters.

Through the Whittle optimisation process, a combination of ore blocks will be selected to deliver the highest value in an OP operation for a given set of design, operating and economic assumptions. It must be noted that the value stated in the optimisation process is a RV based on the Whittle schedule including fixed and variable operational costs. This RV is expressed as a value index ("RVI") in this report. The resultant pit shell represents the optimum pit limit, based on the criteria used.

Revenue factors ("RF") are applied in the pit optimisation software (Whittle in this case) to define the range of nested pits. If the RF is set to 2, the commodity price used to create the specific shell is equal to twice the commodity price relative to the base case. This results in a pit larger than the base case due to higher revenue gained for the same unit costs (a pit of the same size will have a higher value when applying a higher RF). If the RF is set to 0.5, it generates a pit shell that assumes only half the commodity price of the base case. This results in a smaller pit (a similarly sized pit will have a lower value when applying a lower RF). The higher the RF, the larger the pit, or the higher the RVI for the same-sized pit.

15.8.3.Pit selection process

The Whittle pit-by-pit graphs and the marketability curves form the basis of the pit shell selection process. When assessing the pit-by-pit graphs in this section of the report, it is important to consider the following:

- The Whittle pit optimisation process is based on the creation of a range of nested pit shells of increasing sizes, constructed by varying the product price by the application of a RF to generate a techno-economic pit shell at each price point
- The pit numbers increase with increasing RF. The RF1 pit represents the pit shell created with the base case revenue inputs
- Pits smaller than the RF1 pit were created by systematically reducing the commodity prices (revenue), whilst the pits larger than the RF1 pit used commodity prices (revenue) that were systematically increased beyond the base case parameters – simulating hypothetical low- and high-price environments and the resultant effect on optimal pit size
- Pit sizes created in this process represent the pits which will yield the maximum value for that RF (commodity prices) as an instantaneous or "snapshot" pit size analysis, i.e., as if all ore and waste were mined in an instant
- The appropriate value for each pit is determined by the way it is mined and how the duration of mining, combined with the discount factor impacts the time value of money and thus the comparative RV of each pit
- The pits are therefore scheduled in three ways: best case, worst case and specified case to compare the RV of each pit on each of the three scheduling philosophies
- The value indicated on the RVI is relative (not absolute) and must be read relative to other scenarios
- The RVI comparison excludes initial capital requirements
- The darker brown bars represent ROM production while the light brown bars show the associated waste tonnes
- To enable a meaningful comparison, the RV of each pit on the family of nested pits was estimated on the base case prices, irrespective of which RF was used to create the shell.

Each pit-by-pit graph produced as part of the Whittle process should be read considering the principles listed above in terms of the nested pits and their scheduled RVs for each of the scheduling philosophies.

15.8.4. Pit selection strategies

The pit selection process is the most critical activity of the pit optimisation process. To allow for an informed decision, four pit selection strategies were considered. A brief description of the selection process is outlined below. It must be noted that producing realistic schedules in pit optimisation software requires careful consideration and application of scheduling targets, scheduling constraints and Whittle pushbacks.

No pit optimisation or pit selection is complete without careful and appropriate consideration of pit selection strategies.

15.8.4.1. Revenue factor 1 pit

The RF1 pit shell is generated based on commodity prices equal to the base case inputs. This selection rarely coincides with the mathematical maximum RVI. This pit selection strategy is most commonly used for bulk material operations when there is a material historic variance in the long-term price forecast and the actual spot price of the commodity. This strategy can be seen as an aggressive pit selection strategy in terms of size of the selected pit.

15.8.4.2. Mathematical maximum relative value – specified case

No interpretation is required for this pit selection strategy: it is a theoretical and mathematical selection of the maximum RVI, as derived from the pit-by-pit graph. The maximum value pit is often selected as the basis for design, as it represents the mathematical maximum value in relation to the input parameters, inclusive of production and discount rates, as used in the Whittle optimisation.

15.8.4.3. Mathematical maximum plus life

This pit selection strategy aims to identify the opportunity to materially extend the pit life, without material mathematical RV destruction. A larger pit results in a longer LOM at the same production rate. This can benefit the stakeholders (communities amongst others) and limit the risk to the shareholders by extending the life of the operation through multiple commodity price cycles. This strategy can materially increase the size of the pit, without having a material negative impact on the RVI.

15.8.4.4. Marketability curve

This pit selection strategy aims to consider the incremental OP mining cost and strip ratio ("S/R") on a mining cost basis and a total product basis. Based on this, cost or S/R cut-offs can be applied to ensure that the incremental cost of mining OP does not exceed a targeted cost profile. A strategic incremental S/R cut-off can be applied. This can inform a potential transition to UG mining methods, depending on the strategy of the owner or the UG mining method(s) selected. The latter falls outside the scope of this study.

The average and incremental S/R and mining cost curves relative to the increasing pit sizes are represented in the marketability curves generated for each Whittle scenario.

15.8.5. Pit optimisation results

The optimisation results were based on the input parameters in Section 15.7 and the pit selection strategies outlined in this section. All pit selections were based on the specified case value curve, and it was critical to understand which Whittle algorithm is the most appropriate and what pushback selection strategy will be used.

The following was applied for the optimisation scenarios in the different areas as the basis of the specified case curves:

- Muntanga
- Fixed Lead = 10 algorithm was used
- Pushback pits were selected in increments of three-year ore intervals (approximately 3Mt)
- Dibbwi East
- The Milawa balanced algorithm was used
- Pushback pits were selected in increments of three-year ore intervals (approximately 7Mt).

15.8.5.1. Muntanga results and pit selection

The following subsections discuss the pit selection results for Muntanga as indicated on the pit-by-pit graph and marketability curve in Figure 15-7 and Figure 15-8 respectively.

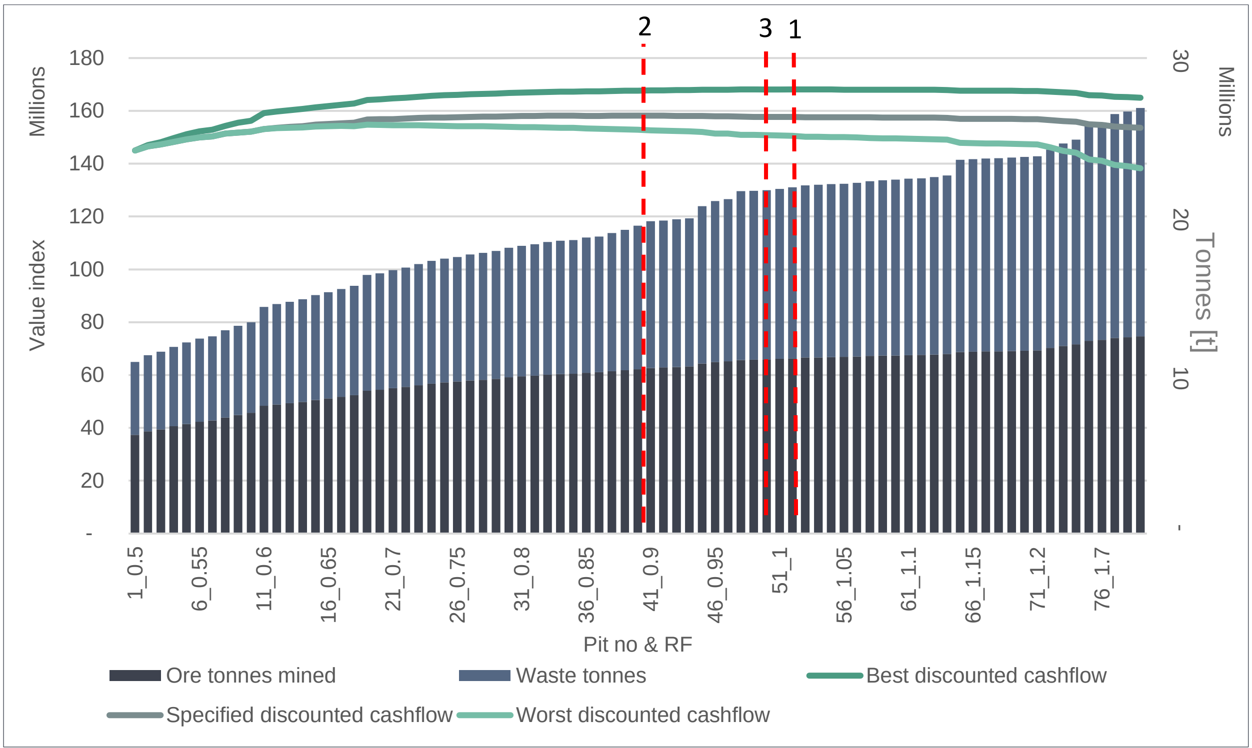

Figure 15-7: Pit by pit graph – Muntanga

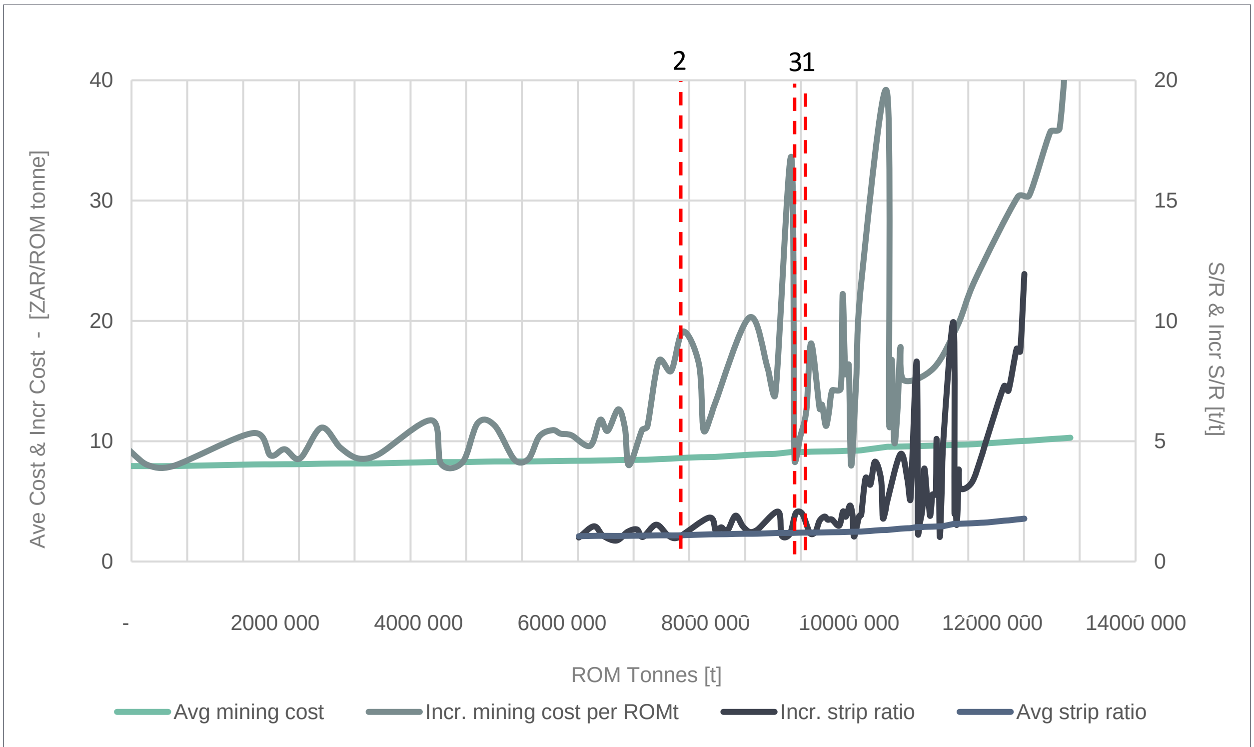

Figure 15-8: Marketability curve – Muntanga

15.8.5.1.1. Optimum relative value – Muntanga

This selection strategy considered the optimum (theoretical maximum) RV based on the specified case of the Whittle pit-by-pit graph. Pit 40 with RF = 0.89 was selected which contained 8.5 Mt of ROM ore, a S/R of 1.29 (t:t) and a life of 8.9 years. This results in a RV of USD158.2 million with an average U_3O_8 grade of 330 ppm.

The selected pit is shown in the pit-by-pit graph (Figure 15-7) identified by “2”, relative to the best case (top curve), the worst case (bottom curve) and the specified case (middle curve).

15.8.5.1.2. Maximised relative value with extended life – Muntanga

This selection strategy considered the theoretically maximum RV of the specified case, based on the selection of a larger pit, without materially affecting the RV. This would result in a selected pit that is larger, and at a value similar to the theoretical maximum. Pit 49 with RF= 0.98 was selected which contained 9.0 Mt of ROM ore, a S/R of 1.43 (t:t) and a life of 9.3 years. This results in a RV of USD157.9 million with an average U_3O_8 grade of 325 ppm.

The RV was 0.22 % lower relative to the theoretical maximum-value selection strategy but increased the life of the operation by 0.4 years or five months.

The selected pit is shown in the pit-by-pit graph (Figure 15-7) identified by “3” relative to the best case (top curve), the worst case (bottom curve) and the specified case (middle curve).

15.8.5.1.3. Revenue factor = 1 – Muntanga

Pit 51 was selected and contained 9.0 Mt of ROM ore, a S/R of 1.44 (t:t) and a life of 9.3 years. This results in a RV of USD157.7 million with an average U_3O_8 grade of 324 ppm.

The selected pit is shown in the pit-by-pit graph (Figure 15-7) identified by “1” relative to the best case (top curve), the worst case (bottom curve) and the specified case (middle curve).

15.8.5.1.4. Summary of results – Muntanga

Table 15-3 summarises the results of the pits selected according to the selection strategies in Section 15.8.4.

Table 15-15: Muntanga pit selection results summary

Description	Unit	[1] RF =1	[2] Max value	[3] Max value + life
Pit no		51	**40**	49
RF		1	**0.89**	0.98
S/R	t:t	1.44	**1.29**	1.43
Value	USDm	157.7	**158.2**	157.9
Life	years	9.3	**8.9**	9.3
ROM	Mt	9.0	**8.5**	9.0
U_3O_8 grade	ppm	324	**330**	325

Based on the optimum RV, Pit 40 (shown in Figure 15-9) was selected as the preferred pit based on the following:

- Highest value compared to RF=1 and maximised RV with life pits
- Lowest S/R
- Highest average uranium grade.

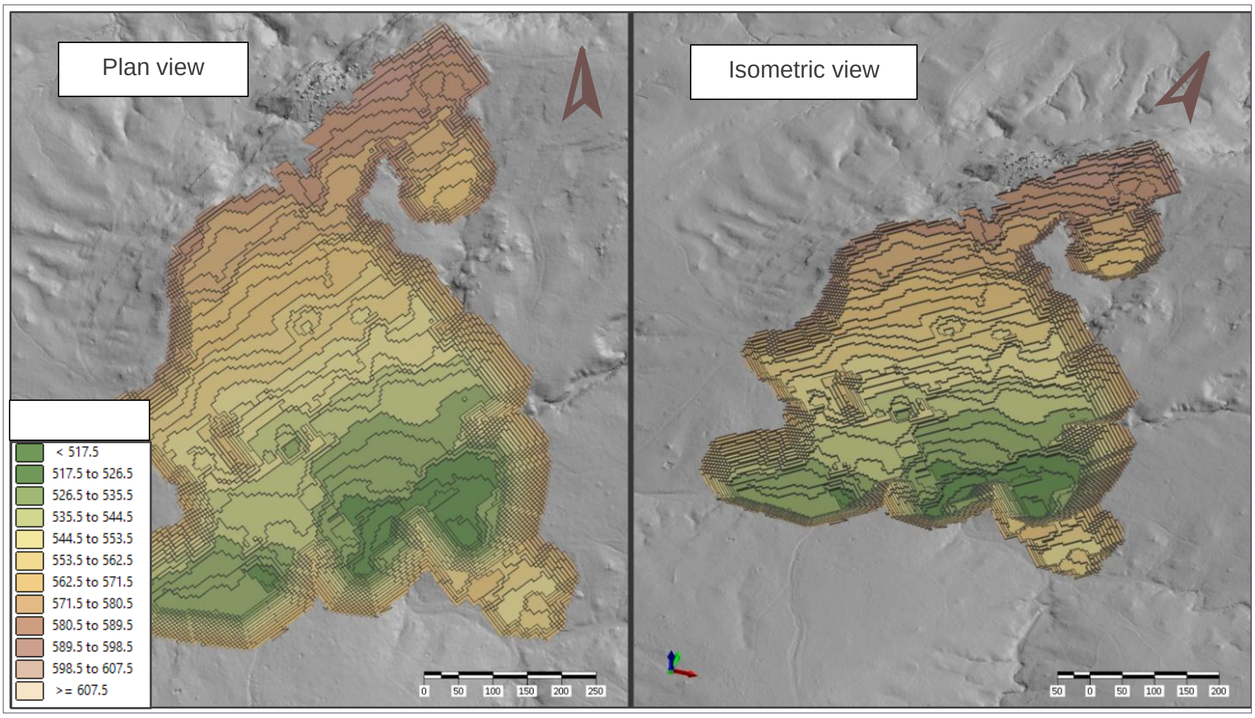

Figure 15-9: Plan view and Isometric view of Pit 40 - Muntanga

15.8.5.2. Dibbwi East results and pit selection

The following sub-sections discuss the pit selection results for Dibbwi East as indicated on the pit-by-pit graph and marketability curve in Figure 15-10 and Figure 15-11 respectively.

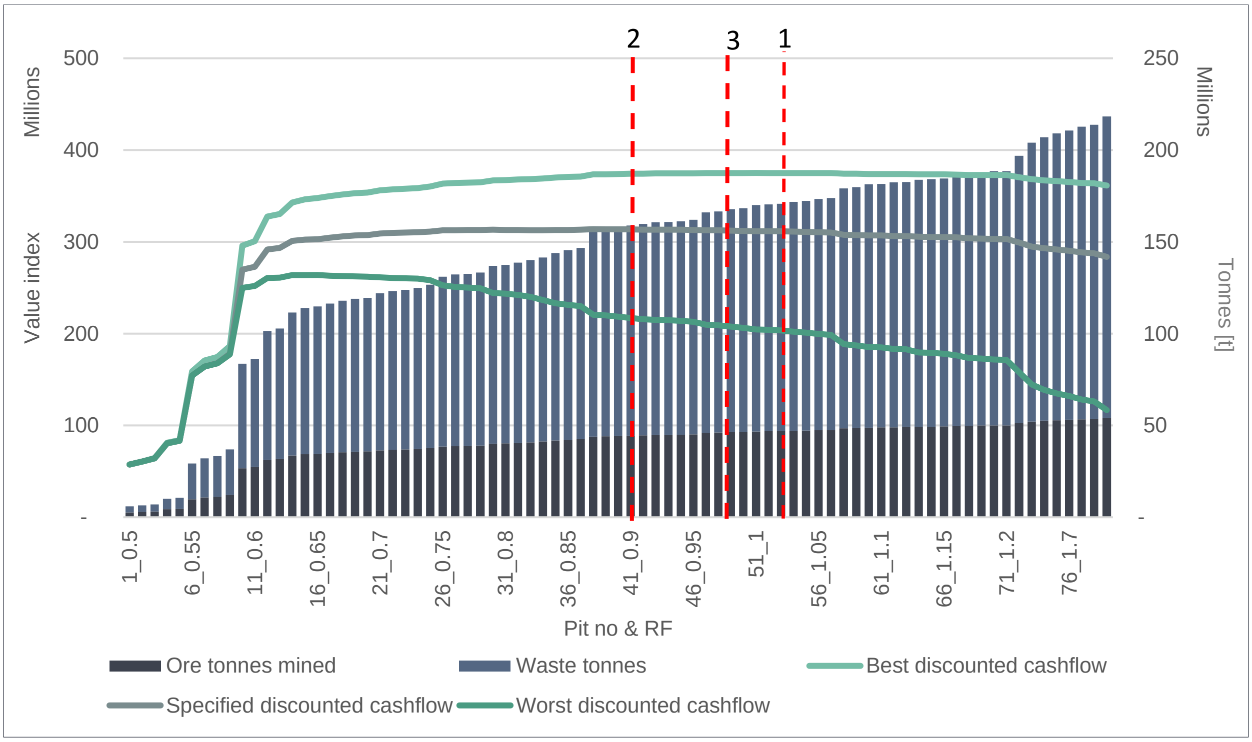

Figure 15-10: Pit by pit graph – Dibbwi East

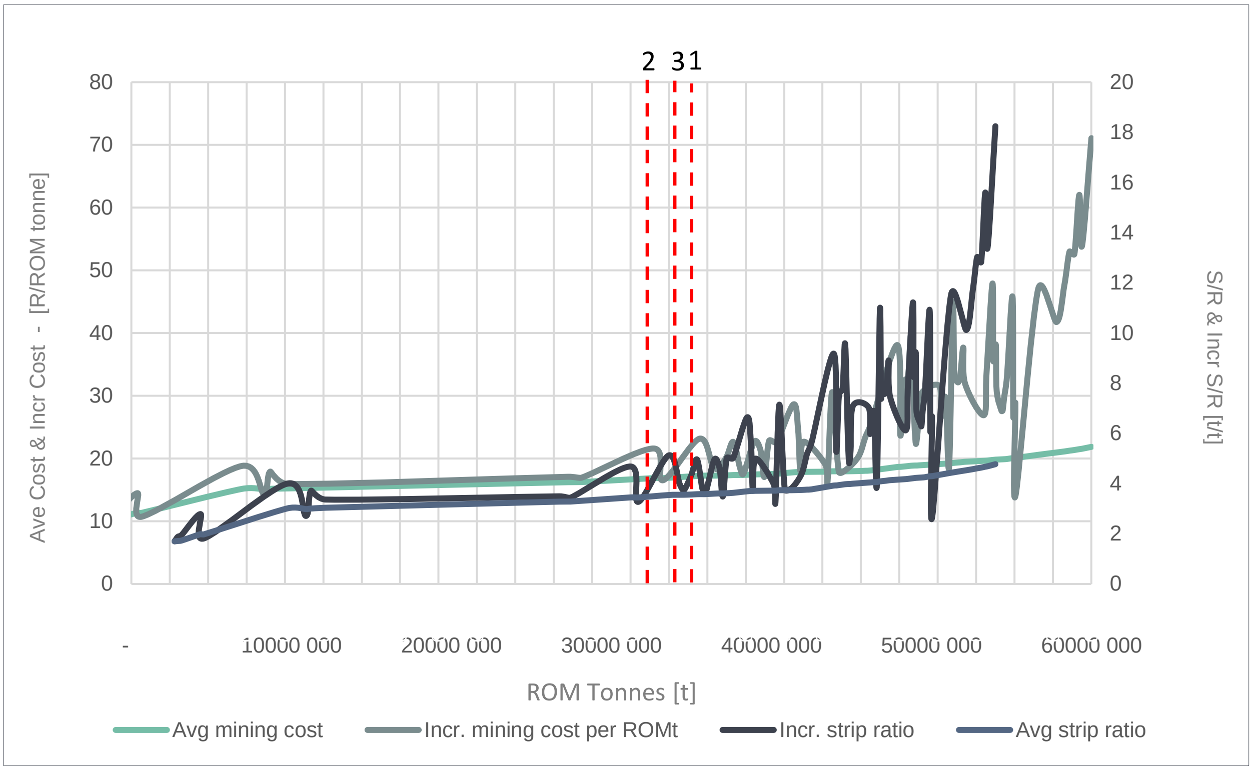

Figure 15-11: Marketability curve – Dibbwi East

15.8.5.2.1. Optimum relative value – Dibbwi – East

This selection strategy considered the optimum (theoretically maximum) RV based on the specified case of the Whittle pit-by-pit graph. Pit 38 with RF = 0.87 was selected which contained 31.3 Mt of ROM ore, a S/R of 3.92 (t:t) and a life of 13.6 years. This results in a RV of USD313.8 million with an average U_3O_8 grade of 321 ppm.

The selected pit is shown in the pit-by-pit graph (Figure 15-10) identified by "2", relative to the best case (top curve), the worst case (bottom curve) and the specified case (middle curve).

15.8.5.2.2. Maximised relative value with extended life – Dibbwi East

This selection strategy considered the theoretically maximum RV of the specified case, based on the selection of a larger pit, without materially affecting the RV. This would result in a selected pit that is larger, and at a value similar to the theoretical maximum. Pit 46 with RF= 0.95 was selected which contained 32.1 Mt of ROM ore, a S/R of 4.00 (t:t) and a life of 14.0 years. This results in a RV of USD312.8 million with an average U_3O_8 grade of 320 ppm.

The RV was 0.3 % lower than the theoretical maximum-value selection strategy but increased the life of the operation by 0.34 years or four months.

The selected pit is shown in the pit-by-pit graph (Figure 15-10) identified by "3", relative to the best case (top curve), the worst case (bottom curve) and the specified case (middle curve).

15.8.5.2.3. Revenue Factor = 1 – Dibbwi East

Pit 51 was selected and contained 33.1 Mt of ROM ore, a S/R of 4.09 (t:t) and a life of 14.4 years. This results in a RV of USD311.6 million with an average U_3O_8 grade of 219 ppm.

The selected pit is shown in the pit-by-pit graph (Figure 15-10) indicated by "1", relative to the best case (top curve), the worst case (bottom curve) and the specified case (middle curve).

15.8.5.2.4. Summary of results – Dibbwi East

Table 15-16 summarises the results of the pits selected according to the selection strategies in Section 15.8.4.

Table 15-16: Dibbwi East pit selection results summary

Description	Unit	[1] RF =1	[2] Max value	[3] Max value + life
Pit no		51	**38**	46
RF		1	**0.87**	0.95
S/R	t:t	4.09	**3.92**	4.00
Value	$m	311.6	**313.8**	312.8
Life	years	14.4	**13.6**	14.0
ROM	Mt	33.1	**31.3**	32.1
U_3O_8 grade	ppm	319	**321**	320

Based on the optimum RV, Pit 38 (shown in Figure 15-12) was selected as the preferred pit based on the following:

- Highest value compared to RF=1 and maximised RV with life pits
- Lowest S/R
- Highest average uranium grade.

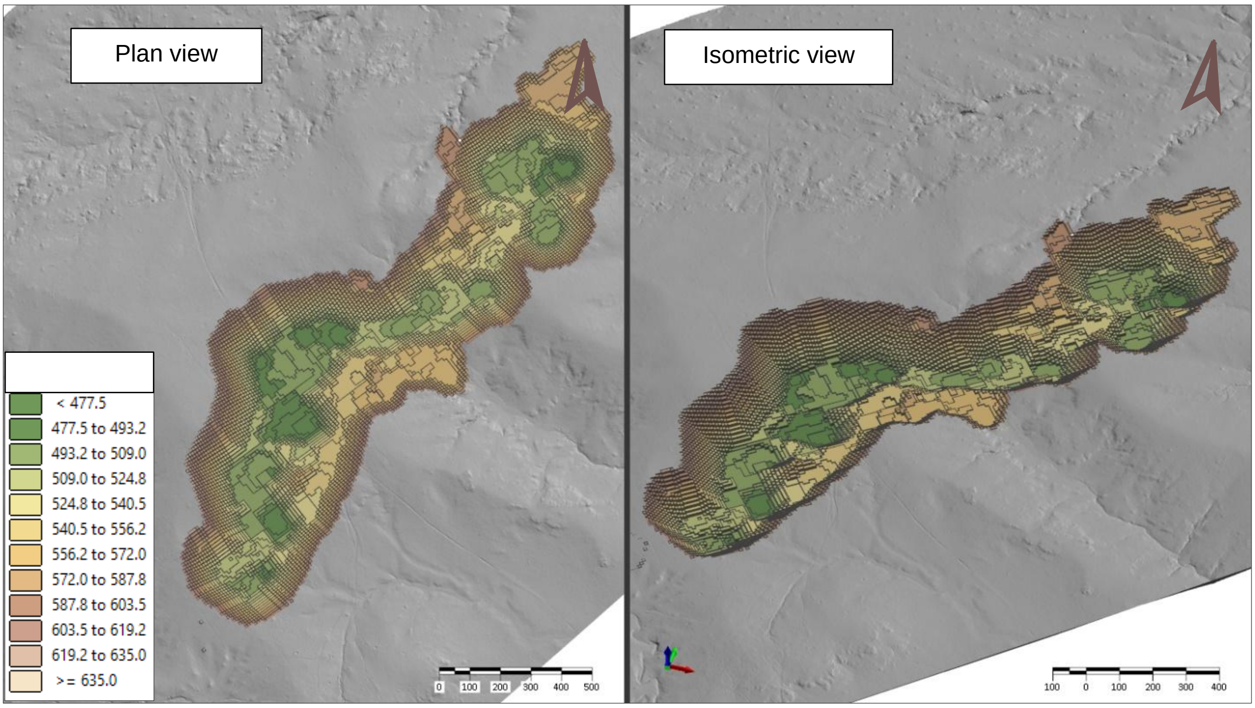

Figure 15-12: Plan view and Isometric view of Pit 38 – Dibbwi East

15.8.6. Sensitivity analysis

A sensitivity analysis was conducted on the selected pit shells for each area. The three input variables subjected to sensitivity testing were price, cost and slope angles. Each input variable was increased and decreased by 5 % increments to determine the impact on the RV of each pit, should they change over time.

The following sections show the results of the sensitivity analysis:

15.8.6.1. Muntanga sensitivity analysis

In Figure 15-13 Muntanga is more sensitive to price than cost variations and has a negligible effect on value due to variation in overall slope angle.

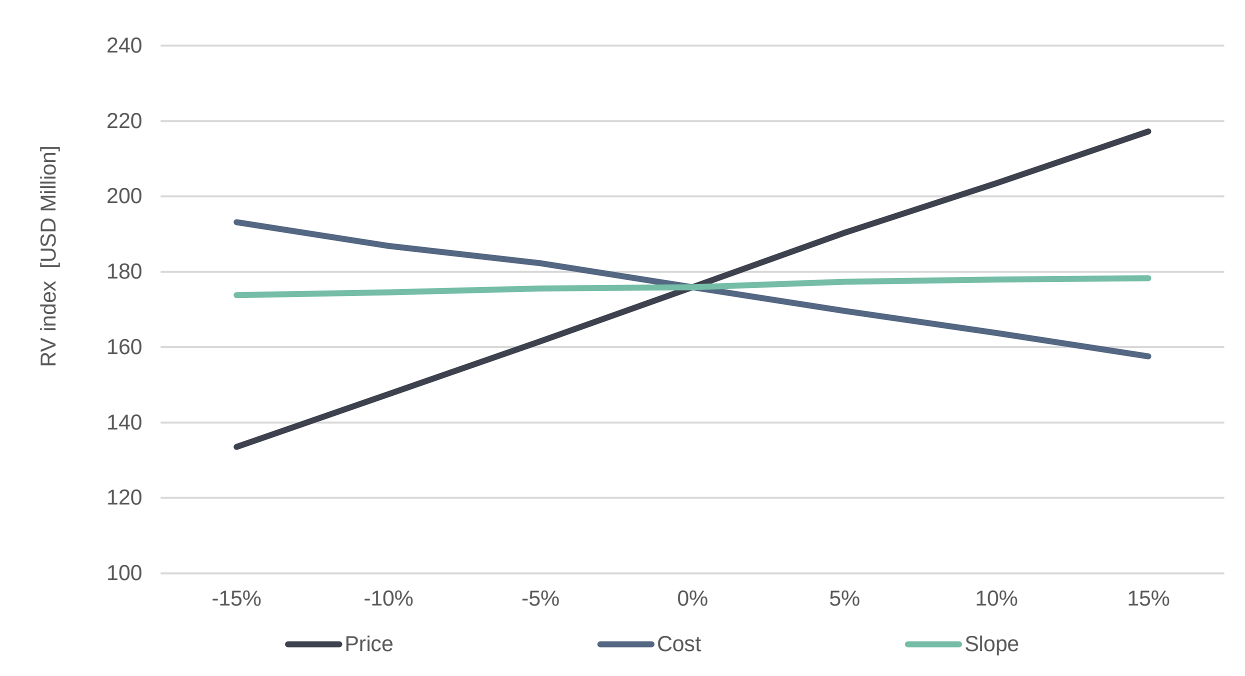

Figure 15-13: Muntanga sensitivity analysis

15.8.6.2. Dibbwi East sensitivity analysis

Dibbwi East is sensitive to price and cost variations, as in both instances the value of the pit can decrease or increase. It is also sensitive to a decrease in overall slope angle that destroys value as shown in Figure 15-14.

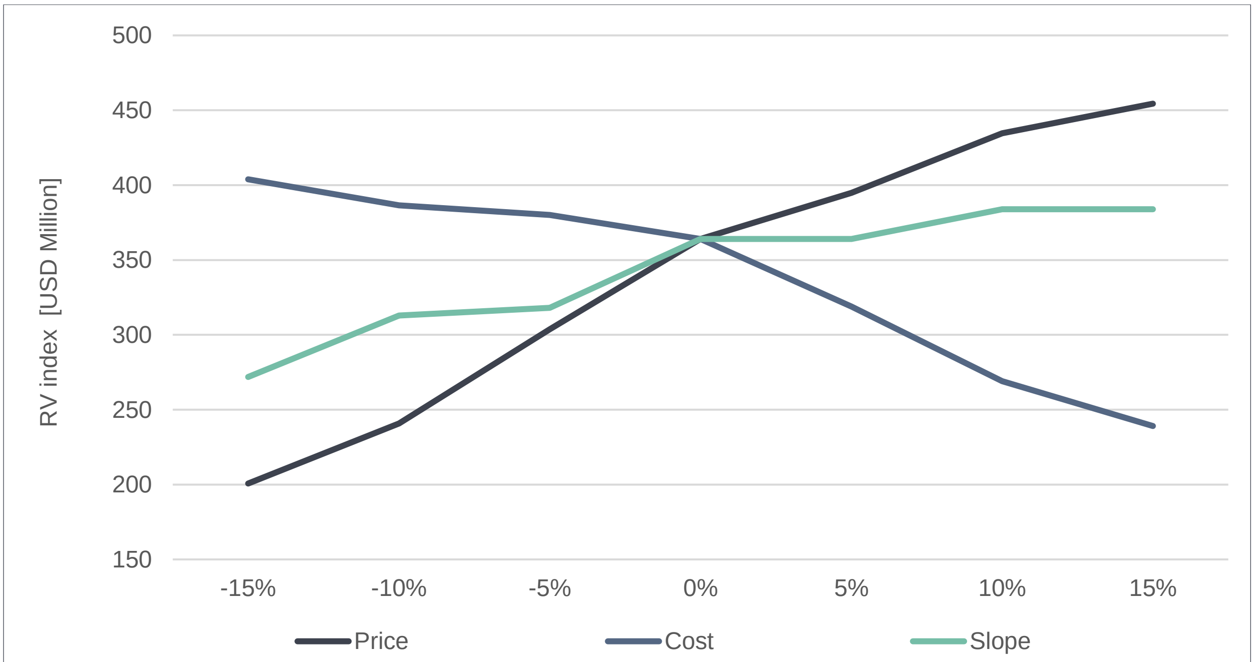

Figure 15-14: Dibbwi East sensitivity analysis

15.9. Mine design

Geovia Surpac general mine planning software was used for the detailed pit designs based on the selected Whittle pit shells for each area. This involved the addition of berms, ramps and haul roads to the pit shells generated by the Whittle pit optimisation process. A cutback and/or pushback design methodology was followed to create a series of manageable phases that can be exploited with the available mining equipment and that can meet practical geotechnical and operational mining constraints. Production scheduling is based on the underlying set of cutbacks or pushbacks that form the basis of determining the Mineral Reserves and provide guidance to eventual short-term mine planning.

15.9.1. Mine design criteria

Table 15-17 lists the pit slope parameters used for detail pit design. Further geotechnical information in relation to these design parameters and domains is outlined in Section 16.2.4.2 and general design parameters for haul roads in Section 16.2.4.1.

Table 15-17: Mine design criteria

Area	Controlling mechanism	Description	Maximum slope height [m]	Bench height [m]	Batter angle [deg]	Berm width [m]	Geotechnical berm width [between	Ramp width [m]	Inter – ramp angle – toe to toe [deg]	Inter – ramp angle – toe to crest [deg]	Overall slope angle – Toe to crest [deg]
Overburden	Sloughing	Topsoil and CW rock	10	5	35	5					
Muntanga	Weathering from mudstone	Sandstone/Siltstone/ mudstone	60	10	80	13	N/A	25	34	39	34
Dibbwi East	Weathering from mudstone	Sandstone/Siltstone/ mudstone	140	10	80	10	20	25	40	46	39

15.9.2. Final pit designs and deployment strategies

The final pit design is dependent on the deployment strategy followed, as different deployment strategies require different haul routes and access ramps. Cutback and pushback deployment strategies were considered to deploy the pits in a practically executable manner and to ensure that sufficient access to the mining blocks is always available. Deployment was guided by the Whittle pushbacks, chosen to ensure constant ore supply.

The deployment strategies in Table 15-18 were considered most appropriate and practically executable for each area.

Table 15-18: Deployment strategies per area

Area	Deployment strategies
Muntanga	Cutback
Dibbwi East	Pushback

The following sections describe the deployment strategies and final designs for each area.

15.9.3. Muntanga

As per Section 15.8, Whittle shell 40 based on the optimum RV was selected as the pit shell to proceed with final pit designs. Figure 15-15 shows the Muntanga Whittle shell 40 with selected pushbacks as guidance for the pit deployment and design. Whittle pushbacks were chosen in three-year intervals (3 Mt) to supply constant ore per pushback in support of the yearly ore requirement.

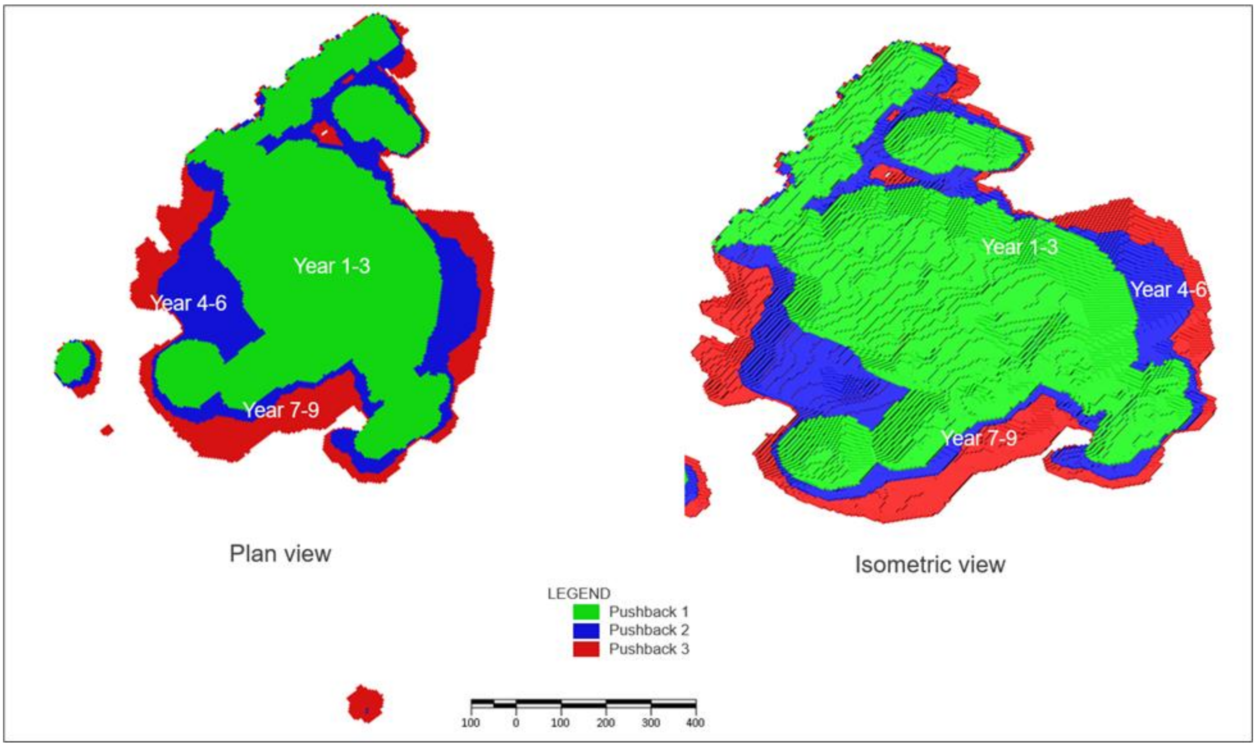

Figure 15-15: Muntanga Whittle shell 40 with selected pushbacks

15.9.3.1. Muntanga deployment and final design

Muntanga is a shallow pit outcropping on the surface at a drivable grade allowing access from multiple fronts.

Figure 15-16 shows the cutback strategy and the resultant final pit design evaluated for Muntanga based on the Whittle pushbacks in Figure 15-15.

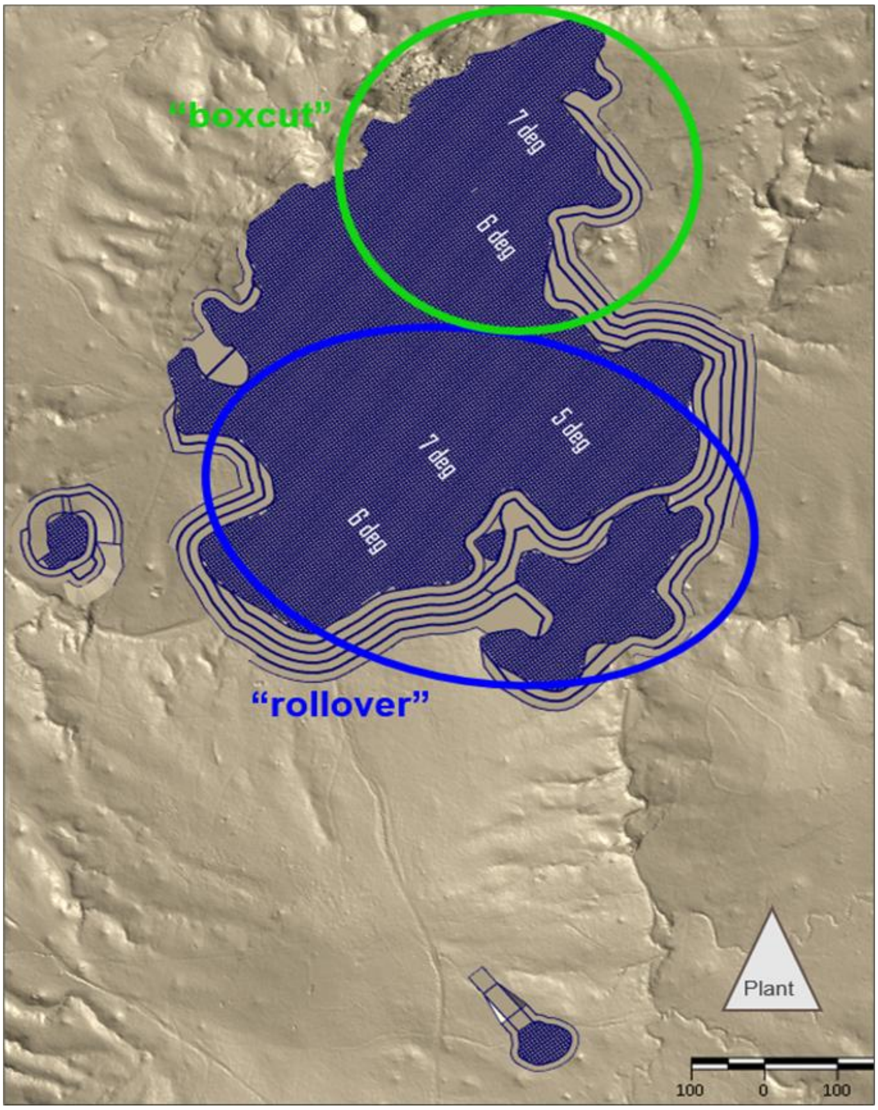

Figure 15-16: Final pit design evaluated for Muntanga

Table 15-19 and Figure 15-17 shows the comparison between Whittle shell 40 and the final design. Ore tonnes reduced by 0.85 % and the grade by 0.01 %, while the strip ratio increased by 2.89 %.

Table 15-19: Comparison – Whittle shell 40 and the final design - Muntanga

Description	Unit	Whittle shell 40	Design	Variance [%]
Ore tonnes	Mt	8.5	8.4	-0.9
Ore grade	ppm	331	331	-0.0
Waste tonnes	Mt	9.9	10.1	2.0
S/R	t:t	1.17	1.21	2.9

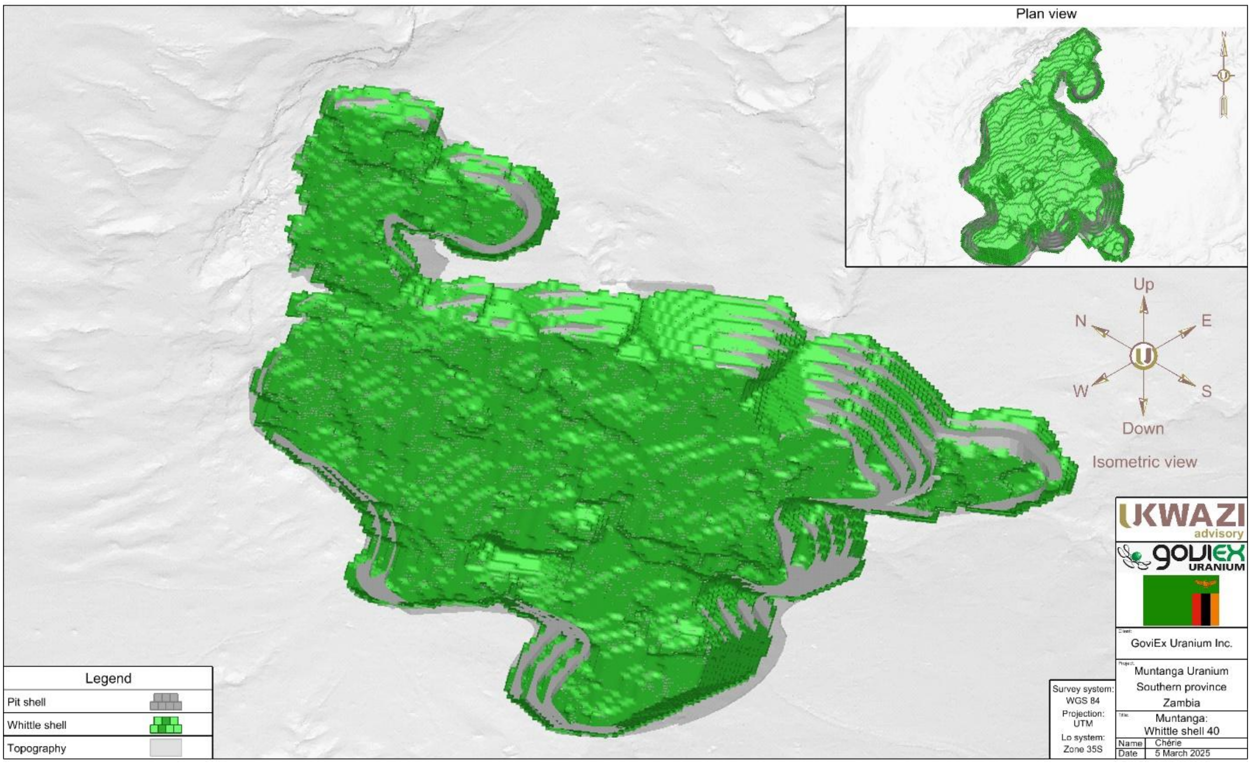

Figure 15-17: Muntanga Whittle shell 40 versus final design

15.9.4.Dibbwi East

As discussed in Section 15.8, Whittle shell 38, based on the optimum RV was selected as the pit shell to proceed with final pit designs. Figure 15-18 shows the Dibbwi East Whittle shell 38 with selected pushbacks as guidance for the pit deployment and design. Whittle pushbacks were chosen in three-year (7 Mt) intervals to supply constant ore per pushback in support of the yearly ore requirement.

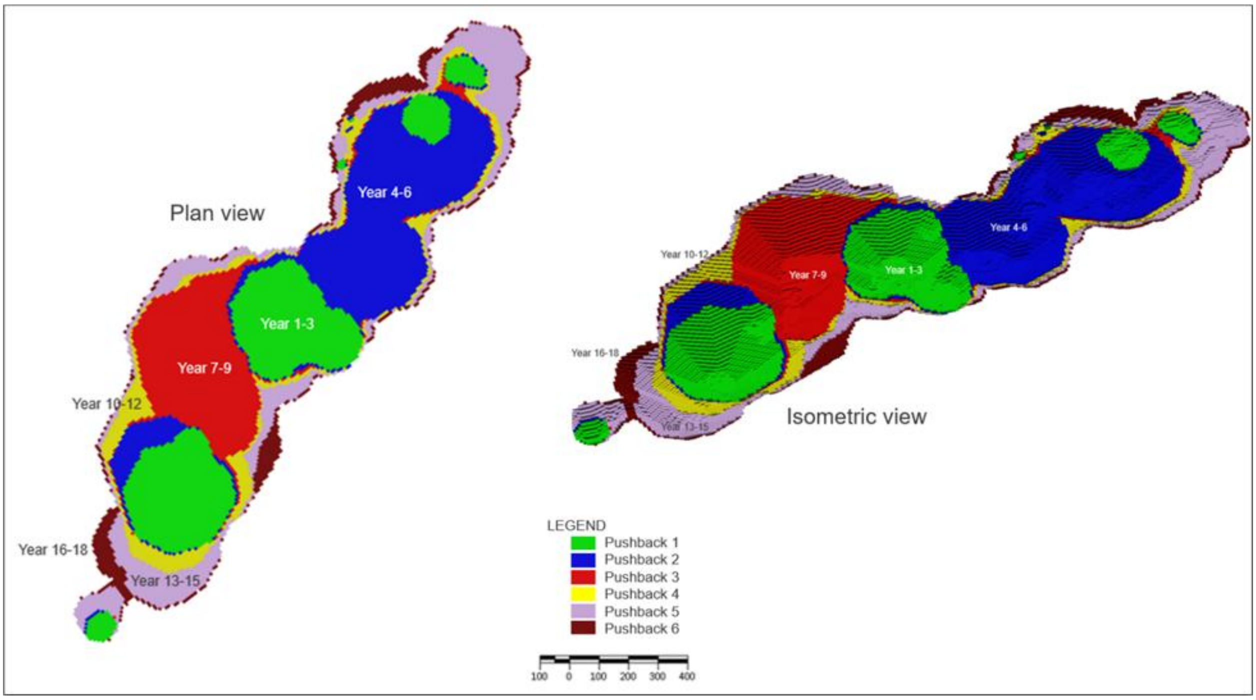

Figure 15-18: Dibbwi East Whittle shell 38 with selected pushbacks

15.9.4.1. Dibbwi East deployment and final design

Dibbwi East is the largest pit and synchronised pushback mining will assist in achieving a reasonably consistent grade profile.

Figure 15-19 shows the final design and pushback strategy evaluated for Dibbwi East based on the Whittle pushbacks in Figure 15-18.

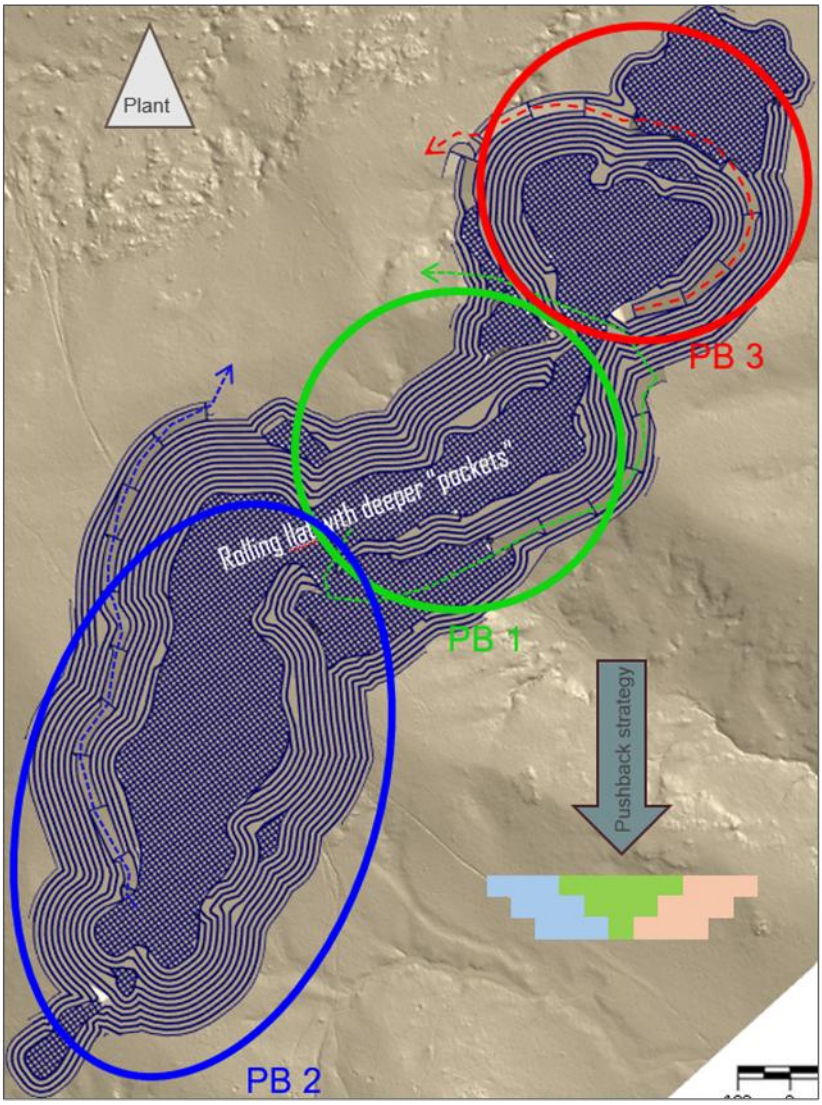

Figure 15-19: Final pit design evaluated for Dibbwi East

Table 15-20 and Figure 15-20 show the comparison between Whittle shell 38 and the final design. Ore tonnes are reduced by 0.09 % and the grade by 0.97 %, while the S/R increased by 10.52 %.

Table 15-20: Comparison – Whittle shell 38 and the final design – Dibbwi East

Description	Unit	Whittle shell 38	Design	Variance [%]
Ore tonnes	Mt	31.3	31.3	-0.1
Ore grade	ppm	321	317	-1.0
Waste tonnes	Mt	121.4	134.1	10.4
S/R	t:t	3.88	4.29	10.5

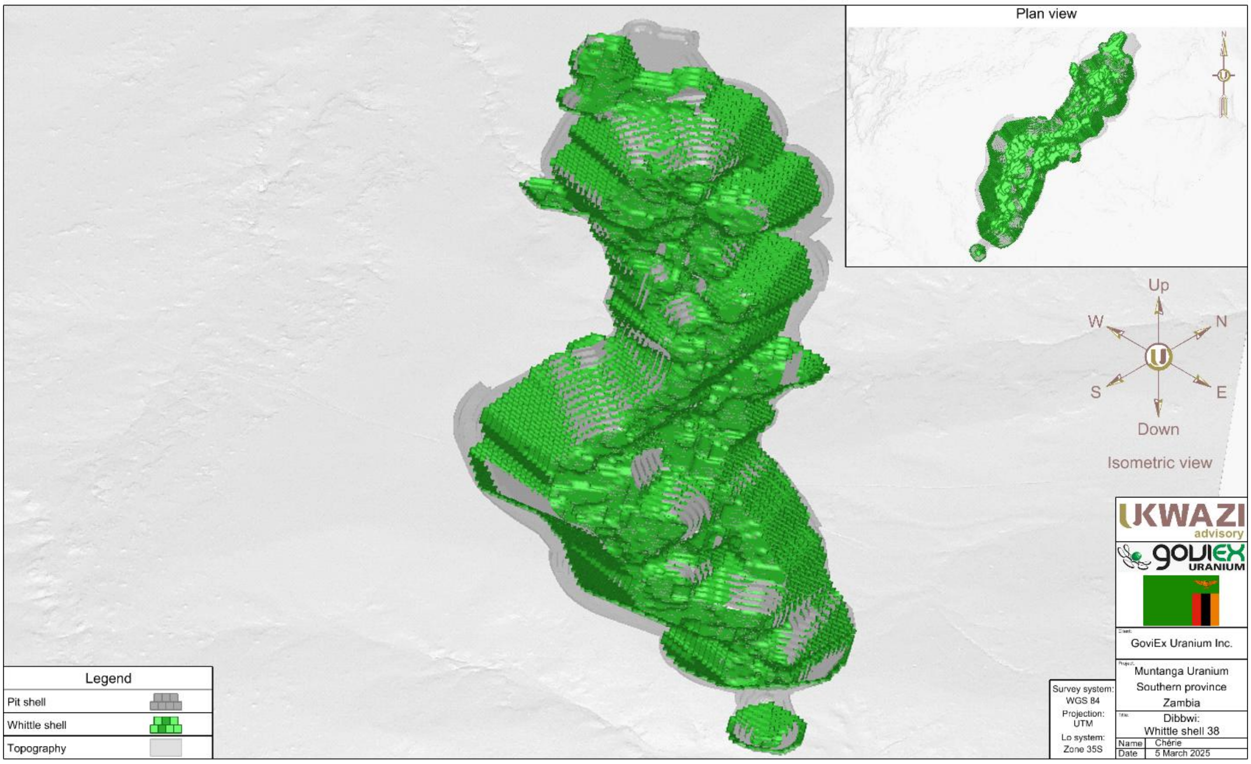

Figure 15-20: Dibbwi East Whittle shell 38 versus final design

15.10. Mineral Reserve statement

Ore Reserve estimates for the Project were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) by Jacobus Johannes Lotheringen, Pr.Eng. an independent QP as defined in NI 43-101.

Appropriate modifying factors were developed during the FS to convert the estimated Mineral Resources to a Mineral Reserve. These factors included, mining, processing, metallurgical, infrastructural, economic, marketing, legal, environmental, social and governmental factors.

A detailed LOM plan was completed as the basis for the Mineral Reserve estimate. Various technical aspects were considered in the mine design and schedule including the determination of the economic pit limits, geotechnical parameters, mining methodology, mining sequence, pit access, ramp placement, equipment capability, production rates and practical mining considerations. Details of the mine design and associated production schedule, that formed the basis of the estimated Mineral Reserve are described in detail in Chapter 15 and 16 of this TR.

The Mineral Resources were reported inclusive of the Mineral Reserve. The Mineral Reserve estimate included the M&I Mineral Resources contained within the LOM plan. No Inferred Mineral Resources were included. No Proven Mineral Reserves were estimated for the Muntanga and Dibbwi East pits, the Probable Mineral Reserves were derived from the Indicated Mineral Resources.

The Mineral Reserve estimate was based on the open pit operations at Muntanga and Dibbwi East sandstone-hosted uranium deposits. The basis of the Mineral Reserve estimate was the delivery of ROM material to the central plant or associated ROM stockpile.

The consolidated Mineral Reserve as at January, 1 2025 was estimated at 39.6 Mt at 320 ppm U_3O_8 and comprises only Probable Mineral Reserves. The consolidated Mineral Reserve estimate is shown in Table 15-21.

Table 15-21: Muntanga Mineral Reserve statement as at January 1, 2025

Mineral Reserve class	Tonnes [Mt]	U_3O_8 Grade [ppm]	U_3O_8 Contained [Mlb]
Muntanga pit			
Proven	-	-	-
Probable	8.4	331	6.1
Subtotal	**8.4**	**331**	**6.1**
Dibbwi East pit			
Proven	-	-	-
Probable	31.2	317	21.9
Subtotal	**31.2**	**317**	**21.9**
Total project			
Proven	-	-	-
Probable	39.6	320	28.0
Total Mineral Reserve	**39.6**	**320**	**28.0**

Notes:

1. All figures are rounded to reflect the relative accuracy of the estimate and have been used to derive sub-totals, totals and weighted averages. Such estimates inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, Ukwazi does not consider them to be material.
2. The Concession is wholly owned and operated by GoviEx.
3. The standard adopted in respect of the reporting of Mineral Reserves for the Project, following the completion of required technical studies, is in accordance with the NI 43-101 guidelines and the 2014 CIM Definition Standards, and have an Effective date of January 1, 2025.
4. The OP Mineral Reserves were reported with engineered pit designs using a COG per area varying between 70 ppm U_3O_8 and 85 ppm U_3O_8, which is based on a selling price of USD80 /lb U_3O_8, reference mining cost of USD3.30 /t rock, additional ore mining cost of USD0.55 /t ore, additional ore hauling cost of USD0.18 /t ore/km, incremental depth mining cost of USD0.05 /t/10m bench, processing cost of USD9.00 /t ore, royalty of 5 %, G&A cost of USD1.50 /t ore, port costs of 1.50 /lb U_3O_8 and recoveries varying per location between 74.6 % and 93.3 %.
5. The OP Mineral Reserves were derived from a regularised block models of 5 m x 5 m x 2.5 m (Muntanga) and 10m x 10 m x 2.5 m (Dibbwi East) and include additional dilution and 5 % mining loss.
6. Jaco Lotheringen of Ukwazi is an appropriate "independent QP" as defined in NI 43-101 and completed personal site inspections of the project area.

Several aspects were identified that could impact on the estimated Mineral Reserve, these aspects mainly relate to:

- Risks identified that relate to the estimated Mineral Resources as stated in Section 14.9 of this TR. The Mineral Resource estimate serve as basis of the estimated Mineral Reserve. Any aspect that may impact on the estimated volume or quality parameters of the Mineral Resource will impact the Mineral Reserve.
- Infrastructural risks identified and discussed in Section 18.1.2 of this TR. These risks mainly relate to:
  - Site specific civil geotechnical investigations and conditions
  - Additional study work required relating Machinga River bridge
  - Upgrade of the Muntanga access road for transport of ROM material
  - Potential requirement to import layer works materials
- Risks identified that relate to ESG aspects as stated in Section 20.12 of this TR. These risks mainly relate to
  - Resettlement requirements of several small villages
  - Project permitting schedule requirements based on the updated mine design and ESIA update requirements
  - Scarce water resources and appropriate management measures
  - Impacts on biodiversity
- Mining-specific risks mainly relate to the potential variation in the estimated tonnage an associated ROM grades. The Muntanga and Dibbwi East deposits consists of multiple overlaying ore bodies with internal waste requiring highly selective mining methods. Defined ore and waste is not visually distinguishable with identical RDs of 2.1 t/m³. Appropriate grade control practices must be developed, implemented and maintained during the operational phase to achieved planned production outputs. Not implementing appropriate grade control procedures may have a material impact on the ROM grades resulting in reduced process recoveries and increased unit costs. Reduced process recoveries and increased unit costs may have a material impact on the estimated Mineral Reserve. Appropriate schedule delays (15-hour delay per mining block) were incorporated in the production schedule to cater for grade control requirements.
- Other project related risks discussed in detail in Chapter 25 of this report. These mainly relate to the:
  - Implementation of the resettlement action plan
  - Availability of labour
  - Availability of bulk electrical supply
  - Logistics
  - Market related risks.

16. Mining methods

16.1. Mine site layout

The mine site layout is shown in Figure 16-1. The main features shown are:

- Muntanga and Dibbwi East pits
- Muntanga and Dibbwi East waste dumps
- Surface haul routes
- The Central complex, including ROM tip, crushers and conveyors, HLF, processing plant, offices and mining workshops and offices
- Stockpile area.

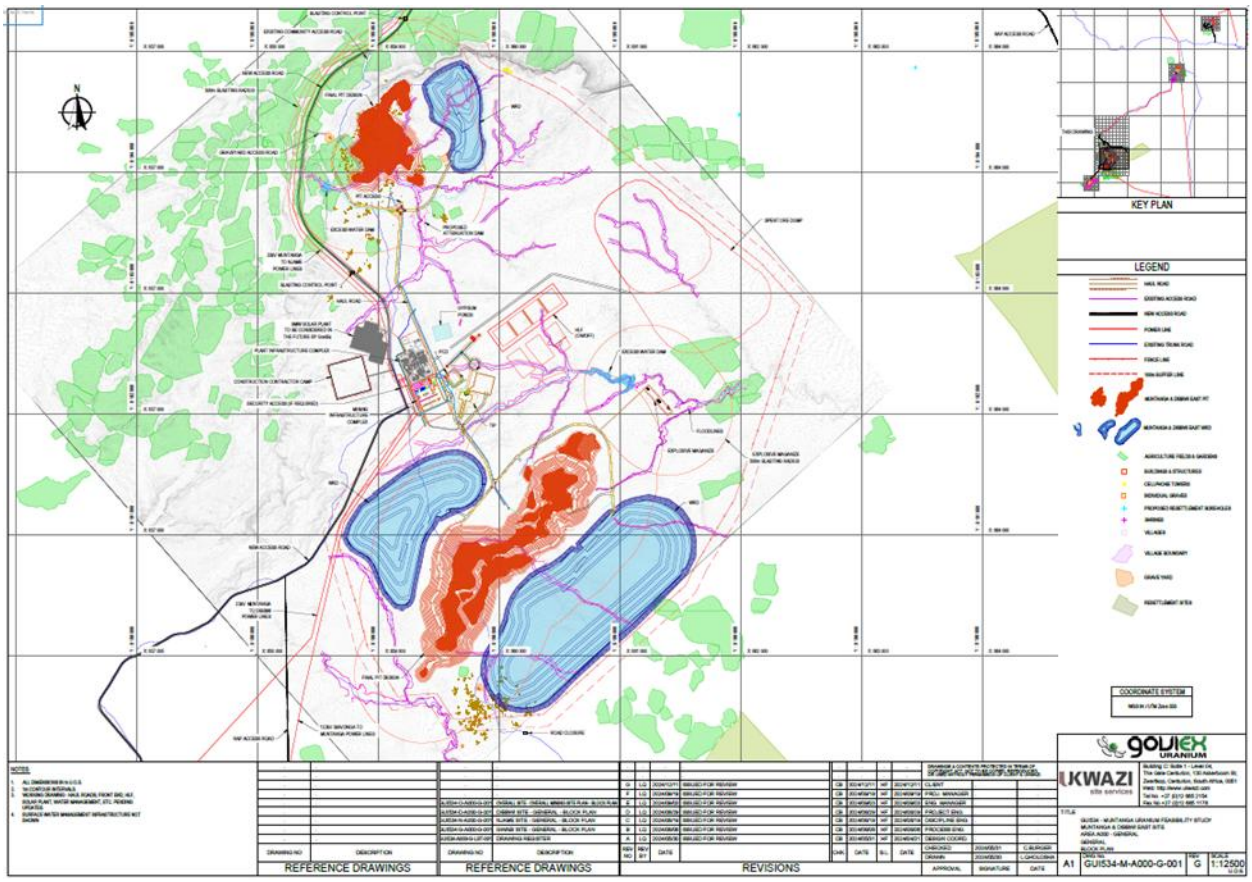

Figure 16-1: Mine site layout

16.2. Mining method and operations

16.2.1. Mining details

Ore production will commence with mining at the Muntanga deposit, due to its low S/R at 1.21 x, and then continue simultaneously at Dibbwi East deposit with a 4.29 S/R. Once mining at Muntanga is depleted, Dibbwi East will serve as the sole source of ore feed.

Muntanga is mined by means of a cutback mining approach starting with a north-eastern boxcut and progressing westwards in a series of 40 m-wide mining benches. In-pit dumping takes place once sufficient void space is created. Since the ore body outcrops at the northern side of the pit and dips at a haulable gradient of approximately 6°, access is either via the outcrop or by means of temporary inpit ramps with a minimum of a 120 m space between any two mining benches allowed.

While Muntanga initially provides the bulk of the ore, Dibbwi East makes up the difference in the monthly target. Dibbwi East is mined in three pushbacks as described in section 15.9.4.1. The table below indicates the start and end period of each pit/ pushback.

Table 16-1: Mining start and end periods

	Start period [month]	End period [month]
Muntanga	1	54
Dibbwi East - Pushback 1	1	100
Dibbwi East - Pushback 2	1	138
Dibbwi East - Pushback 3	1	144

As depicted in Table 16-1, early waste stripping of Pushback 2 at Dibbwi East helps ensure monthly ore targets are achieved once Muntanga is depleted. The initial mining that takes place in Pushback 3 is limited since Pushback 1 makes use of a surface ore haul road cutting across Pushback 3 to reduce the overhaul haul distance and to save on cost.

The total loading capacity is ramped up over a period of 12 months to about 17.3 Mtpa for the remaining LOM.

16.2.2.Mining methods and operations

Mining follows conventional drill and blast, shovel and truck mining practice. The sequence of mining activities is conventional and is generally as follows:

- Grade control drilling delineates the ore zones
- A grade control model is developed from which blast limits and digging blocks are designed
- Ore, waste or mixed blocks are blasted to design, according to layouts based on hole patterns and powder factors to suit ground conditions
- Trim blasts and perimeter blasting techniques are used to ensure pit wall profiles are cut to the correct angle and to minimise wall damage
- Diesel/ hydraulic excavators load the blasted rock into a fleet of articulated dump trucks (“ADT”) of 25 m^3 capacity
- Ore is hauled directly to the central plant, whilst waste is hauled to surface dumps or dumped in pit once space is available.

16.2.3.Drilling and blasting

Production drilling and blasting will be carried out by the owners’ team mining personnel. DTH explosives quotations were sourced from three suppliers, of which two provided comprehensive proposals and are recommended for further negotiations and discussions. Pricing is based on these proposals. Emulsion will be delivered to site and stored in 50 t silos, from where it will be transferred to dedicated MMU trucks for DTH delivery as required.

The appointed blaster, assisted by blasting assistants, will be responsible to oversee the drilling and blasting operation including priming of blast holes, charging up, stemming, and tie-up, up to final detonation.

16.2.3.1. Blast designs

For the purposes of blast designs, every 40 x 40 x 10 m blast block was subdivided into:

- Ore blocks (any blast block that contains ore or a combination of ore and waste) – 44 % of total
- Waste blocks (pure waste blocks that contain no ore) – 46 % of total
- Trim blocks – 10 % of total.

From the available geological data, it was concluded that the rock is reasonably soft and highly likely to easily fragment with blasting. As indicated in Figure 16-2, a typical drill core is easily crushed by foot, confirming that the rock is soft with a resultant rock factor of five selected as per Table 16-2.

Figure 16-2: Drill core test

Table 16-2: Rock factor classification

Classification	Rock factor
Soft rock	0 ~ 5
Average rock	6 ~ 9
Very hard rock	10 ~ 12

Since both ore and waste material have similar rock properties and density (2.1 t/bcm) and occur within the same blast bench, no differentiation was made between the blast designs of the two material types.

An 165 mm hole diameter was selected for both production drilling and trim blast patterns. The final blast designs selected for production blocks, trim blocks and pre-splits are tabled in Table 16-3.

Table 16-3: Blast design and powder factor

Blast design	Production	Trim	Presplit[1]
Burden (25-35 x the hole Ø) - m	5.50	5.00	1.00
Spacing factor (1-1.5 x the burden)	1.50	1.50	
Spacing - m	8.00	7.00	2.00
Check spacing factor (1-1.5 x the burden)	1.45	1.40	
Depth (bench height) - m	10.00	10.00	10.00
Subdrill factor (0.1-0.3 times the burden)	0.10	0.10	0.00
Subdrill – m	0.55	0.50	
Hole Ø – mm	165.00	165.00	165.00
Stemming (guide: 20-30 x the hole Ø) – m	3.30	3.30	
Stemming applied - m	3.00	3.00	9.74
Powder factor calculation			
Explosives in hole density	1.20	1.20	1.20
Charge per meter – kg	25.66	25.66	25.66
Charge Length – kg	7.55	7.50	0.26
Charge per hole – kg	193.76	192.48	6.60
Actual Powder Factor – kg/bcm	0.440	0.550	0.330

1. Presplit blasting make use of five innopak super (50x580mm) cartridges

To provide improved blasting initiation flexibility, an 80:20 split assumption was made between electronic and conventional pyrotechnic basting with the potential to optimise once the Project is operational and prevailing ground conditions are better understood. Presplit blasting designs are purely based on pyrotechnic systems.

The selected blast design predicts that 50 % of all material will be smaller than 331 mm and 80 % of all material will be smaller than 768 mm as shown in Table 16-4. It is expected that that post-blasting activities will result in further fragmentation considering the soft nature of the material.

Table 16-4: Blast design fragmentation

% Passing	Size [mm]
0.0	0.0
10.0	50.3
20.0	106.4
30.0	170.1
40.0	243.7
50.0	330.6
60.0	437.1
70.0	574.3
80.0	767.7
90.0	1 098.4
99.0	2 196.8

Drill rig quotations were sourced from two Zambian based OEM suppliers, namely Epiroc and Sandvik. The suitable rigs are:

1. Epiroc – FlexiROC D60FS and
2. Sandvik – Leopard DI650i

Table 16-5 compares the two drill rigs.

Table 16-5: Drill rig comparison

Feature	Sandvik - Leopard DI650	Epiroc - FlexiROC D60-10SF
Drilling type	High pressure	High pressure
Primary use	Surface mining	Surface mining
Drilling diameter (mm)	115 - 203	110 -178
Depth capability (m)	53.6	55.5
Compressor pressure (bar)	25 – 30	25 max
Hammer size	4”; 5”; 6”	4”; 5”; 6”
Engine power (kW)	403	353
Weight (kg)	25 100	22 600
Mobility	Crawler-mounted	Crawler-mounted
OEM useful life (hours)	27 000	20-25 000

As the contribution of the drill rigs to capital costs is insignificant relative to other mining equipment, Sandvik was selected as the preferred supplier. Further discussions at the next phase are suggested with both suppliers with potential discount negotiations.

Figure 16-3 shows a drill rig engaged in blasthole drilling.

Figure 16-3: An example of a drill rig (source: mining magazine.com)

At peak production, the blast hole drill fleet consists of four DTH hammer drills, drilling 165 mm holes.

The fleet is used to drill trim, ramp, presplit and production holes. The fourth unit is allocated to the technical department for geological drilling, but provides additional production capacity or can stand in for another rig undergoing repair or maintenance if required.

16.2.4.Loading and hauling

The selected SMU sizes are ideal for operations requiring high selectivity and precise grade control but demand smaller, more manoeuvrable loading equipment which increases operational complexity. Benches must accommodate precise access to these smaller units, which could limit the use of large equipment which may not fully utilise its capacity, leading to inefficiencies.

A typical mining face (SMU blocks of 5 x 5 x 2.5 m) is illustrated in Figure 16-4, highlighting the importance of selecting appropriately sized loading equipment to mitigated against ore dilution and/ or ore loss. The mining sequence should consider blasting 10 m benches of suitable length to establish bench access post blasting by means of temporary ramps constructed with inpit dozers, whereafter material can be loaded by excavators in in 4 x 2.5 m flitches.

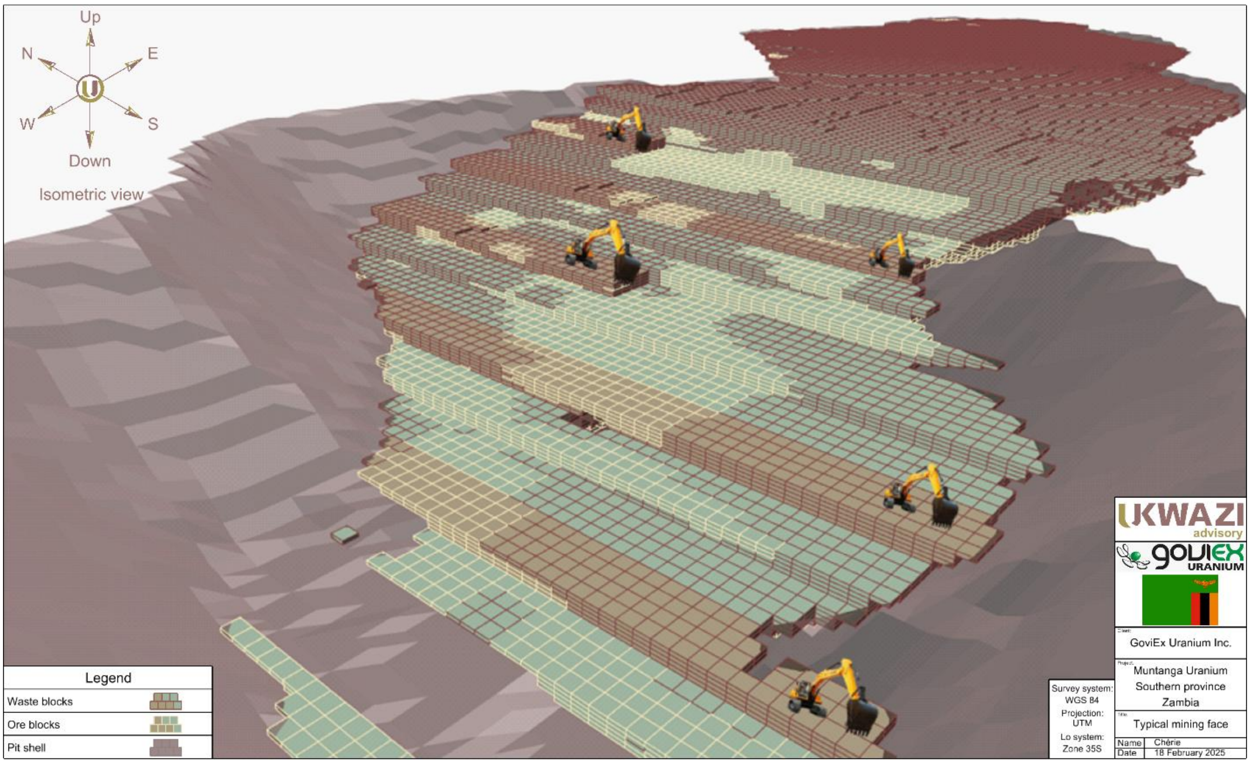

Figure 16-4: Typical mining face

16.2.4.1. Primary load and haul equipment

To meet the production requirements with sufficient levels of selectivity without congesting the pits, 90 t to 120 t excavators with a bucket size of 4.5 m^3 to 6 m^3 were considered. Table 16-6 shows a summary of excavator models from various suppliers considered.

Table 16-6: Excavator comparison

Parameter	Cat 395	Volvo EC950	Komatsu PC1250	Lieberr 9100
Power (kW)	405	446	560	565
Operating weight (tonnes)	95	90	116	113
Rock Bucket Capacity (SAE)	5.7	5.6	6.7	7.5
Arm length (m)	2.92	2.95	3.4	3.2
Bucket width (m)				2-3
Suitable 2.5 bench height (m)	Very good	Very good	Good	Good
OEM useful life (Hours) – supplied	30 000	20 000[2]	48 000	30 000

1. Volvo supplied life cycle costs (LCC) rates based on a service and maintenance contract up to 20 000 hours.

The shorter arm length of both the Cat and Volvo excavators is better suited for the 2.5 m bench height.

Suppliers were requested to supply the quantity of supported, in country operational units as shown in Figure 16-5 to assess the impact of spares availability where more units will likely result in higher OEM in-country spares stock levels, which would mitigate against excessive downtime caused by unavailability of parts due to unexpected breakdowns. Three potential OEM suppliers responded.

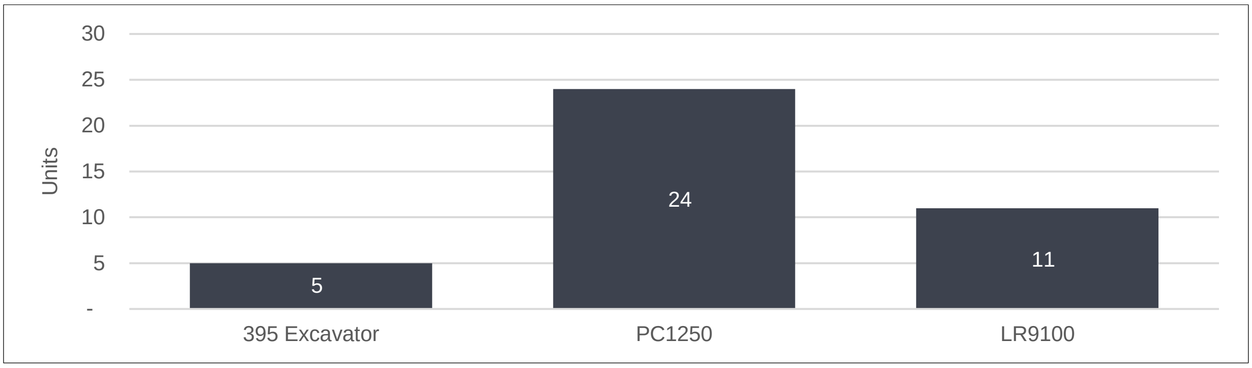

Figure 16-5: OEM operational excavator units in Zambia

With cognisance of the ground conditions and terrain with relation to the ore body and pit floor, ADTs are the preferred option and match well with the loading units.

ADTs typically come in 30 t to 40 t capacity, up a maximum 55 t for a Volvo A60 unit. Table 16-7 shows that for a full capacity swing cycle, a Cat 395 excavator pairs best with Cat 745 or similarly sized dump truck.

Table 16-7: Excavator truck matching

	Cat 740	Cat 745	Volvo A60
Passes (nr)	4	5	6
% Fill volume	75 %	86 %	76 %
% Fill tonnes	75 %	83 %	75 %

Based on loose rock density 1.6 t/m³, 85 % truck fill factor and 75 % excavator bucket fill factor

Although similar matching may be achieved by a Liebherr 9100 excavator and Volvo A60 truck, the 45 t size ADTs have a wider availability from different suppliers to choose from and therefore offer some risk reduction and higher negotiating powers when engaging suppliers during the next phase.

Table 16-8 shows a summary of haul truck models considered from various suppliers.

Table 16-8: Haul truck comparison

Parameter	Cat 745	Volvo A45	Komatsu HM400	Bell B45
Power (kW)	376	354	334	390
Bin Capacity SAE (m³)	25	25.1	24	25
Truck load over height (meter)	3.2	3.2	3.2	3.3
Payload (tonnes)	41	41	40	41
OEM useful life (Hours) – supplied	25 000	20 000[2]	30 000[3]	12 000[4]

1. Volvo supplied life cycle costs (LCC) rates based on a service and maintenance contract up to 20 000 hours.
2. Komatsu HM400 life of 30 000 is an outlier and may be overestimated, negatively impacting equipment availability.
3. Bell supplied life cycle costs (LCC) rates based on a service and maintenance contract up to 12 000 hours. This is a low useful life compared to other suppliers, and will require more frequent replacements. The OEM seems hesitant to do major repairs on equipment.

Suppliers were requested to supply the number of supported in country operational units as shown in Figure 16-6 to access the impact of spares availability where more units will likely result in higher OEM in-country spares stock levels, which would mitigate against excessive downtime caused by unavailability of parts due to unexpected breakdowns. Caterpillar and Komatsu responded with number of units indicated, shown in Figure 16-6.

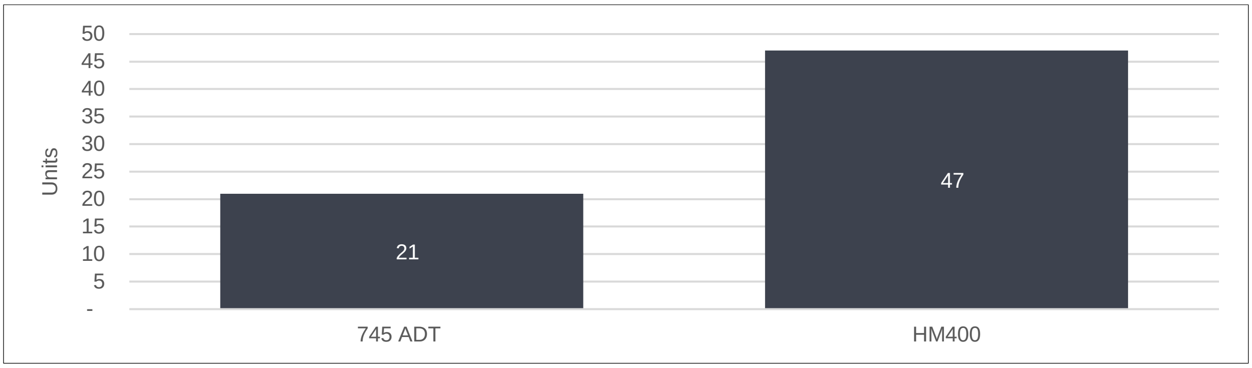

Figure 16-6: OEM operational ADT units in Zambia

To assist with the final equipment selection further high-level comparisons were done based on expected equipment utilisation, with suitable adjustments for labour in equipment supplied on repair and maintenance contracts.

16.2.4.2. Support equipment

The load and haul operation will be supported by dozers, graders and water bowsers, to ensure loading faces are clean, bench access and haul roads are established and maintained in addition to waste dump maintenance and levelling of tipped material.

To limit the quantity of dozers while still having sufficient capacity for inpit bench and floor preparation and waste dump maintenance and material levelling, Cat D8 dozers were selected.

Table 16-9 shows a summary of dozer models considered from various suppliers.

Table 16-9: Dozer comparison

Parameter	Cat D8	Komatsu D155	Liebherr 756 Litronic
Power (kW)	271	268	275
Operating weight (tonnes)	40	42	36
SU blade capacity (m³)	8.7	9.4	8.9
OEM useful life (Hours) – supplied	35 000	30 000	24 000
In country operational supported units – supplied	13	28	3

The longer life of the Cat D8 suits the project better as no replacements are required. Cat D8s were selected, but it is recommended that other suppliers are not ruled out the in future discussions when project implementation begins.

Dust suppression is by means of 23 000 litre articulated water bowsers. As shown in Table 16-10, two companies provided information for graders. Caterpillar was the final selection.

Table 16-10: Grader comparison

Parameter	Cat 140 series	Komatsu GD675
Power (kW)	143	163
Blade width (meter)	3.7	4.32
OEM useful life (Hours) – supplied	30 000	30 000
In country operational supported units – supplied	42	30

16.2.4.3. Rehandle equipment

Initial ore mined before plant startup is stockpiled on a ROM stockpile to be loaded and hauled to the crusher when required. The ROM stockpile is situated at the bottom of the raised ROM tip near the HLF crushers as shown in Figure 16:1. The one-way haul distance from the ROM pad to the crushers is 800 m.

To provide maximum flexibility for rehandle operations, FELs with a sufficient lift height were selected to load the Cat 745 ADTs.

A summary of the FELs considered is show in Table 16-11.

Table 16-11: FEL comparison

Parameter	Cat 988K	Volvo L350H	Komatsu WA600
Power (kW)	401	297	374
Operating weight (tonnes)	51	51	53
Clearance at maximum lift (meter)	3.6	3.8	3.7
OEM useful life (Hours) – supplied	35 000	20 000	30 000
In country operational supported units – supplied	3	Not Indicated	11

Komatsu was was the final selection.

16.2.4.4. Minor equipment

Quotations were sourced from OEM suppliers for the following minor mining and support equipment as shown in Table 16-12. The units selected are highlighted in green.

Table 16-12: Minor equipment and workshop equipment suppliers

Equipment	Cat	Volvo	Komatsu	Bell
Rock breaker	C330	300D	PC350	SK300
Diesel bowser	C730	A30	HM300	B18
Mobile lighting plant	Minor equipment suppliers			
TLB	C428		WB93	JCB 3DX
40-seater site-based busses	Minor equipment suppliers			
Light delivery vehicles	Minor equipment suppliers			
Flatbed	Minor equipment suppliers			
Lowbed	Minor equipment suppliers			
Forklift	Minor equipment suppliers			
Tyre handler	Komatsu WA500 unit fitted with tyre handler attachment			

16.2.5.Waste dumping

Appropriate ex-pit waste dumps were designed for the Muntanga and Dibbwi East mining areas to minimise hauling distances and keep the waste dump as close as possible to the final void estimates for backfilling at the end of the LOM if required in future. The positions of the waste dump were dictated by:

- Position of the ore body (both existing Inferred Mineral Resources and areas with exploration potential) and mining area
- Surface terrain and water stream positions
- The practicality of controlling stormwater
- Position of the initial box cut position.

The selected 20° overall slope angle of the waste dumps aligns with the end of mine rehabilitation slope angle to reduce material handling costs during post-mine WRD rehabilitation.

Details on the WRD designs are supplied in Section 16.2.4.3

16.2.6.Mine planning and operations

The following information relates to the detail needed to be considered for designing surface layouts and practical mining phases around the pit shell outline.

16.2.6.1. Mine design parameters

The basic mine design criteria ("MDC") relating to pit slope design are described in Section 15.9.1. The following parameters were adopted to design the in-pit and surface haul roads.

- Road width – 20 m with a travel width of 15 m
- Ramp gradient – 10 %.

The geometric design of the haul roads is shown in Figure 16-7, as related to truck width.

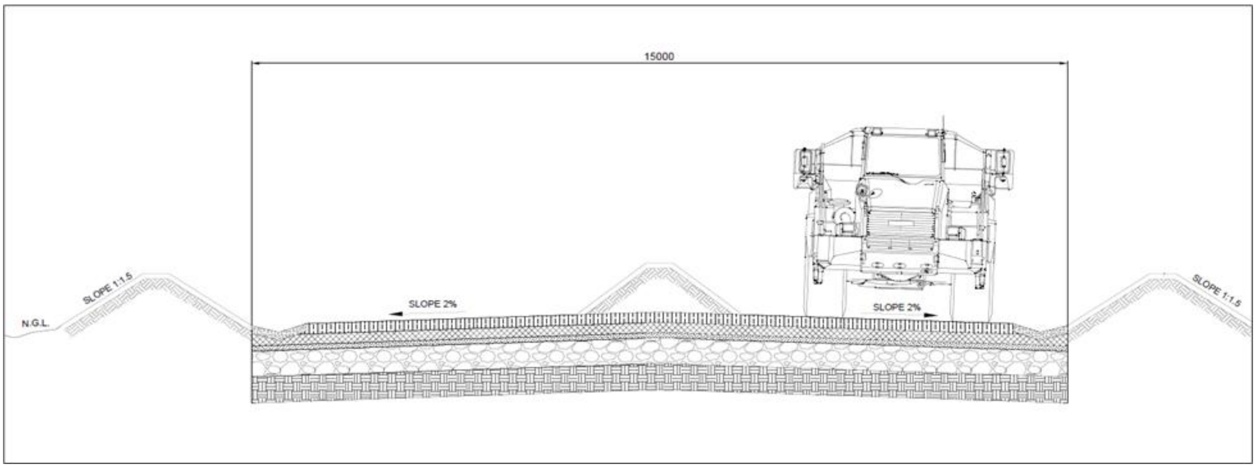

Figure 16-7: Typical haul road section

16.2.6.2. Geotechnical engineering

Section 26.1 describes the design parameters and domains evaluated to derive the MDC relating to the pit slope design.

16.2.6.3. External waste dump design

The waste dump slopes were designed with 15 m high benches, with 40 m wide berms and with wall angles at the natural angle of repose (assumed to be 37°). The toe of the dump nearest to the pit was designed to be at least 100 m from the pit crest to conform with geotechnical recommendations.

Figure 16-8 and Figure 16-9 show the ex-pit waste dump locations for the mining areas with Table 16-13 depicting the respective waste dump capacities.

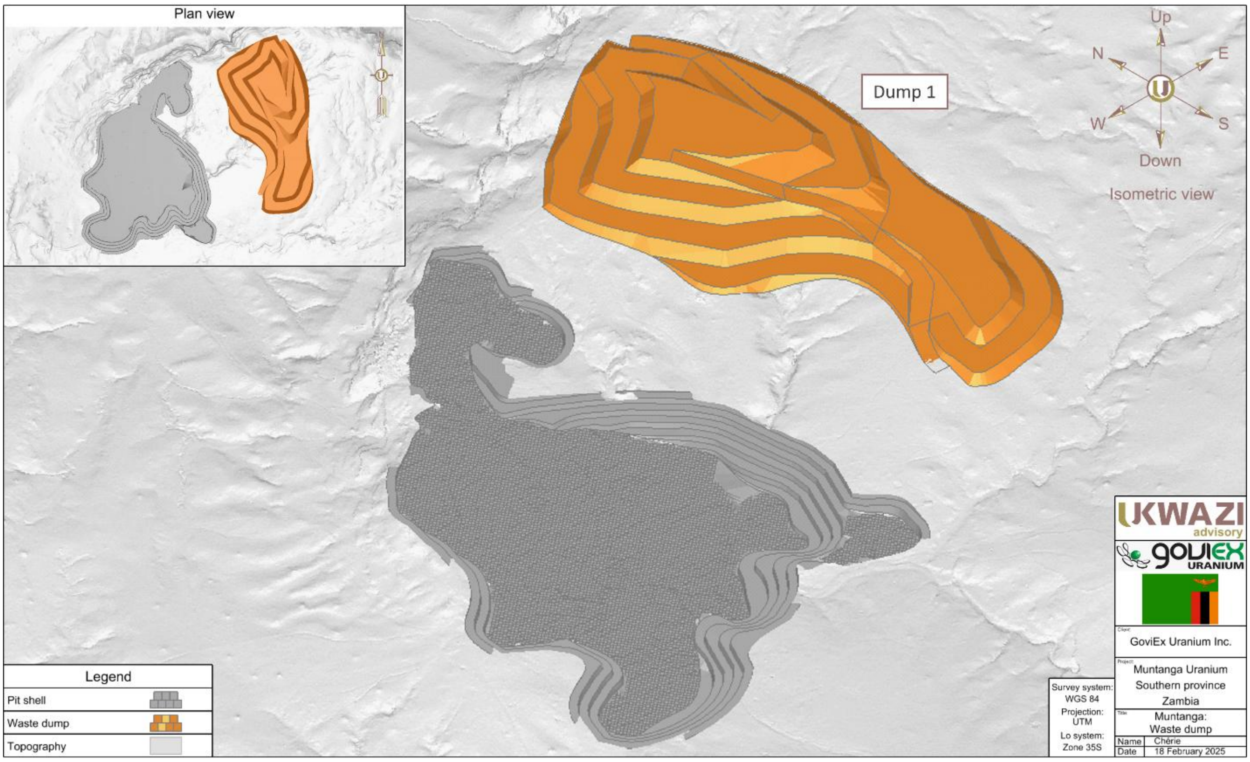

Figure 16-8: Muntanga waste dump

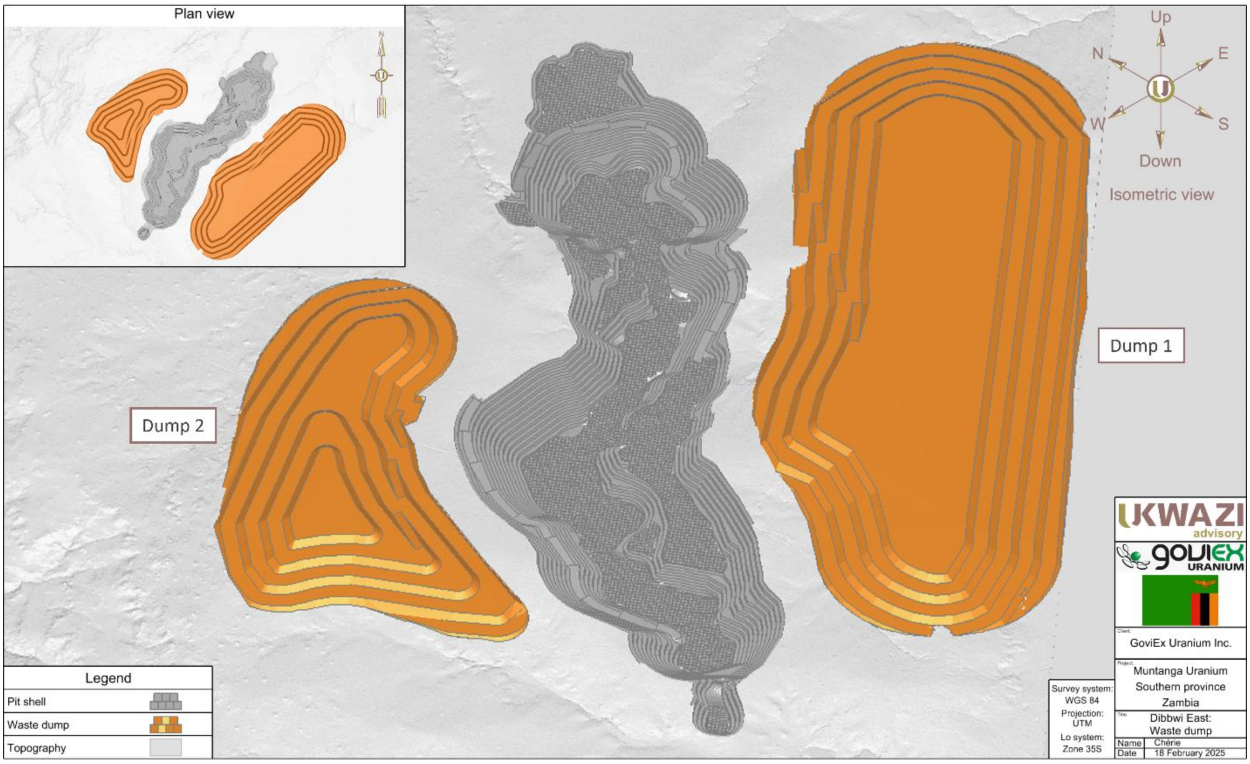

Figure 16-9: Dibbwi East waste dumps

Table 16-13: Waste dump capacities

Area	Dump	Design file name	Capacity volume [million BCM]	Used volume [million LCM]
Muntanga	Muntanga Dump 1	Muntanga_dump_v3_Solid_Cut.DXF	4.3	2.9
Dibbwi East	Dibbwi East Dump 1	Dibwieast_Dump1_Cut.DXF	59.1	53.3
	Dibbwi East Dump 2	Dibwieast_Dump2_Cut.DXF	16.2	16.2
Total			**79.6**	**72.5**

LCM = BCM * 1.3 (swell)

All waste scheduled at the start of each pit was scheduled for placement on the designated waste stockpiles until sufficient in-pit space was available to commence with backfill in the pits by virtue of the staged sequence of the mining phases. Detail on the destination schedule results are discussed in Section 16.2.6.4.1

16.2.6.4. Ore stockpiling

An outcome of the production scheduling described in Section 16.2.5.1 is the requirement for an ore stockpile in the first year of mining operations. This stockpile is on surface, with an 800 m one-way haul distance from the crusher, as shown in Figure 16-10. Initially this stockpile will be used as a start-up stockpile while the plant is operating at 50 % capacity and thereafter as short-term operational stockpiles when required to de-risk mining operations. Figure 16-10 and Figure 16-11 below show the initial start-up stockpile with a value in year one of 105 kt whereafter it reduces to zero for the remainder of the LOM.

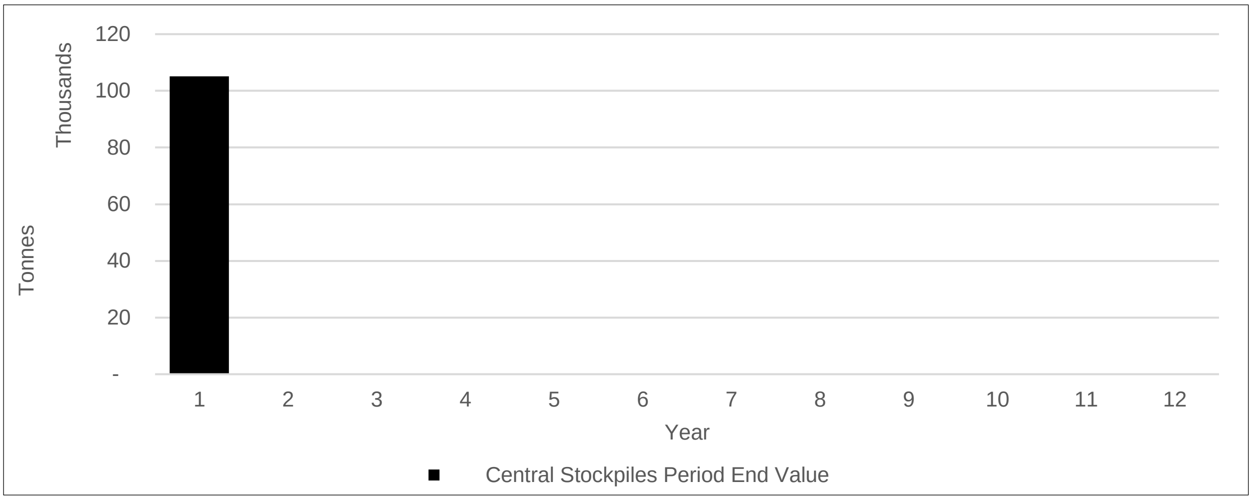

Figure 16-10: Central plant stockpiles

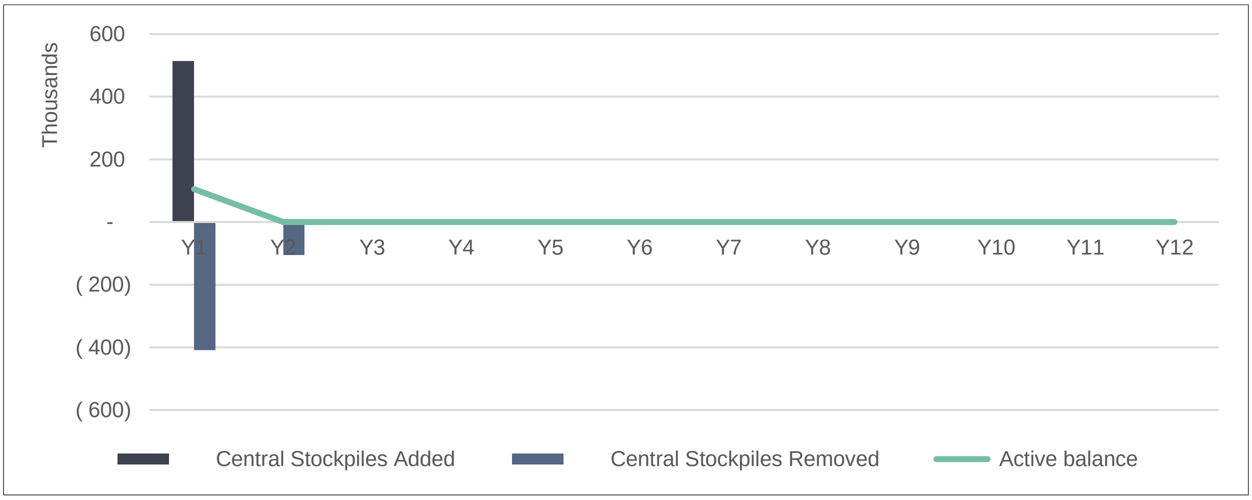

Figure 16-11: Active stockpiles

16.2.6.5. Hydrology – pit dewatering

Details of the pit dewatering system are shown in Section 18.3.8.

16.2.7.Mining and processing schedules

With the completion of the detailed ultimate pit designs, detailed LOM production scheduling was completed.

Mine production scheduling is the process of assigning physical mine production units and specified equipment fleets/ total capacities to periods of time, and reporting the results by periods in terms of tonnes of ore and waste with associated grades. The process results in a production schedule that reports the mine production units per period from the start of the schedule to the depletion of the ultimate open pit "OP. The OP mine production scheduling was done using RPMGlobal Stratigraphic Metal Solutions ('SMS") scheduling software, which addresses the complex challenges involved in mine scheduling and optimises each scheduling period at a time.

16.2.8.Life of mine schedule

16.2.8.1. Modifying factors

The following modifying factors were considered when developing the LOM schedule:

- Dilution
  - Dilution is defined as the waste material added during the mining process. Site-specific dilutions are added to the in-situ Mineral Resources to define the quality of the practically mineable unit. Dilution reduces the overall grade of the practically mineable cut when compared to the original geology by adding material adjacent to the mineralised material. The tonnage containing mineralised material therefore increases relative to the Mineral Resource, but at a lower grade
- COG
  - Whittle COGs are used as the basis for the ore classification process and are applied to the SMU diluted grade. Table 16-14 shows the COGs that were applied.

Table 16-14: COGs applied in developing the LOM schedule

Area	COG USD 80 /lb U_3O8 [ppm]
Muntanga	76.6
Dibbwi East	70.1

- Mining loss
  - The estimation of mining loss requires an understanding of the MRE methodology, structural geology, blasting practice, and loading fleet. The dip, strike, width and length of the zones within the deposit are the most significant considerations for mining loss and mining dilution. In addition to the absolute values, the variability in geometry has a significant influence on the efficiency of ore mining. A 5 % mining loss was applied to the ore in each mining area
- Geological loss
  - The geological loss is an indication of MRE error, modelling inaccuracies or structural complexity of the deposit. The confidence level of the project study, the complexity of the deposit and the rigidness of the topography normally influences the assumption of geological loss. No geological loss was applied, based on guidance from the Mineral Resource QP. This is not an unreasonable approach, based on the three-dimensional modelling and MRE methodology applied.

16.2.8.2. Time usage models

The assumed time usage model ("TUM") for the OPs is shown in Table 16-12. The effective working hours of the pits is approximately 4 568 hours per year.

Table 16-15: Effective time usage model (average)

Operating hours	Unit	Value
Working shifts	no.	2
Hrs per shift	hrs	10
Days per month	days	30
Number of months per year	no.	12
Total days per year	no.	365

Operating hours	Unit	Value
Total hrs per month	hrs	608
Total hrs per year		**7 300**
Environmental losses		
Weather	days/year	11.14
Public holidays	days/year	15
Industrial action	days/year	3
Total days lost	days/year	29.14
Total "off" time loss	hrs/year	583
Site scheduled hrs	**hrs/year**	**6 717**
Site-specific losses		
Equivalent scheduled shifts	number of shifts	672
Mobilise and startup checks	minutes/shift	30
Lunch and other breaks	minutes/shift	60
End of shift, blast & demobilise	minutes/shift	30
Total utilization loss	minutes/shift	120
Utilization loss machine	hrs/year	1 343
Scheduled hours utilised	**hrs/year**	**5 374**
Availability & efficiency		
Machine availability	percentage	85 %
Machine operating hours	hrs/year	4 568
Job efficiency	percentage	100 %
Effective hours	**hrs/year**	**4 568**
Overall, job efficiency	percentage	63 %
Minutes/effective hour	**min/hr**	**37.54**
Actual working hours/day	**hrs/day**	**12.51**
Actual working hours/month	**hrs/month**	**373**
Equipment utilisation (operational factors)	percentage	100 %
Direct operating hours (“DOH”)	**hrs/month**	**373**

16.2.8.3. Open pit scheduling parameters

To support the practicality of the OP LOM schedule, rules were applied in the OP scheduling software; RPMGlobal SMS. Hauling distances are reported from the scheduling sources using the coordinate of the mining block, ramp locations, pit exits, surface haul roads, waste dump entrances and coordinates of the destination block. The following specific rules were applied in the scheduling software:

16.2.8.3.1. Bench deployment rules

In the scheduling process, various dependencies and rules were put in place to replicate practical mining. This includes for example, safe working area, no undermining and loader density per bench.

The safe working distances for loaders impose a restriction on the radius within which they can operate on a bench. Each resource must maintain a minimum distance from others, defined by a projection around the block it occupies. This projection extends from all sides of the block by the specified minimum distance and continues along the associated bench. Since the exact location of the resource within the block is unknown, the projection is applied relative to the block, this is illustrated in the Figure 16-12. A minimum working area of 40 m was applied.

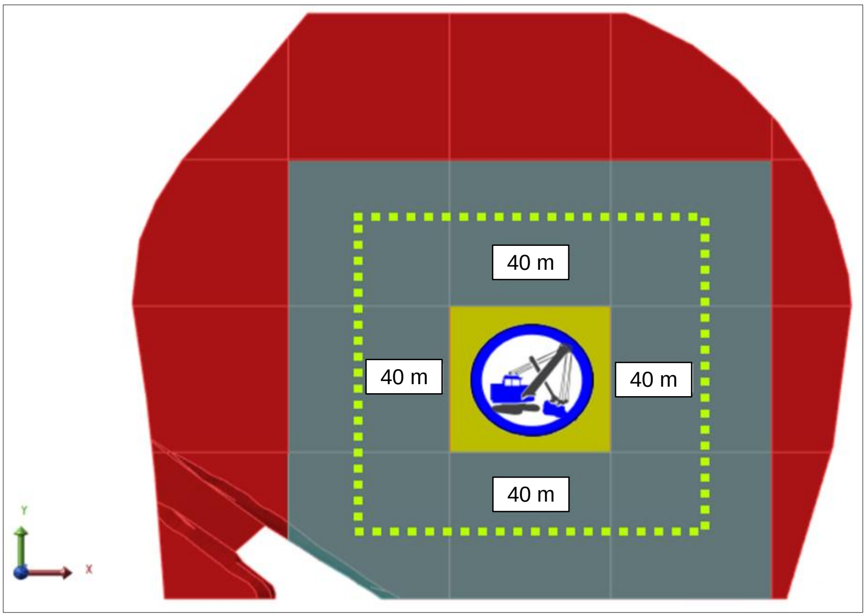

Figure 16-12: Safe working area for loaders

Vertical progression rules establish automatic dependencies between blocks across benches or lifts, defining predecessor-successor relationships as illustrated in Figure 16-13. These rules can be based on either mining or filling requirements. Areas were limited to 100 m to 120 m as to allow space for temporary inpit access ramps between various benches.

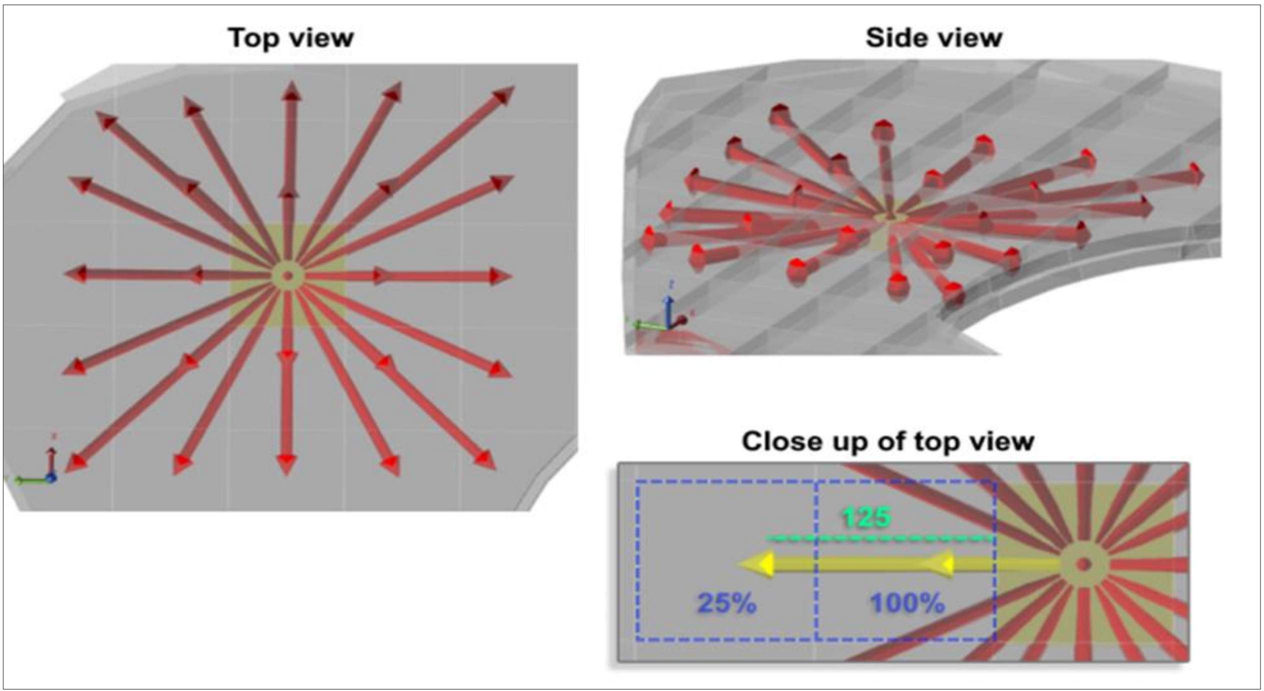

Figure 16-13: Vertical progression rules

16.2.8.3.2. Loader density limits

Maximum loader density limits per bench and pushback were also set per mining area as indicated in Table 16-16.

Table 16-16: Loader density limits (maximum)

Area	Bench Limit	Pushback limit
Dibbwi East – Pushback 1	2	5
Dibbwi East – Pushback 2	3	8
Dibbwi East – Pushback 3	2	4
Muntanga	3	5

16.2.8.3.3. Delay allowance

Delays are non-productive activities that must be scheduled. These occur under specific conditions and last for a defined duration. Delays can account for instances such as equipment downtime or relocation. A grade control delay of 15 hours was used in the schedule and must be completed on every mining block prior to it being available for mining.

16.2.8.3.4. Hauling routes

To estimate accurate hauling distances, it was important that the pit ramps, waste dump ramps and surface roads were appropriately reflected in scheduling haulage network software called RPMGlobal Hauled. Figure 16-14 and Figure 16-15 show the configuration of ramp and surface roads for each area design.

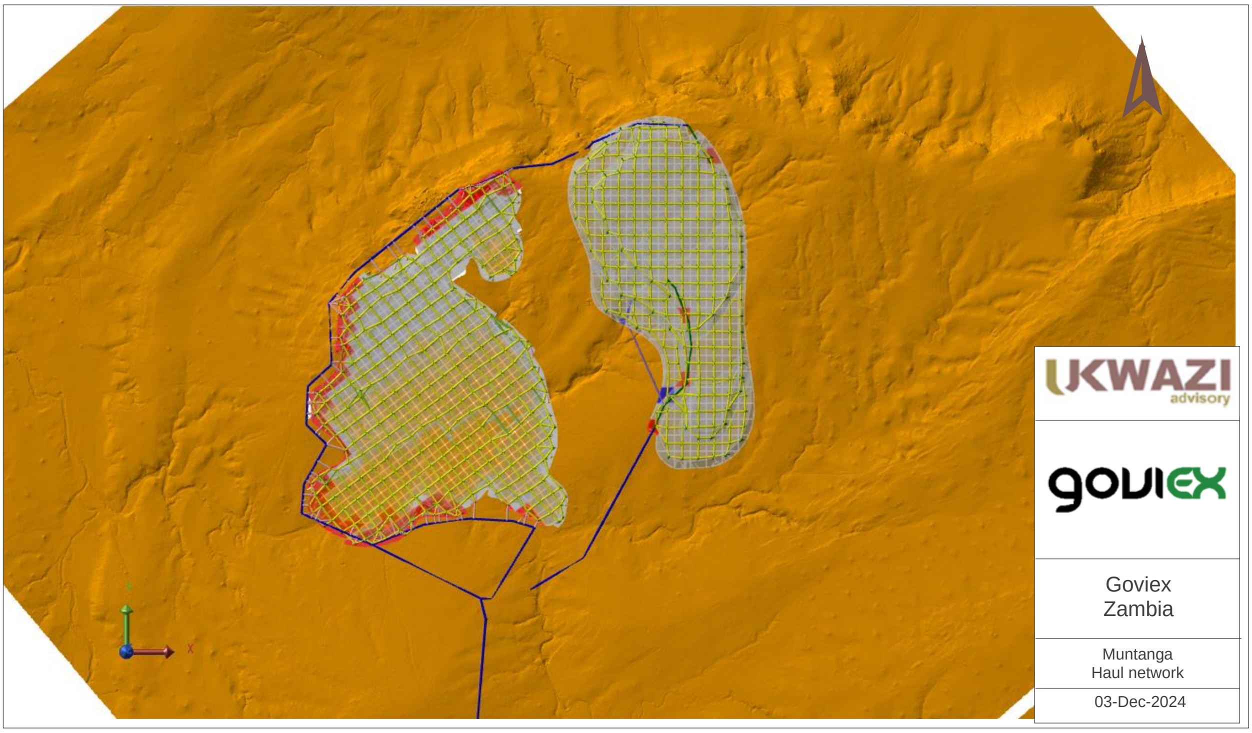

Figure 16-14: Muntanga haulage network

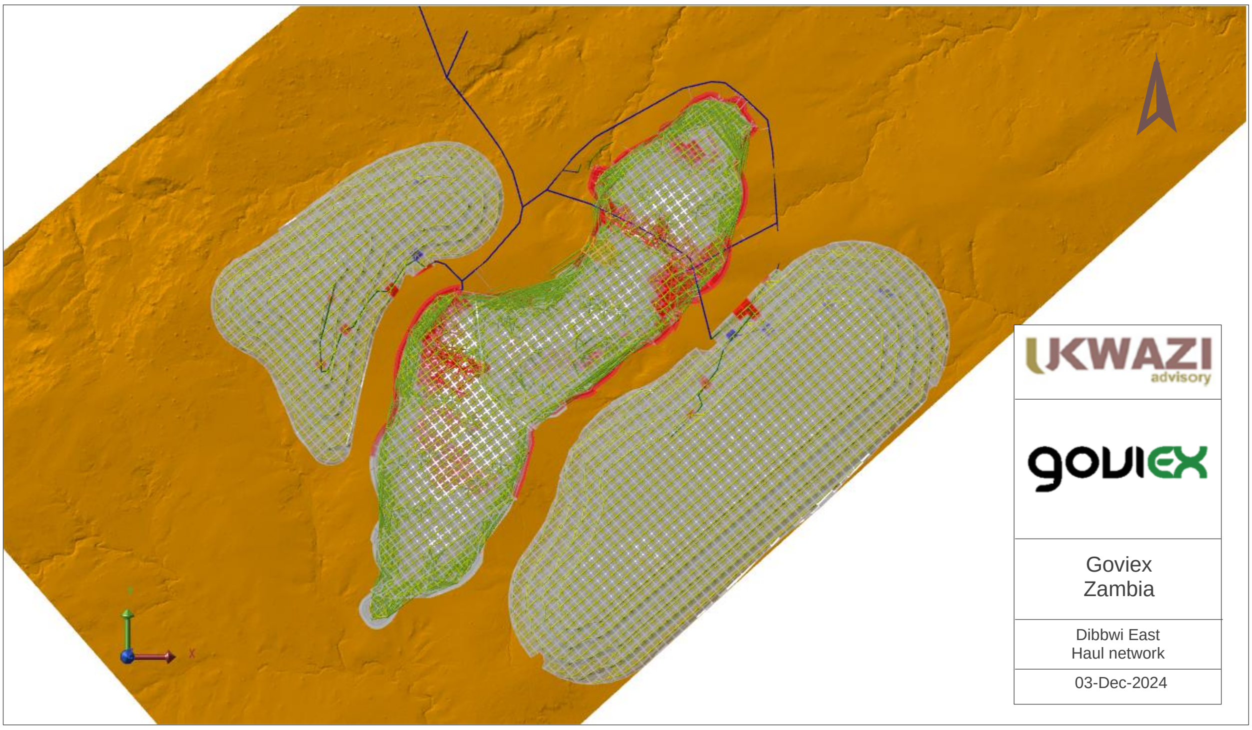

Figure 16-15: Dibbwi- East haulage network

16.2.8.3.5. Scheduling equipment

The mining operation will apply a typical OP load and haul mining method that requires ore and waste blasting. The mining sequence consists of de-vegetation, construction of haul roads, diversion drains, removal of overburden to waste stockpiles located adjacent to each of the pits, followed by mining of the ore and delivery directly to the beneficiation plant. The equipment in Table 16-17 is used in the resource-based schedule.

Table 16-17: Loading equipment schedule productivity

Description	Bulk	Selective [1]	Average
Contribution	94 %	6 %	100 %
Material in situ density	2.10	2.10	2.10
Swell	30 %	30 %	30 %
Loose density	1.62	1.62	1.62
Loader BH Cat 395F			
HD Bucket capacity	5.70	5.70	5.70
Bucket fill [2]	75 %	40 %	73 %
Hauler ADT Cat 745			
Heaped capacity (m^3)	25	25	25
Number of passes	5.0	9.0	5.0
Truck tub factor (m^3)	21	21	21
% of truck capacity [3]	86	82	83
Cycle time			
Cycle time/loader bucket load (seconds)	27	33	27.4
Loading cycle time (seconds)	135	297	137
Spotting time (seconds)	60	60	60.0
Total cycle time (seconds)	195	357	197
Total cycle time (minutes)	3.25	5.95	3.3
Efficiency factor	100 %	80 %	99 %
Minutes per hour	60	48	59
Passes per hour (number)	18	8	18
Nominal Production Rate (tonnes per hour)	637	267	599
Availability	85 %	85 %	85 %
Utilisation	As per the time usage model		
Effective Production Rate (tonnes per hr)	542	227	509

1. Selective percentage is based on an area of 300 mm around ore zones
2. Hydraulic excavator bucket fill for well-blasted rock 60-75 % (Caterpillar)
3. Fill factor of hauler body that is used for well-basted rock 80-95 % (Caterpillar)

16.2.8.3.6. Production schedule material flow strategy

All waste material scheduled from the pit will initially be placed on the designated waste dumps. As pits are depleted, placement of waste material back into the pits was scheduled. The ore scheduled from the pit is fed into the Central plant directly from the central pits (Muntanga and Dibbwi East). Figure 16-16 shows the material flow strategy for the Project.

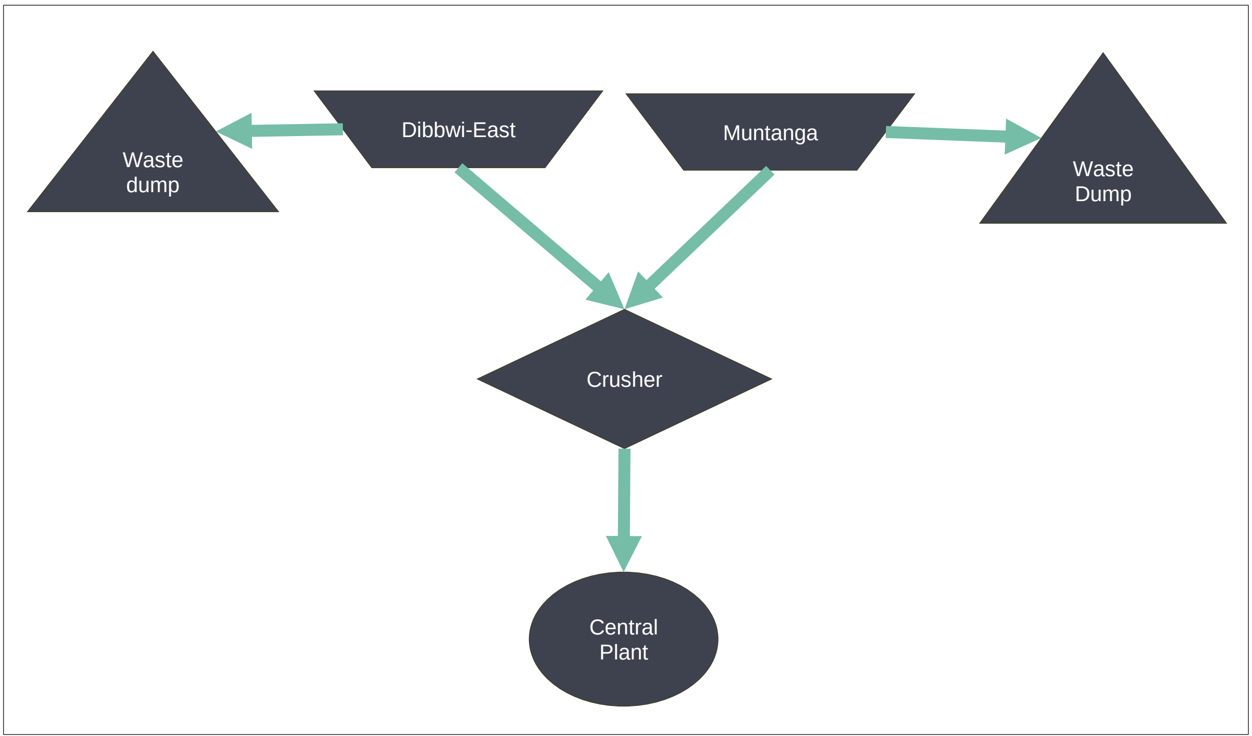

Figure 16-16: Material flow strategy for the Project

16.2.8.3.7. Production schedule drivers

Production scheduling drivers were identified prior to commencing the scheduling processes. Prior to the production schedule, several key constraints were identified which include:

- Maximum feed to central plant of 3.5 Mtpa
- Optimise equipment utilisation and direct ore feed to plant (minimise external stockpiles)
- Muntanga mined as hard as possible initially with Dibbwi East filling up plant feed
- Achieve a steady balance of production and product specification to the plant throughout the LOM
- Maintain practical mining face advancement
- Adhere to steady-state waste stripping requirements to facilitate the build-up strategy
- SMU blocks containing mineralised material from M&I Mineral Resources above the COG was considered for plant feed
- SMU blocks containing mineralised material from Inferred and Unclassified Mineral Resources were treated as waste material.

16.2.8.4. Schedule results

A total of 39.6 Mt ore and 144.2 Mt waste was scheduled over the 12-year LOM as shown in Table 16-18. The maximum scheduled total mining tonnes (waste and ore) was limited to 17.3 Mtpa, the installed capacity of eight primary load and haul teams. Production ramps up to 82 % in year one, runs at the full production rate of 17.3 Mtpa from years two to ten, and ramps down in years 10 and 11. The progressive S/R over the LOM is 3.64 [t:t].

Table 16-18: Annual mining production schedule

	Units	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
ROM Product														
Plant feed	Mt	**39.6**	1.6	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.0
ROM Grade	ppm	**320**	287	332	330	257	234	234	260	317	342	370	398	489
Stockpiles														
Central Stockpiles Added	Mt	**0.5**	0.5											
Central Stockpiles Added Grade	ppm		288											
Central Stockpiles Removed	Mt	**0.5**	0.4	0.1										
Central Stockpiles Removed Grade	ppm		298	248										
Central (Dibbwi East & Muntanga)														
Total Tonnes (Central)	Mt	**183.8**	14.1	17.3	17.3	17.2	17.1	17.1	17.3	16.8	16.8	16.3	10.6	5.6
Total Ore														
Ore Tonnes	MT	**39.6**	1.7	3.4	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.0
Ore Volume	Mbcm	**18.9**	0.8	1.6	1.7	1.7	1.7	1.7	1.7	1.7	1.7	1.7	1.7	1.4
Ore Grade	ppm	**320**	284	335	330	257	234	234	260	317	342	370	398	489
Total Waste														
Waste Tonnes	Mt	**144.2**	12.4	14.0	13.8	13.7	13.6	13.6	13.8	13.3	13.3	12.8	7.1	2.6
Waste Volume	Mbcm	**68.7**	5.9	6.6	6.6	6.5	6.5	6.5	6.6	6.4	6.4	6.1	3.4	1.3
Waste Grade	ppm	**15**	4	10	14	11	14	12	18	12	17	19	25	65
Dibbwi East														
Total Tonnes	Mt	**165.3**	9.0	10.9	12.7	15.3	16.8	17.1	17.3	16.8	16.8	16.3	10.6	5.6
Total Ore														
Ore Tonnes	Mt	**31.3**	0.0	0.5	1.2	2.3	3.2	3.5	3.5	3.5	3.5	3.5	3.5	3.0
Ore Volume	Mbcm	**14.9**	0.0	0.2	0.6	1.1	1.5	1.7	1.7	1.7	1.7	1.7	1.7	1.4
Ore Grade	ppm	**317**	154	280	257	230	232	234	260	317	342	370	398	489
Total Waste														
Waste Tonnes	Mt	**134.1**	9.0	10.4	11.4	13.0	13.6	13.6	13.8	13.3	13.3	12.8	7.1	2.6
Waste Volume	Mbcm	**63.8**	4.3	5.0	5.4	6.2	6.5	6.5	6.6	6.4	6.4	6.1	3.4	1.3

Waste Grade	ppm	**14**	1	4	11	9	14	12	18	12	17	19	25	65
PB1 Ore														
Ore Tonnes	Mt	**9.2**		0.5	1.2	1.6	1.7	1.5	1.4	1.2	0.2			
Ore Volume	Mbcm	**4.4**		0.2	0.6	0.8	0.8	0.7	0.7	0.6	0.1			
Ore Grade	ppm	**299**		287	263	254	261	267	297	460	466			
PB1 Waste														
Waste Tonnes	Mt	**32.6**	6.1	7.8	6.3	3.5	3.0	2.8	2.5	0.4	0.1			
Waste Volume	Mbcm	**15.5**	2.9	3.7	3.0	1.7	1.4	1.3	1.2	0.2	0.0			
Waste Grade	ppm	**15**	0	4	17	16	24	27	51	72	19			
PB2 Ore														
Ore Tonnes	Mt	**16.0**			0.0	0.7	1.4	1.6	1.4	1.3	2.4	3.1	2.9	1.1
Ore Volume	Mbcm	**7.6**			0.0	0.3	0.7	0.8	0.7	0.6	1.1	1.5	1.4	0.5
Ore Grade	ppm	**330**			138	179	203	219	240	234	363	389	424	516
PB2 Waste														
Waste Tonnes	Mt	**71.7**	2.4	2.4	4.6	8.5	8.2	7.6	8.9	8.8	9.5	8.0	2.3	0.5
Waste Volume	Mbcm	**34.1**	1.1	1.2	2.2	4.1	3.9	3.6	4.2	4.2	4.5	3.8	1.1	0.2
Waste Grade	ppm	**13**	-	4	4	8	11	9	11	11	19	23	43	70
PB3 Ore														
Ore Tonnes	Mt	**6.1**	0.0	0.0	0.0	0.0	0.1	0.4	0.7	0.9	0.9	0.4	0.6	1.9
Ore Volume	Mbcm	**2.9**	0.0	0.0	0.0	0.0	0.1	0.2	0.4	0.4	0.4	0.2	0.3	0.9
Ore Grade	ppm	**312**	154	164	149	138	174	168	227	249	259	218	284	474
PB3 Waste														
Waste Tonnes	Mt	**29.8**	0.5	0.2	0.6	0.9	2.4	3.3	2.4	4.1	3.8	4.8	4.8	2.1
Waste Volume	Mbcm	**14.2**	0.2	0.1	0.3	0.5	1.1	1.6	1.2	2.0	1.8	2.3	2.3	1.0
Waste Grade	ppm	**14**	9	7	1	1	10	8	11	8	10	12	17	63
Muntanga														
Total Tonnes	Mt	**18.5**	5.1	6.5	4.7	1.9	0.3							
Total Ore														

Ore Tonnes	Mt	**8.4**	1.7	2.9	2.3	1.2	0.3							
Ore Volume	Mbcm	**4.0**	0.8	1.4	1.1	0.6	0.1							
Ore Grade	ppm	**331**	285	344	369	309	265							
Total Waste														
Waste Tonnes	Mt	**10.1**	3.4	3.5	2.4	0.7	0.0							
Waste Volume	Mbcm	**4.8**	1.6	1.7	1.2	0.3	0.0							
Waste Grade	ppm	**23**	11	28	30	32	75							

The LOM annual material movements and progressive S/R are shown in Figure 16-17, with the tonnage contribution of each mining area in Figure 16-18. Muntanga and Dibbwi East PB1 have the lowest S/R as shown in Figure 16-19 and were targeted first to support early ore production based on a low stripping requirement. The primary motivation for the initial stripping at Dibbwi East PB2 was to optimise upfront waste stripping to secure a consistent ore feed later in the LOM.

Dibbwi East PB3 provides an additional mining face but was throttled back initially to maintain short haul access for pushback ore material via a temporary surface haul road crossing.

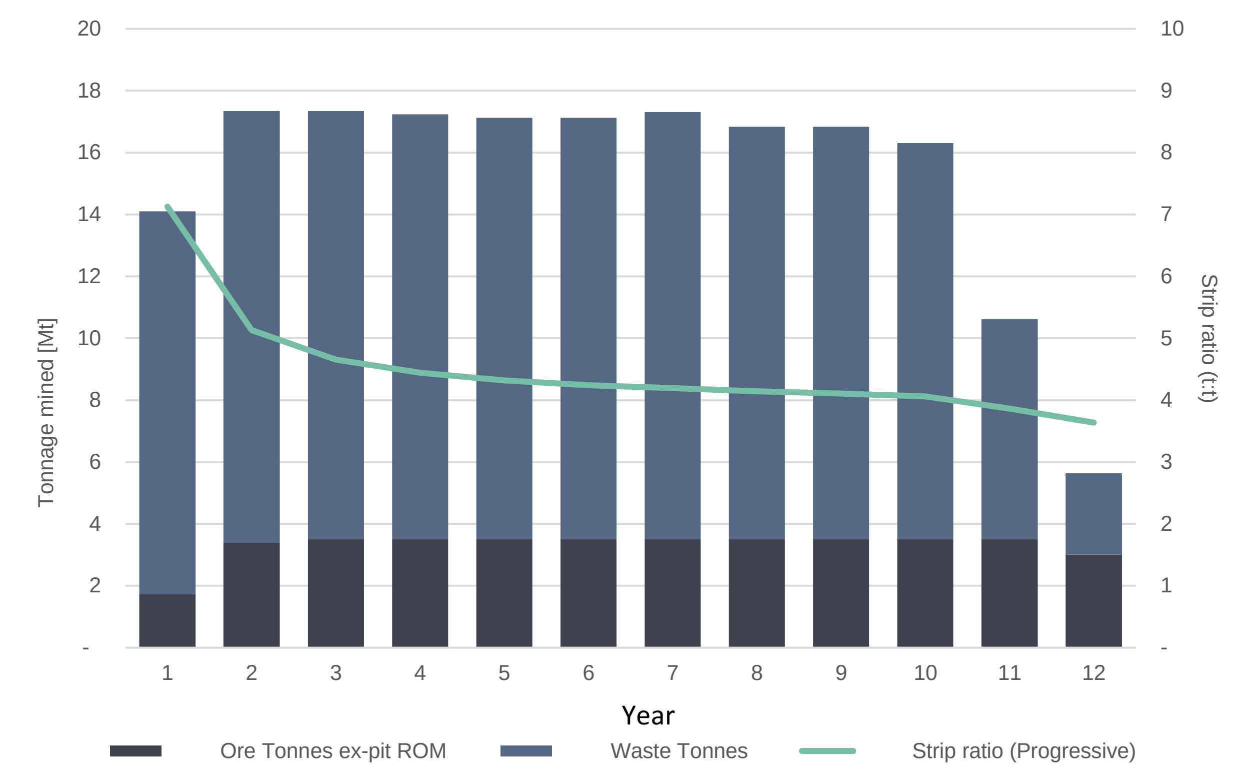

Figure 16-17: LOM schedule annual material movements and progressive strip ratio

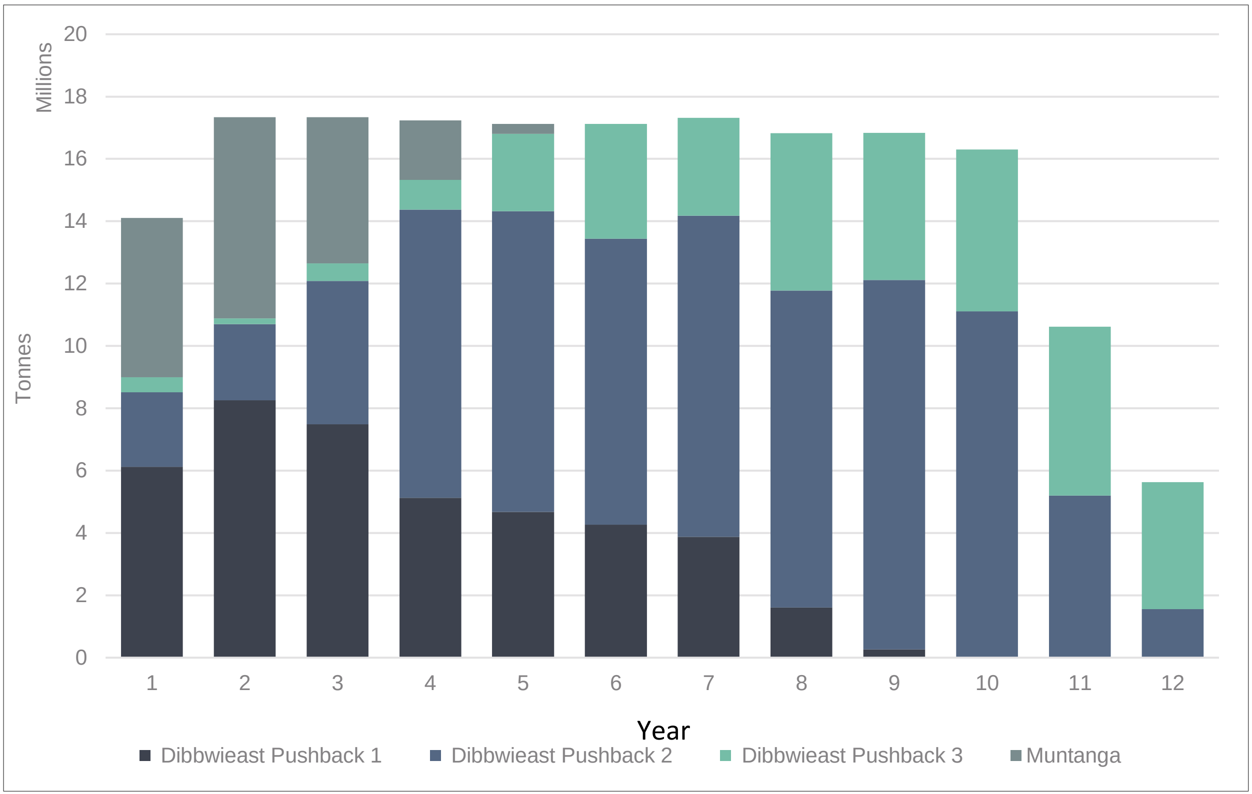

Figure 16-18: LOM schedule annual material movement per mining area

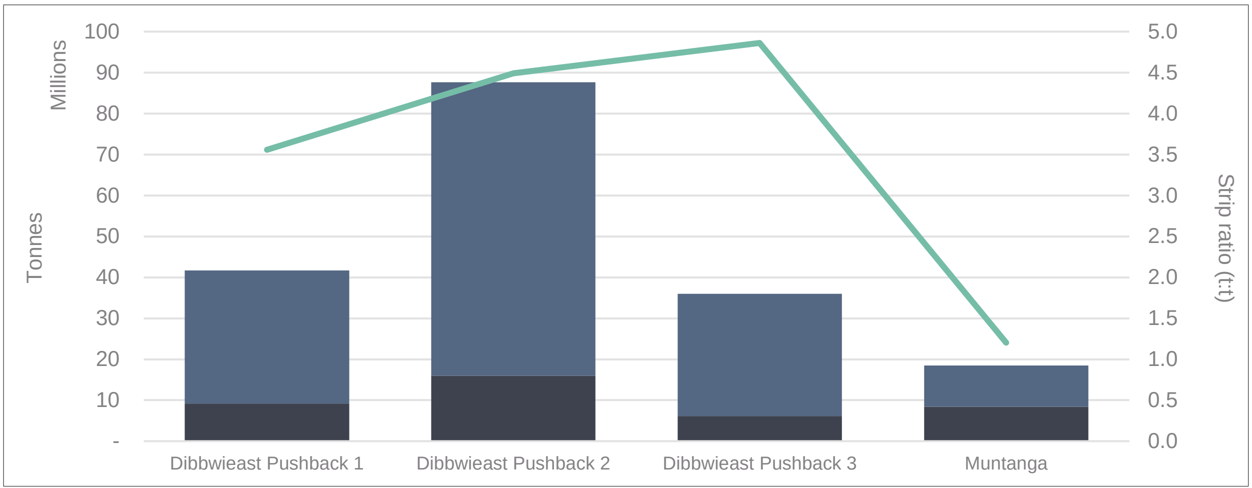

Figure 16-19: LOM material movement per mining area

Total ROM ore mined from the pits is 3.50 Mtpa at steady state with Muntanga at 2.92 Mtpa (peak) and Dibbwi East at 3.50 Mtpa (peak). The total ROM and average U_3O_8 grade over the LOM are shown in Figure 16-20.

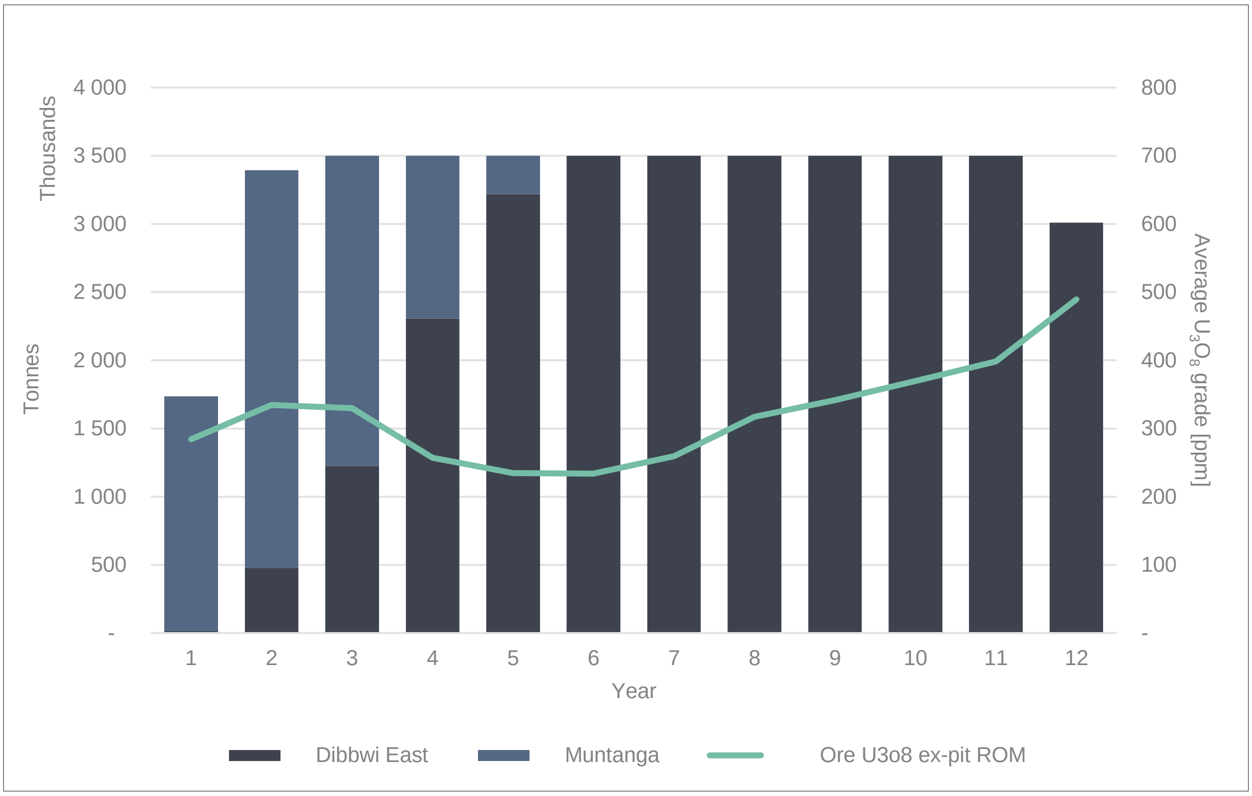

Figure 16-20: ROM tonnage and average U_3O_8 grade over LOM

Plant feed comprises ROM feed from the Muntanga and Dibbwi East pits. Central ROM feed to plant is at 50 % in year one and full capacity of 3.5 Mtpa from year two.

The average U_3O_8 grades for Muntanga and Dibbwi East pushbacks one, two and three are 331 ppm, 299 ppm, 330 ppm and 312 ppm respectively.

Plant feed grade varies between 234 ppm and 489 ppm over the LOM with an overall average of 320 ppm. Central plant feed tonnage and grade is shown in Figure 16-21.

Figure 16-21: Central plant feed tonnage and average U_3O_8 grade

16.2.8.4.1. Destination schedule

Material swell factors of 30 % were applied for the destination schedule where waste material was scheduled to be placed on waste dumps or backfilled and ore fed to the central plant.

All waste scheduled at the start of each pit was scheduled for placement on the designated waste dumps until sufficient in-pit space was available to commence with backfill in the pits by virtue of the staged sequence of the mining phases.

Figure 16-22 to Figure 16-29 show the bi-yearly source and destination development plans for Muntanga and Dibbwi East.

Included in these plans are the following:

- Mining blocks of 40 x 40 m coloured by bench elevation
- Loading equipment
- Pit and waste dump deployment and access
- Haul routes from loading faces to final destinations.

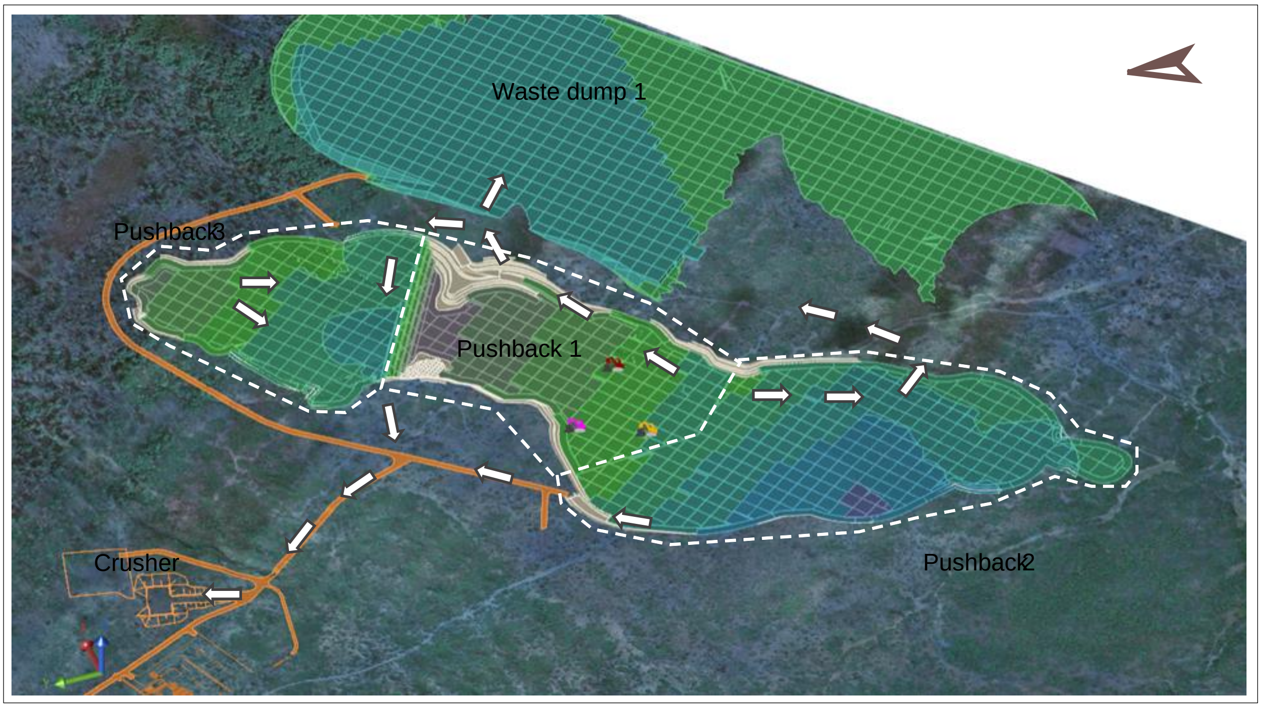

Figure 16-22: Dibbwi East – stage plan, end of year 2

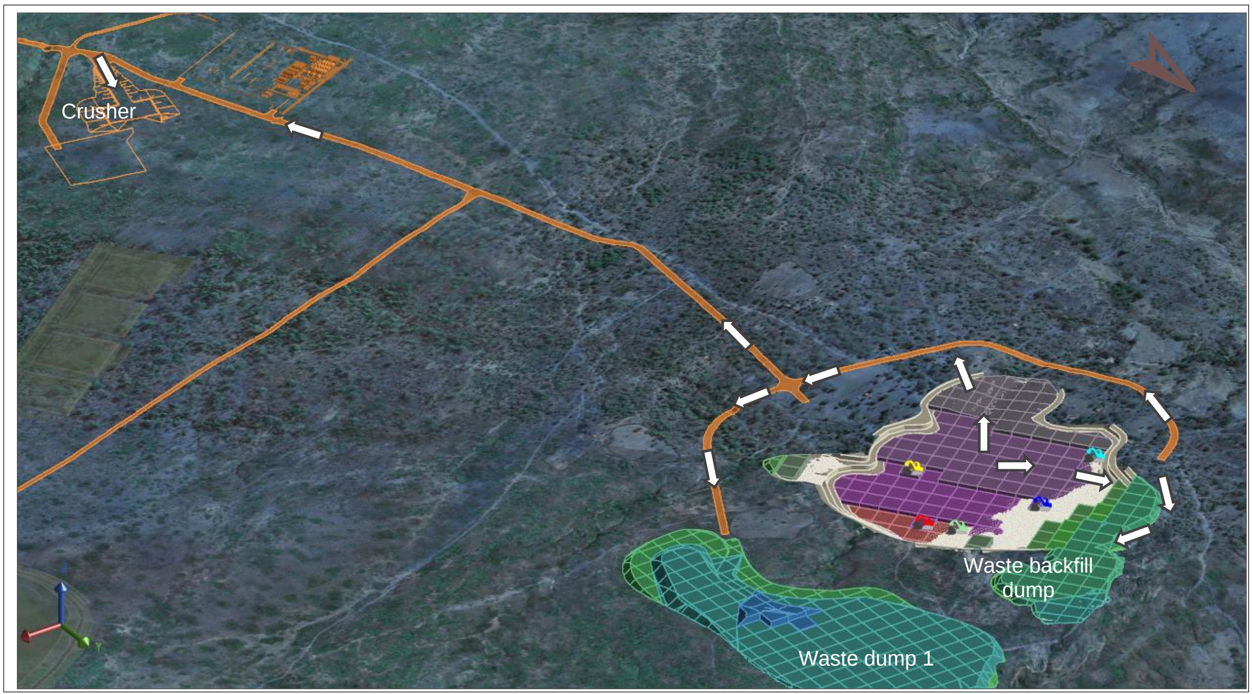

Figure 16-23: Muntanga stage plan, end year 2

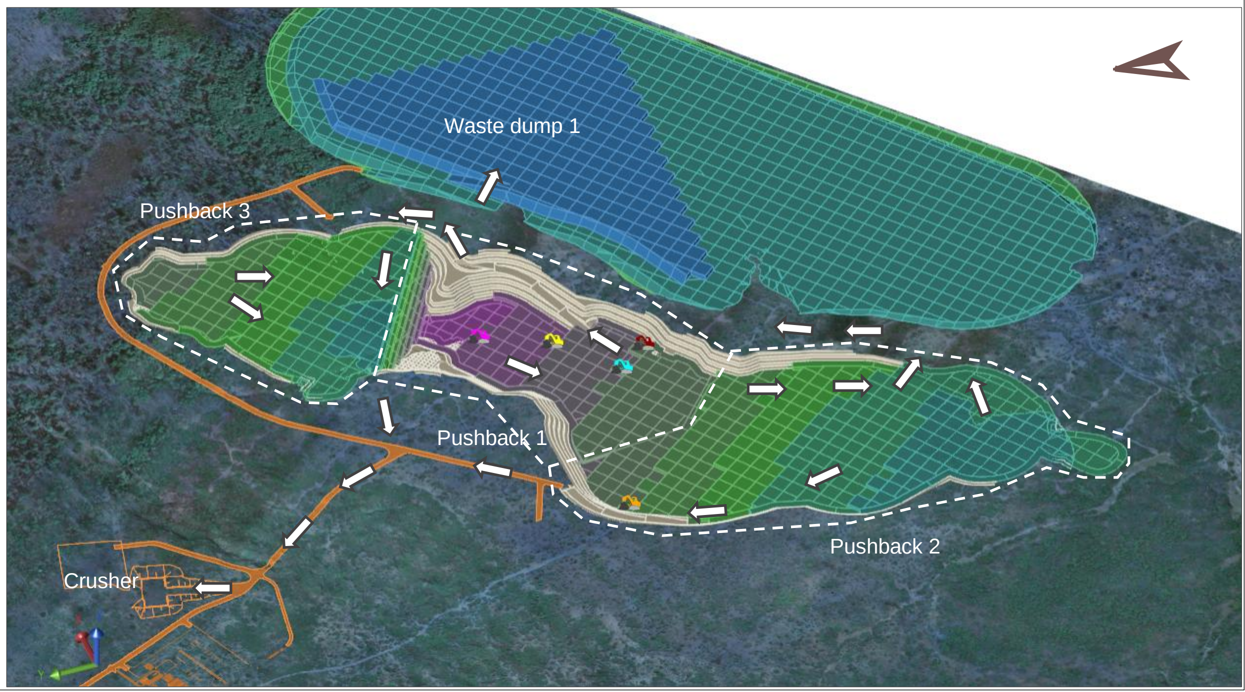

Figure 16-24: Dibbwi East stage plan, end year 4

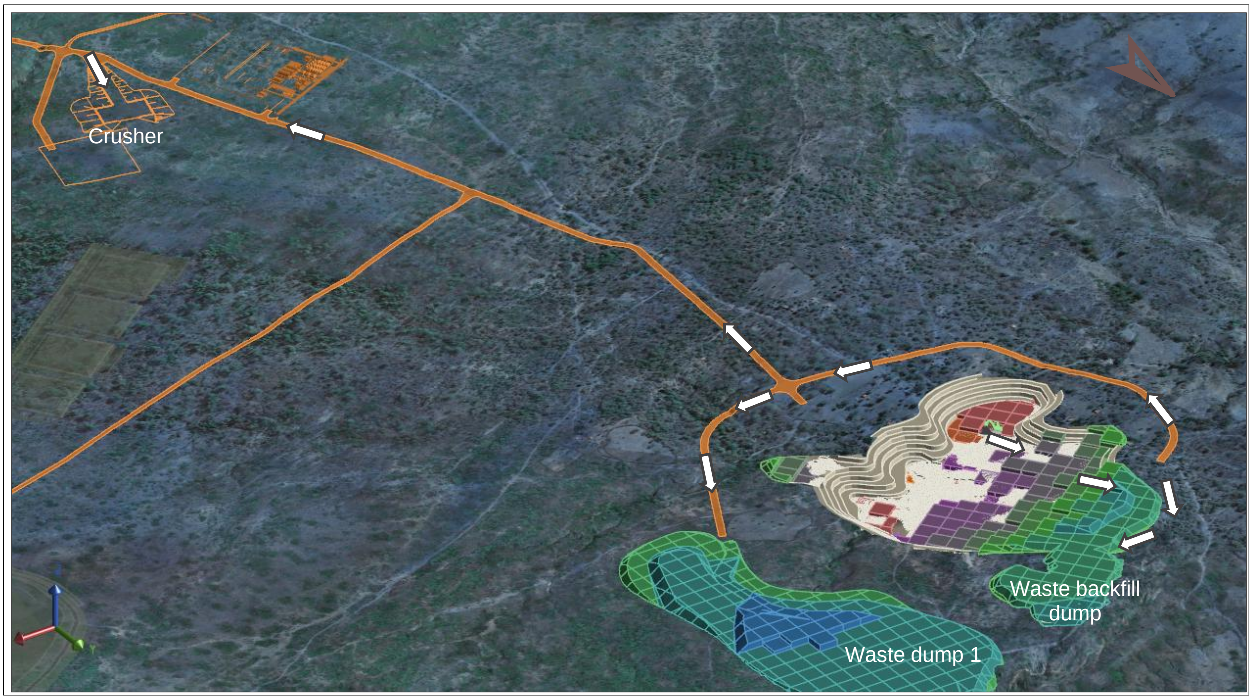

Figure 16-25: Muntanga stage plan, end year 4

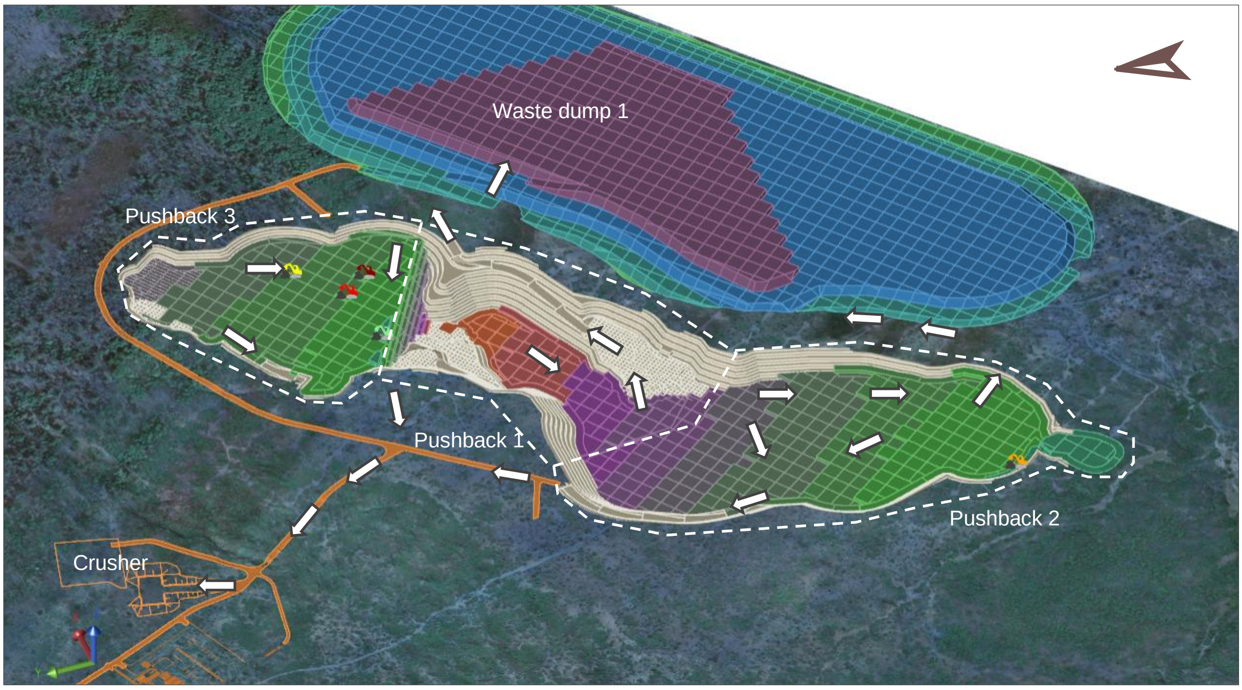

Figure 16-26: Dibbwi East stage plan, end year 6

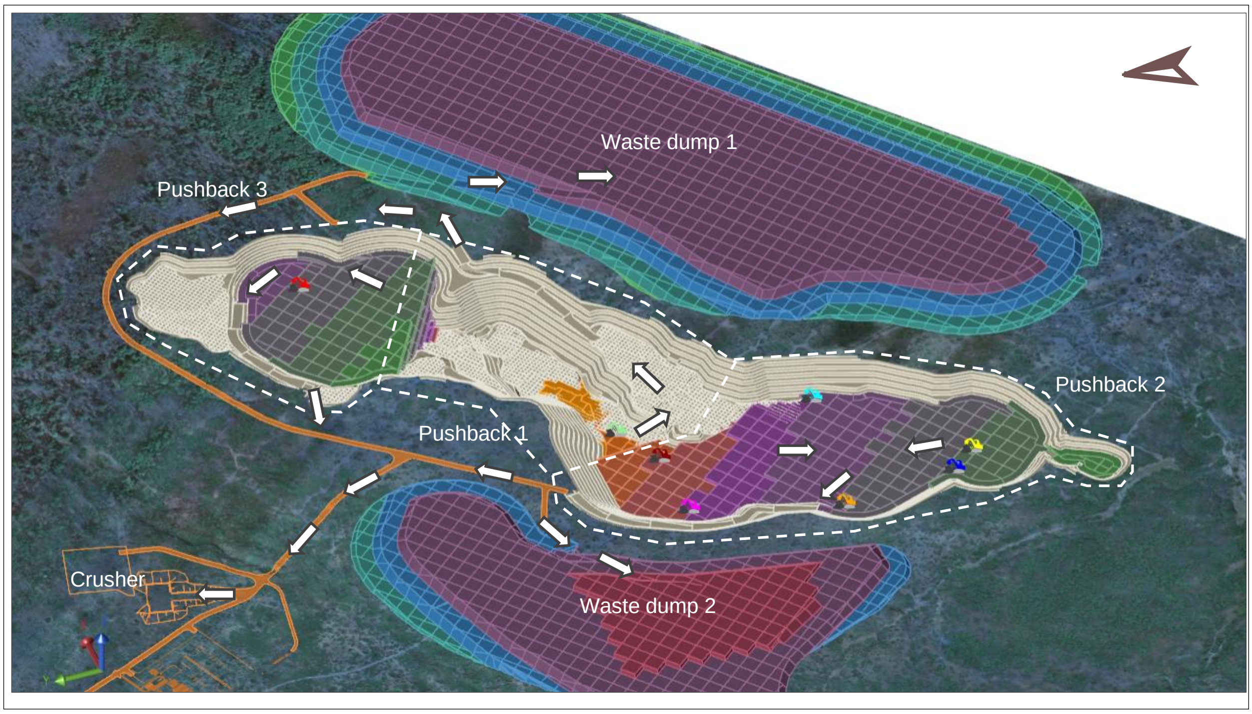

Figure 16-27: Dibbwi East stage plan, end year 8

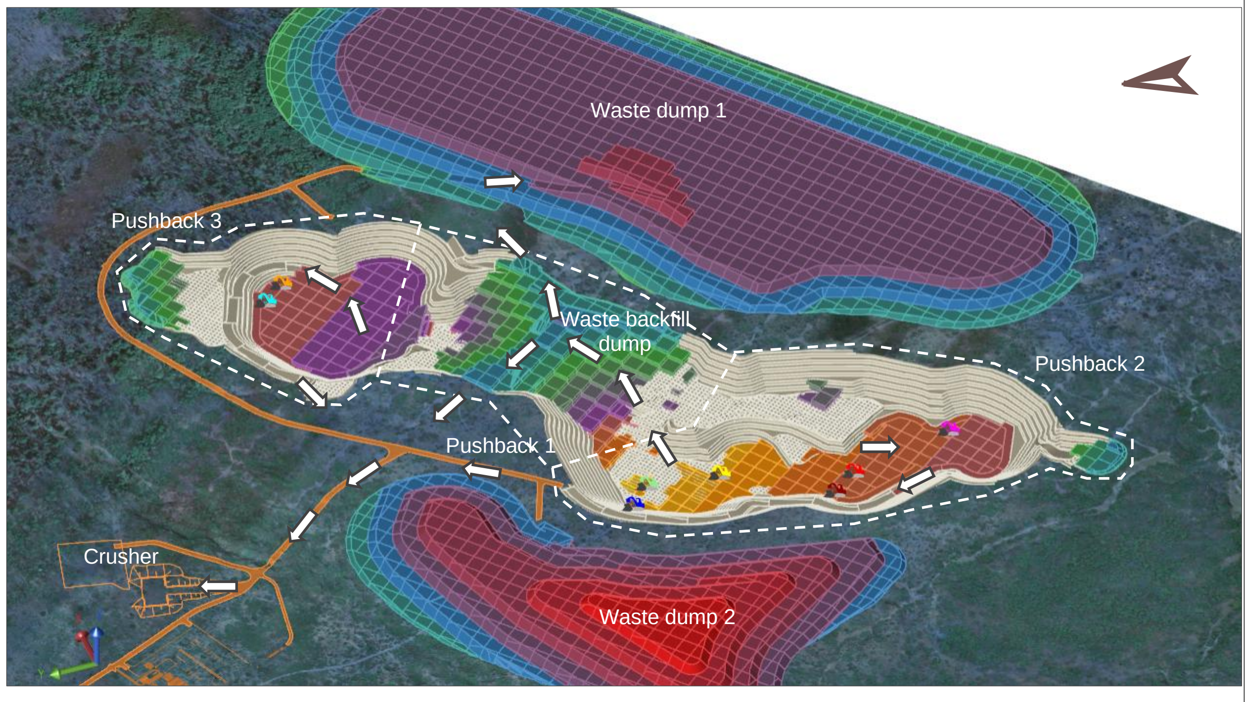

Figure 16-28: Dibbwi East stage plan, end year 10

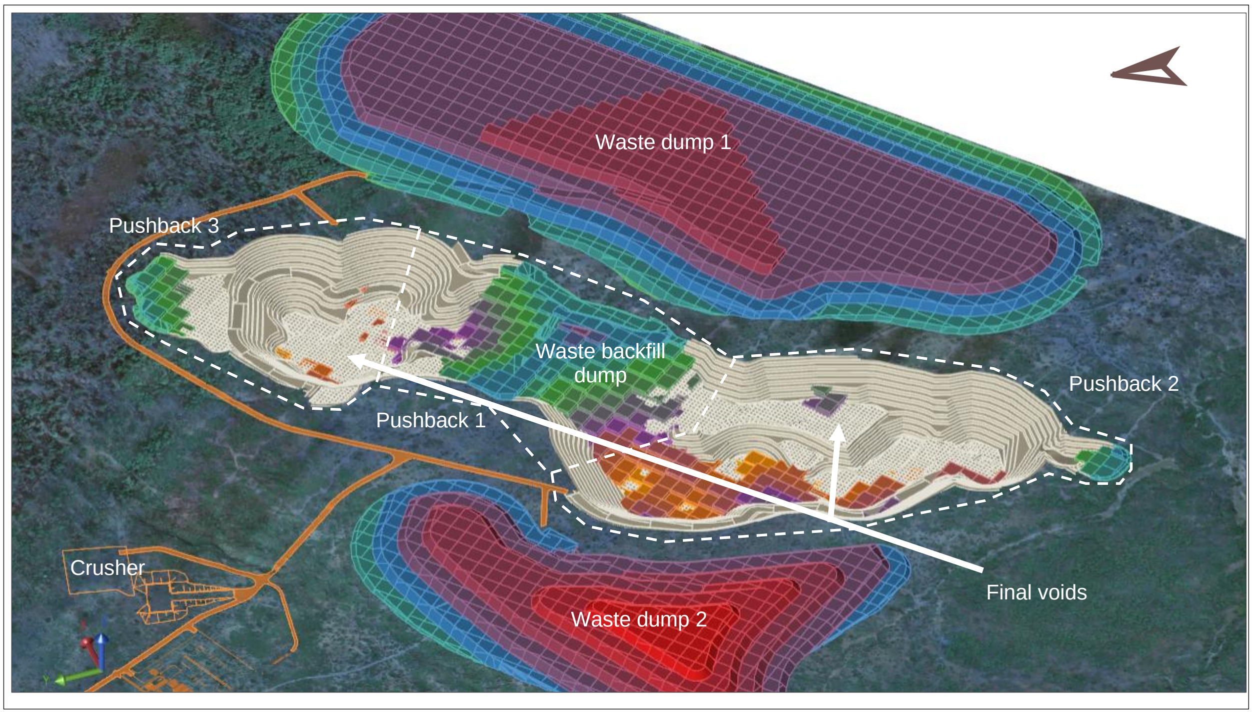

Figure 16-29: : Dibbwi East stage plan, end year 12

The source and destination schedule with associated hauling distances is shown in Table 16-19.

Table 16-19: Source and destination schedule

Tonnes hauled		Unit	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Ore														
Source:	Dibbwi East													
Destination:	Central ROM	Mt	0.01											
Destination:	Central Crusher	Mt		0.48	1.22	2.31	3.22	3.50	3.50	3.50	3.50	3.50	3.50	3.01
Source:	**Muntanga**													
Destination:	Central ROM	Mt	0.50											
Destination:	Central Crusher	Mt	1.22	2.92	2.28	1.19	0.28							
Waste														
Source:	**Dibbwi East**													
Destination:	Backfill	Mt									10.16	5.98	3.05	2.62
Destination:	Dump 1	Mt	8.98	10.41	11.42	13.02	13.59	13.62	9.72	0.38	0.07	0.74	4.07	
Destination:	Dump 2	Mt							4.09	12.95	3.10	6.09		
Source:	**Muntanga**													
Destination:	Backfill	Mt	1.22	1.41	1.93	0.72	0.03							
Destination:	Dump 1	Mt	2.17	2.13	0.49									
One-way hauling distances		**Unit**	**Y1**	**Y2**	**Y3**	**Y4**	**Y5**	**Y6**	**Y7**	**Y8**	**Y9**	**Y10**	**Y11**	**Y12**
Ore														
Source:	**Dibbwi East**													
Destination:	Central ROM	km	1.9											
Destination:	Central Crusher	km		1.8	1.8	2.0	2.3	2.4	2.4	2.3	2.4	2.8	2.5	2.4
Source:	**Muntanga**													
Destination:	Central ROM	km	3.5											
Destination:	Central Crusher	km	3.2	3.6	3.5	3.5	3.0							
Waste														
Source:	**Dibbwi East**													
Destination:	Backfill	km									1.7	1.9	1.7	2.4
Destination:	Dump 1	km	2.9	2.4	2.5	2.0	1.9	1.9	2.4	2.1	3.3	3.2	3.5	
Destination:	Dump 2	km							1.7	2.0	1.7	2.0		
Source:	**Muntanga**													
Destination:	Backfill	km	1.7	1.5	1.0	1.1	0.6							
Destination:	Dump 1	km	1.9	1.8	1.9									

Figure 16-30 shows the cumulative total waste per destination. A cumulative total of 68.7 Mbcm waste was scheduled, of which 55.7 Mbcm (81.2 %) was placed on waste dumps and 12.9 Mbcm (18.2 %) backfilled in the pits as shown Table 16-20.

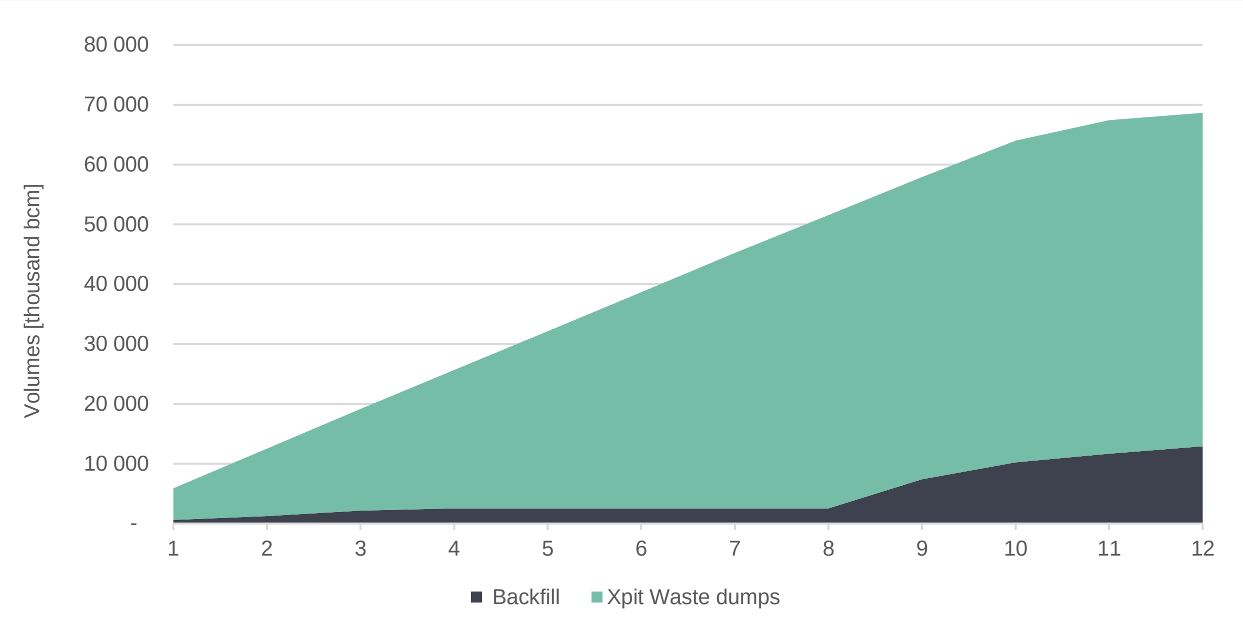

Figure 16-30: Cumulative total waste per destination

Table 16-20: Waste destination per area

Area	Total waste volume [bcm x1000]	Backfill [%]	Waste dump [%]
Muntanga	4 807	52.6	47.4
Dibbwi East	63 838	16.3	83.8
Total	**68 645**	**18.8**	**81.2**

16.2.9.Primary mining equipment

16.2.9.1. Drilling equipment

The annual drilling meters and number of holes reflected in Table 16-21 is a function of the drill and blast designs discussed under Section 16.2.3.1., with an allowance of 11 % for redrilling of holes as and when required.

Table 16-21: Annual drill meters and number of holes

Description	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Central ore drill meters	43 806	80 459	89 752	91 398	95 900	86 363	85 902	83 839	92 706	109 668	73 132	41 760
Central waste drill meters	116 815	116 769	104 970	100 631	93 401	96 090	98 827	91 078	89 752	65 539	32 484	3 343
Central trim drill meters	11 041	13 943	17 094	18 928	20 594	29 174	29 236	34 255	24 868	25 974	27 580	28 331
Central presplit drill meters	16 266	18 951	20 650	25 078	25 797	40 383	48 637	45 721	39 450	31 546	38 474	55 806
Central total drill meters	**187 928**	**230 122**	**232 466**	**236 035**	**235 692**	**252 010**	**262 602**	**254 892**	**246 776**	**232 728**	**171 670**	**129 240**
Central ore drill holes	4 381	8 046	8 975	9 140	9 590	8 636	8 590	8 384	9 271	10 967	7 313	4 176
Central waste drill holes	11 681	11 677	10 497	10 063	9 340	9 609	9 883	9 108	8 975	6 554	3 248	334
Central trim drill holes	1 104	1 394	1 709	1 893	2 059	2 917	2 924	3 426	2 487	2 597	2 758	2 833
Central presplit drill holes	1 620	1 890	2 058	2 502	2 574	4 031	4 859	4 566	3 942	3 147	3 841	5 573
Central total drill holes	**18 786**	**23 007**	**23 240**	**23 598**	**23 564**	**25 194**	**26 256**	**25 483**	**24 675**	**23 265**	**17 161**	**12 916**

The total number of drill rigs required per year is indicated in Figure 16-31. The number of drill rigs is a function the unit hours from estimated drill penetration rates and available monthly hours.

Average penetration rates which include tramming and ore grade control is as follows:

- Waste blocks: 34 m per hour
- Ore blocks: 25 m per hour.

An additional drill was included and allocated for geological infill drilling with an allowance of 3 600 m per annum.

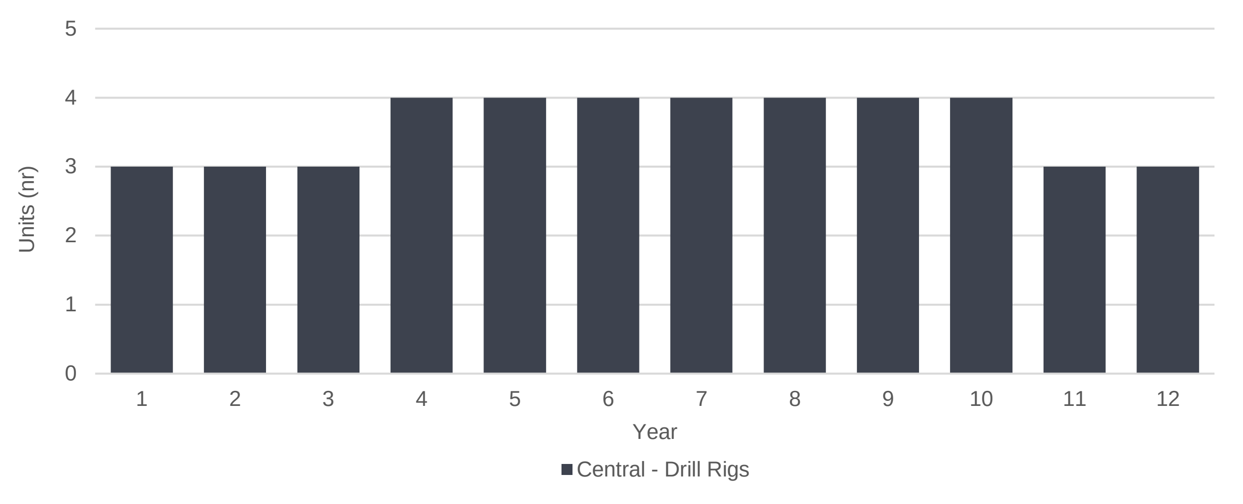

Figure 16-31: Sandvik Leopard DI650i drill rig requirements per year

Equipment is maintained according to best practice. The equipment repair and maintenance ("R&M") Opex includes OEM supplied life cycle costs ("LCC") (parts), wear checks, oil and grease, undercarriage repairs and ground engaging tools ("GET")). An allowance of 2 % of the above costs was made to cover against equipment damage caused by operator abuse. Since the drill rigs don't reach the end of their of 27 000 hour useful life during the project, no replacements are required as shown in Table 16-22.

Table 16-22: Drill rig utilisation and replacements

Unit number	Total operational hours at end of LOM
Drill DI650 DR01	23 767
Drill DI650 DR02	27 724
Drill DI650 DR03	27 611
Drill DI650 DR04	18 571

In summary a total of four drill rigs is purchased during over die duration of the mine as shown in Table 16-23.

Table 16-23: Drill rigs purchased

Equipment	Cat	Initial units purchased [month]													Total	
		M1	M2	M3	M4	M5	M6	M7	M8	M9	M10	M11	M12	M13	Initial units	Sustaining units
Drill rigs	Drilling	2		1											3	1

16.2.9.2. Load and haul equipment

The total number of excavators required was estimated from the scheduling model based on the productivity of the excavator (Table 16-24) and effective working hours, with an additional cost allowance for bench preparation and tramming between mining faces. The excavator ramp-up is as follows:

- Month 1 to 2: Three units

- Month 3: Five units
- Month 4 to 5: Six units
- Month 6: Seven units
- Month 7: At full production with eight units.

As scheduled production reduces, the excavator requirements reduce to five and two towards the end of the project as shown in Figure 16-32.

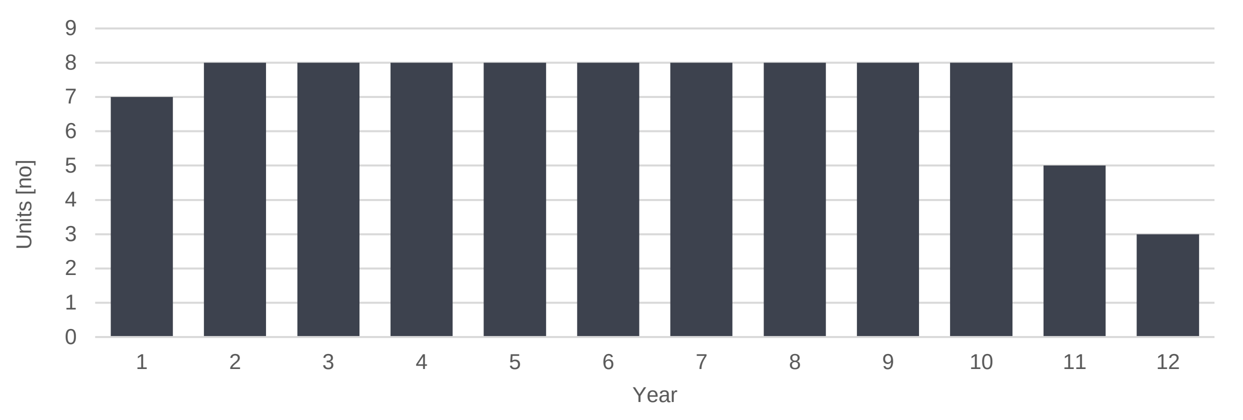

Figure 16-32: Cat 395 excavator requirements per year

Allowance was made to maintain equipment according to best practice. The equipment R&M Opex includes OEM supplied LCCs (parts), wear checks, oil and grease, bucket repairs, undercarriage repairs and GET. An allowance of 2 % of the above costs was made for equipment damage caused by operator abuse.

During the project, excavators will be replaced once their useful life of 30 000 hours is reached, as shown in Table 16-24.

Table 16-24: Excavator utilisation and replacements

Unit number	Hours utilised initial unit	Hours utilised replacement unit
Cat 395 EX01	30 106	15 292
Cat 395 EX02	30 106	16 302
Cat 395 EX03	30 106	17 360
Cat 395 EX04	30 182	17 246
Cat 395 EX05	30 182	20 104
Cat 395 EX06	30 245	21 734
Cat 395 EX07	30 241	21 344
Cat 395 EX08	30 246	20 859

Based on the material tonnes, destination and associated hauling distances estimated from the destination schedule the hauling equipment was simulated. The hauling simulation was completed in RPMGlobal HaulSim ("Haulsim"). The haul road network and stage plan from the scheduling model were applied in the simulation software and hauling distances were simulated to estimate the number of haul trucks required.

The maximum number of trucks required was estimated at 49 at peak production. Haul trucks will ramp up as follows:

- Month 1 to 2: 19 units
- Month 3: 32 units
- Month 4 to 5: 38 units
- Month 6: 43 units
- Month 7: At full production with 49 units.

As scheduled production reduces and backfilling increases, the truck requirements reduce to 45, 32 and 19 towards the end of LOM as shown in Figure 16-33.

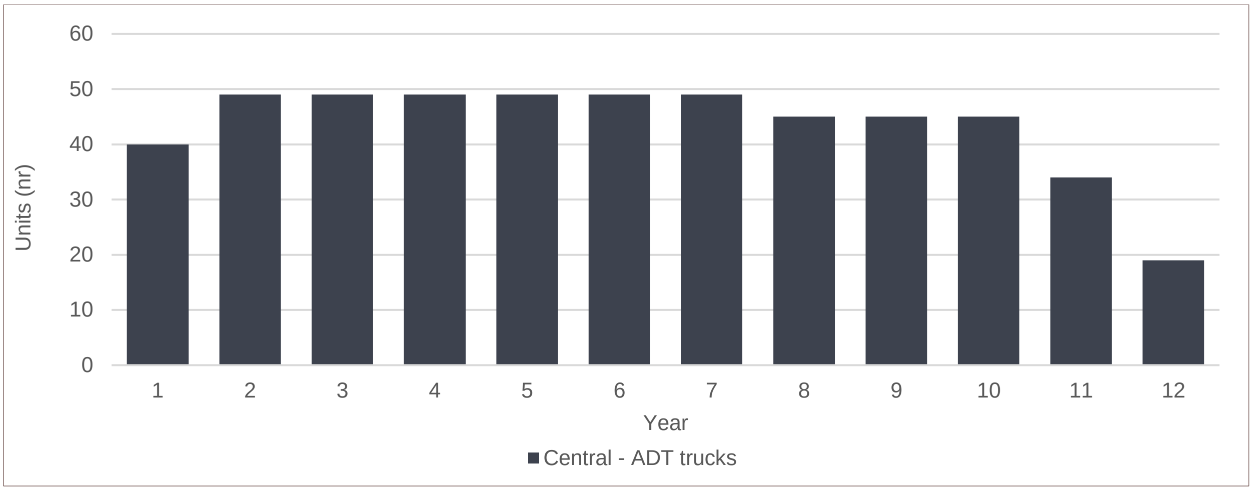

Figure 16-33: Cat 745 ADT requirements per year

Equipment is maintained according to best practice. The equipment R&M Opex includes OEM supplied LCCs (parts), wear checks, oil and grease, bucket repairs, tyre replacements at 5 000 hours and GET. An allowance of 2 % of the above costs was made to cover against equipment damage caused by operator abuse.

During the project, haul trucks are replaced once their useful life of 25 000 hours is reached, as shown in Table 16-25.

Table 16-25: Mining ADT utilisation and replacements

Unit number	Hours utilised initial unit	Hours utilised replacement unit
M ADT01	25 180	13 605
M ADT02	25 180	13 605
M ADT03	25 180	13 605
M ADT04	25 180	13 605
M ADT05	25 180	13 605
M ADT06	25 180	13 605
M ADT07	25 180	13 605
M ADT08	25 180	13 605
M ADT09	25 180	14 578
M ADT10	25 180	14 578
M ADT11	25 180	14 578
M ADT12	25 180	14 578
M ADT13	25 180	14 578
M ADT14	25 180	15 215
M ADT15	25 180	15 215
M ADT16	25 180	15 215
M ADT17	25 180	15 215
M ADT18	25 180	15 215
M ADT19	25 180	15 215
M ADT20	25 187	14 525
M ADT21	25 187	15 823
M ADT22	25 187	17 094
M ADT23	25 187	17 094
M ADT24	25 187	17 094
M ADT25	25 187	17 094
M ADT26	25 187	17 094

Unit number	Hours utilised initial unit	Hours utilised replacement unit
M ADT27	25 187	17 094
M ADT28	25 187	18 896
M ADT29	25 187	18 896
M ADT30	25 187	18 896
M ADT31	25 187	18 896
M ADT32	25 187	18 896
M ADT33	25 214	18 569
M ADT34	25 214	18 798
M ADT35	25 214	18 798
M ADT36	25 214	18 798
M ADT37	25 214	18 798
M ADT38	25 214	18 798
M ADT39	25 140	18 165
M ADT40	25 140	0
M ADT41	25 140	0
M ADT42	25 140	0
M ADT43	25 140	0
M ADT44	25 105	17 915
M ADT45	25 105	17 915
M ADT46	25 105	17 915
M ADT47	25 105	17 915
M ADT48	25 105	17 915
M ADT49	25 105	17 915

An additional allowance of 50 hours per month was added to the total monthly excavator and truck hours as a provision for concurrent clearing and grubbing.

In summary a total of 16 excavators and 94 dump trucks are purchased over the duration of the mine as shown in Table 16-26.

Table 16-26: Primary load and haul equipment purchased

Equipment	Cat	Initial units purchased [month]													Total	
		M1	M2	M3	M4	M5	M6	M7	M8	M9	M10	M11	M12	M13	Initial units	Sustaining units
Excavators	Loading	3		2	1		1	1							8	8
ADTs	Hauling	19		13	6		5	6							49	45

16.2.9.3. Support equipment

The number of dozers required is contingent upon the number of loading faces and the levelling requirements of active waste dumps. An estimated levelling capacity of 372 bcm per hour was applied with the assumption that 95 % of waste material tipped must be pushed over the dump crest. An additional allowance of 25 % of the total excavator hours was applied to calculate inpit dozer utilisation.

The total number of dozers required per annum is shown in Figure 16-34.

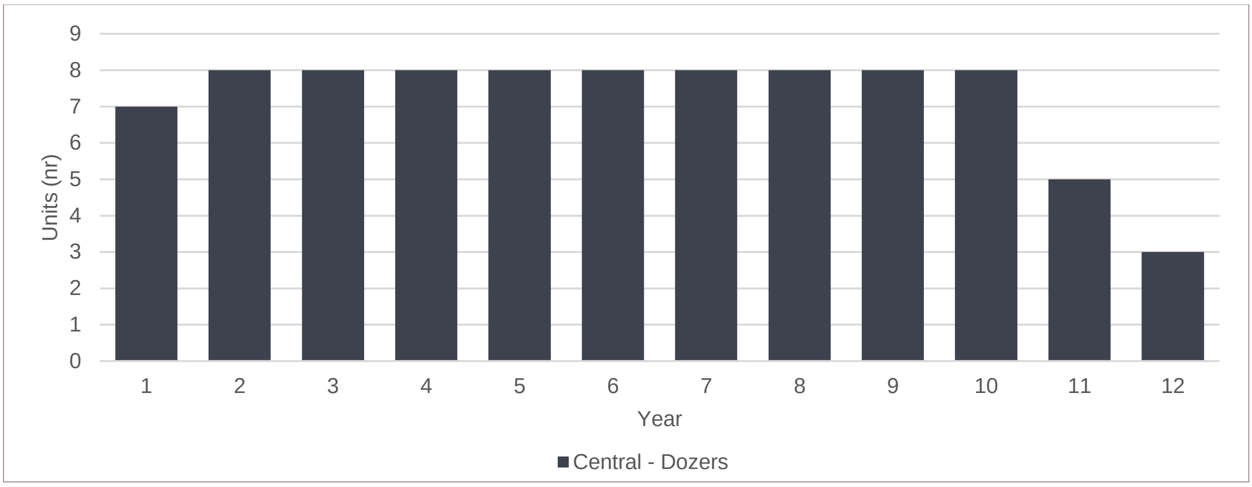

Figure 16-34: Cat D8 Dozer requirements per year

Since the dozers do not reach the end of their useful life of 35 000 hours during the project, no replacements are required as shown in Table 16-27.

Table 16-27: Dozer utilisation and replacements

Unit number	Hours utilised initial unit
Cat D8 DZ01	27 004
Cat D8 DZ02	27 589
Cat D8 DZ03	28 167
Cat D8 DZ04	28 559
Cat D8 DZ05	29 490
Cat D8 DZ06	30 374
Cat D8 DZ07	30 262
Cat D8 DZ08	29 888

The number of water bowsers and graders required depends on the environmental conditions, construction standards, materials, road length, road width and traffic. An average speed of 15 km/hr was assumed with a maintenance pass occurring after 13.5 haul cycles. The annual requirements are shown in Figure 16-35 and Figure 16-36. Once Muntanga is mined out the long ore haul road from Muntanga to the central plant becomes redundant and no further maintenance is required for this road, reducing the number of units from three to two.

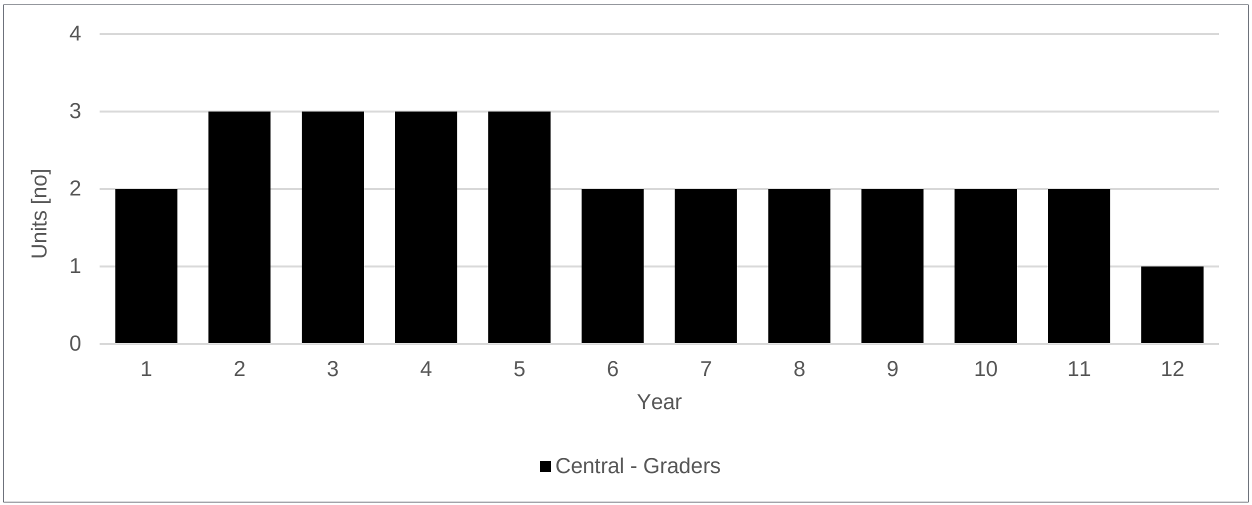

Figure 16-35: Cat 140 grader requirements per year

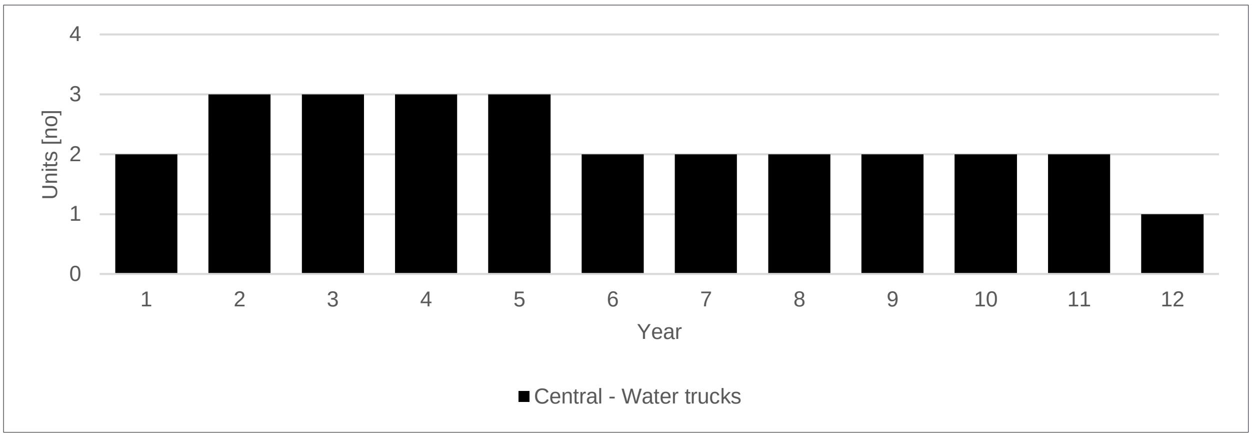

Figure 16-36: Cat 730 water bowser requirements per year

The useful life of graders and water bowsers is 30 000 and 25 000 hours respectively. No grader replacements are required as shown in Table 16-28. Cat GR 01 is no longer required once Muntanga is completed but kept on as a standby unit to cover the risk of a major breakdown of the remaining units causing disruptions to the operation. The useful life of Cat GR 02 is stretched by 5 000 hours to prevent needless Capex during the final two years of the LOM.

Table 16-28: Grader utilisation and replacements

Unit number	Hours utilised initial unit
Cat GR 01	14 291
Cat GR 02	34 970
Cat GR 03	27 873

As shown in Table 16-29, a similar strategy is followed for the water bowsers but with provision of the replacement of one unit and stretching the life of WB03.

Table 16-29: Water bowser utilisation and replacements

Unit number	Hours utilised initial unit	Hours utilised replacement unit
Cat WB01	14 291	0
Cat WB02	25 267	6 200
Cat WB03	31 377	0

An additional allowance of 50 hours per month was added to the total monthly support hours as a provision for concurrent clearing and grubbing.

In summary a total of eight dozers, three graders and four water bowsers are purchased over the duration of the mine as shown in Table 16-30.

Table 16-30: Support equipment purchased

Equipment	Cat	Initial units purchased [month]													Total	
		M1	M2	M3	M4	M5	M6	M7	M8	M9	M10	M11	M12	M13	Initial units	Sustaining units
Dozers	Support	4		2	1		1								8	
Graders	Support	2												1	3	
Water bowsers	Support	2												1	3	1

16.2.9.4. Rehandle equipment

The number of FELs required was calculated from the monthly rehandle tonnage requirements and equipment capacities. The monthly tonnage requirements include ore stockpiled during the start of the mine up until the plant is fully operational, and an operational stockpile required when the daily ore production exceeds the plant's daily capacity.

The rehandle equipment requirements and total utilisation are shown in Table 16-31, with one Komatsu WA600 unit and two Cat 745 ADTs required for the duration of the project.

Table 16-31: Rehandle equipment utilisation and replacements

Unit number	Hours utilised initial unit
R FEL 01	24 946
R ADT 01	22 564
R ADT 02	22 564

In summary a total of one FELs and two ADTs are purchased over the duration of the mine as shown in Table 16-32.

Table 16-32: Rehandle equipment purchased

Equipment	Cat	Initial units purchased [month]													Total	
		M1	M2	M3	M4	M5	M6	M7	M8	M9	M10	M11	M12	M13	Initial units	Sustaining units
FELs	Rehandle				1										1	
ADTs	Rehandle				2										2	

16.2.9.5. Minor equipment

Minor equipment allocated includes secondary rock breakers, diesel bowsers, water pumps, mobile lighting plant, tractor, loader, backhoes ("TLBs"), site use buses and LDVs.

16.2.9.5.1. Secondary rock breakers

Two units are placed for the full duration of the life of mine as shown in Figure 16-37, with the usage calculated as 10 % of ore material requiring further breaking at a production rate of 30m³ per hour and an additional allowance of 50 hours for inpit breaking when required.

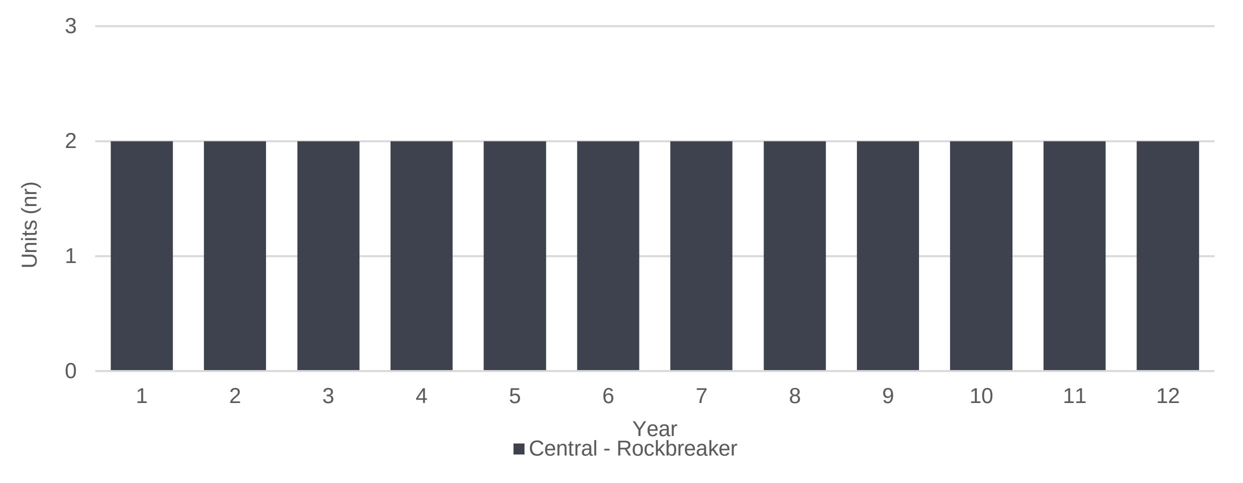

Figure 16-37: Cat 330GC rock breaker requirements per year

During the project, rock breakers are replaced once their useful life of 25 000 hours is reached, as shown in Table 16-33.

Table 16-33: Rock breaker utilisation and replacements

Unit number	Hours utilised initial unit	Hours utilised replacement unit
C330 RB01	25 069	10 160
C330 RB02	25 055	9 836

16.2.9.5.2. Diesel bowsers

Two articulated diesel bowers were included as shown in Figure 16-39, which calculates to one per pit for the purpose of refuelling track and non-mobile equipment such as excavators, dozers, drill rigs, lighting plant and water pumps. A fixed monthly allowance of 220 hours per unit was applied. Once the Muntanga pit is mined out, the second unit is kept on site and acts as a standby unit in the event of a major breakdown of the placed unit.

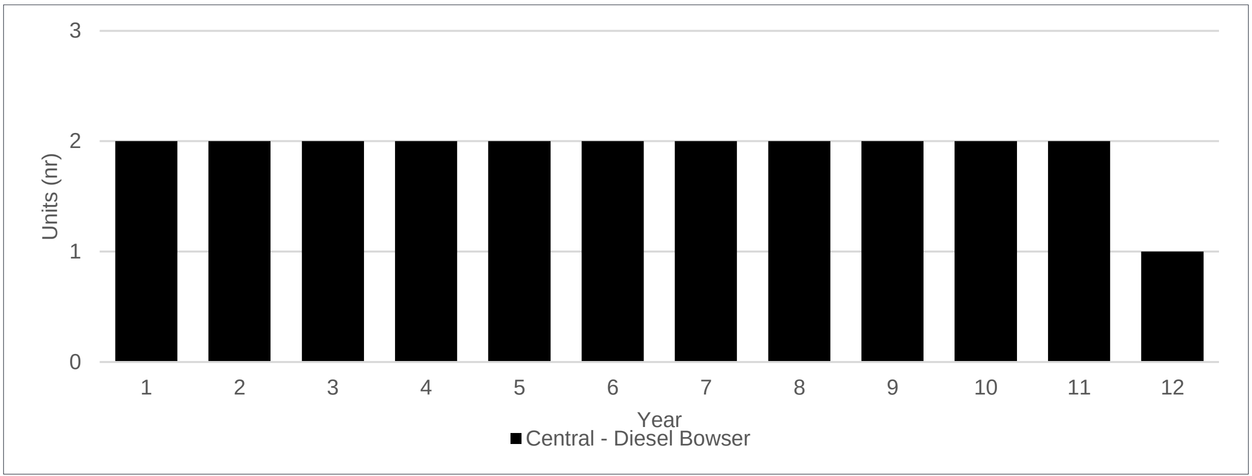

Figure 16-38: Cat 730 diesel bowser requirements per year

Since the diesel bowsers do not reach the end of their useful life of 25 000 hours during the project, no replacements are required as shown in Table 16-34.

Table 16-34: Diesel bowser utilisation and replacements

Unit number	Hours utilised initial unit
Cat 730 DB01	23 540
Cat 730 DB02	20 020

16.2.9.5.3. Water pumps

The pumping requirements were calculated from the number of active mining areas and expected inflows. Refer to Section 18.2 (Mining infrastructure) .

16.2.9.5.4. Mobile lighting plant

Mobile lighting is required during nighttime at the loading faces, inpit water pumps and active waste dump areas. Mobile lighting plants were allocated as one unit per excavator, one unit per water pump and one unit per active waste dump.

The total annual requirements are shown Figure 16-39.

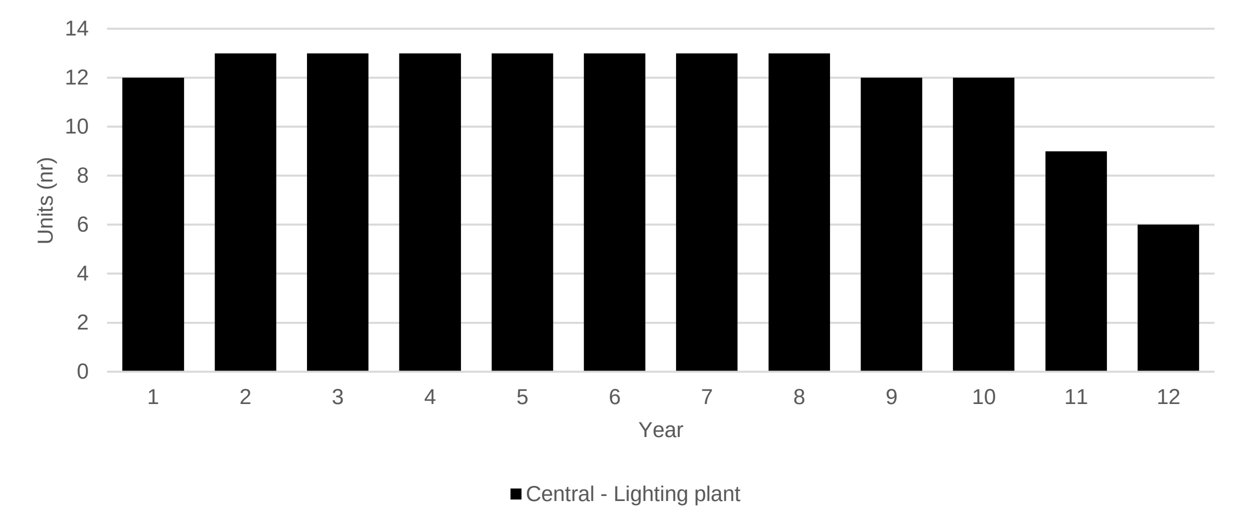

Figure 16-39: Diesel driven mobile lighting plant requirements per year

The useful life assumption for lighting plant is four years or 15 000 hours with replacements as shown in Table 16-35.

Table 16-35: Lighting plant utilisation and replacements

Unit number	Hours utilised initial unit	Hours utilised replacement unit 1	Hours utilised replacement unit 2
LP01	15 167	14 094	N/A
LP02	15 167	15 169	4 553
LP03	15 167	15 169	4 553
LP04	15 167	15 169	6 277
LP05	15 167	15 169	6 277
LP06	15 167	15 169	6 879
LP07	15 167	15 169	9 256
LP08	15 167	15 169	9 827
LP09	15 137	15 160	10 479
LP10	15 137	15 160	10 785

Unit number	Hours utilised initial unit	Hours utilised replacement unit 1	Hours utilised replacement unit 2
LP11	15 134	15 167	10 810
LP12	15 134	15 167	10 810
LP13	15 080	15 201	10 249

16.2.9.5.5. Tractor, loader, backhoe

Two TLBs were placed for the full duration of the project. Their primary functions include various ad hoc activities such as minor berm construction, general housekeeping etc, and to assist the blasting crew with blast hole stemming. A fixed monthly allowance of 220 hours was used per TLB. Since major repairs on these units exceeds the cost of a replacement unit, the replacement strategy targets replacements at 7 000 hours before major repairs are required. The remaining life was stretched to 10 000 hours to prevent unnecessary replacement capital during the last two years of the project as shown in Table 16-36.

Table 16-36: TLB utilisation and replacements

Unit number	Hours utilised initial unit	Hours utilised replacement unit 1	Hours utilised replacement unit 2	Hours utilised replacement unit 3
Cat TLB01	7 040	7 040	7 040	10 560
Cat TLB02	7 040	7 040	7 040	10 560

16.2.9.5.6. Site use buses

Operators are dropped at mine gate and taken to equipment with site use buses. An allowance of 2.4 hours per day was made per bus, with the number of buses determined from the labour histograms in Figure 16-50. The annual requirement for 40-seater buses is shown in Figure 16-40.

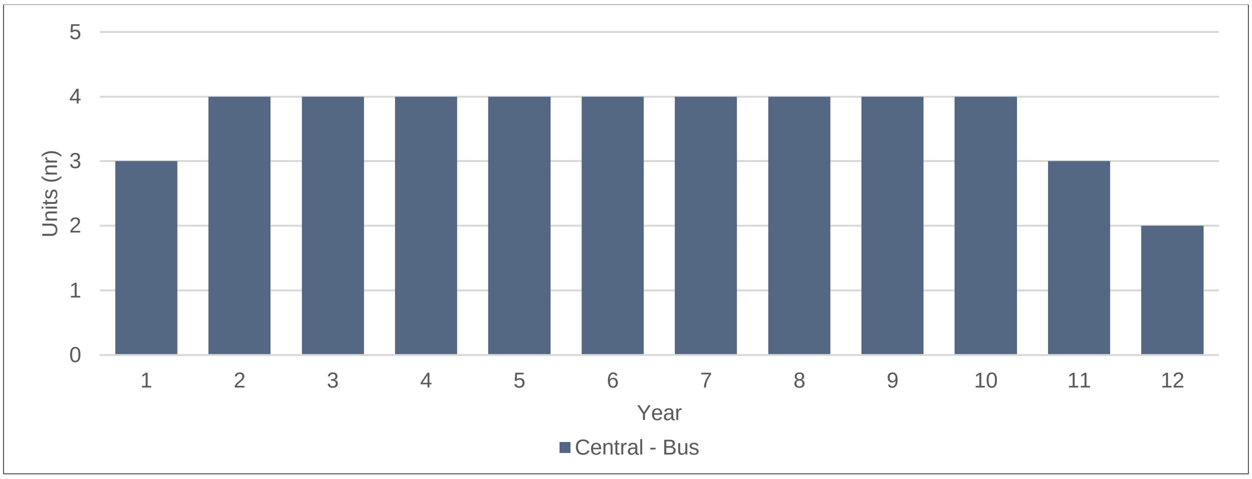

Figure 16-40: Site based 40-seater bus requirements per year

Since these buses only travel on site over short distances, no replacements during the project will be required. The final utilisation in kilometers travelled for each bus is indicated in Table 16-39.

Table 16-37: Site bus utilisation and replacements

Unit number	Kilometer utilised initial unit
Bus 01	86 088
Bus 02	95 184
Bus 03	98 760
Bus 04	97 464

16.2.9.5.7. Light delivery vehicles

LDVs were allocated based on job functions and, where possible, shared across different functions and shifts, with an allowance of 50 km per day for permanent dayshift and 150 km per day for LDVs shared across day and night shifts. The number of units required is shown in Figure 16-41.

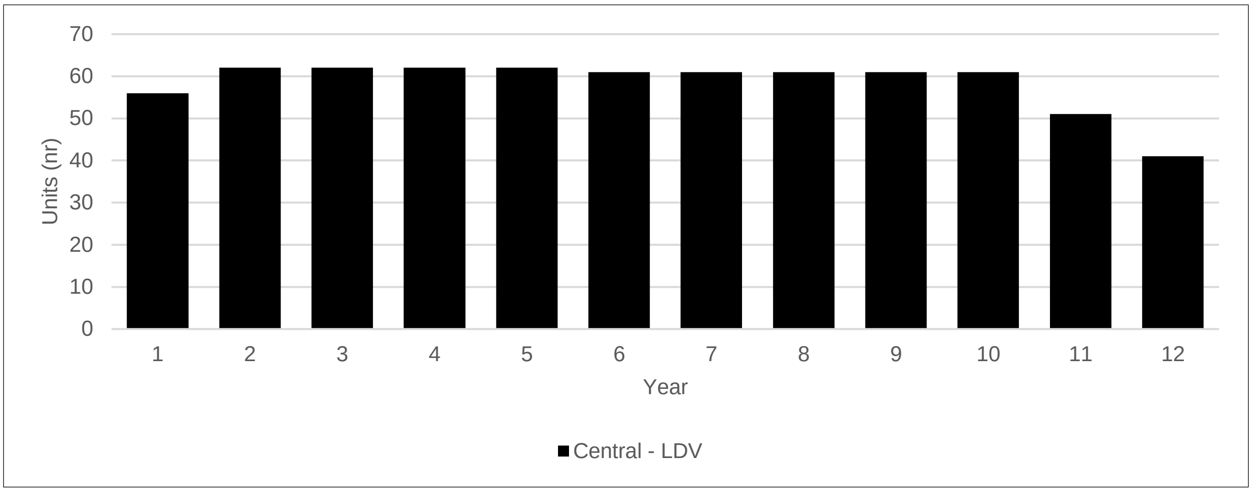

Figure 16-41: Site based LDV requirements per year

The replacement strategy applied with regards to LDVs was 200 000 km, meaning that all units are replaced once during the 12-year LOM.

In summary a total of three water pumps, 38 lighting plants, two diesel bowsers, four rock breakers, eight TLBs, 123 LDVs and four buses are purchased over the duration of the mine as shown in Table 16-38.

Table 16-38: Minor equipment purchased

Equipment	Cat	Initial units purchased [month]													Total	
		M1	M2	M3	M4	M5	M6	M7	M8	M9	M10	M11	M12	M13	Initial units	Sustaining units
Water pumps	Minor	3													3	
Lighting plants	Minor	8		2	2		1								13	25
Diesel bowsers	Minor	2													2	
Rock breakers	Minor	1					1								2	2
TLBs	Minor	2													2	6
LDVs	Minor	43		8	3		2	6							62	61
Busses	Minor	2		1			1								4	

16.2.9.6. Workshop equipment

Please refer to the engineering section for workshop mobile equipment requirements and utilisations. In summary, a total of four flat beds, two low beds, two forklifts and one tyre handler are purchased over the duration of the mine as shown in Table 16-39.

Table 16-39: Workshop equipment purchased

Equipment	Cat	Initial units purchased [month]													Total	
		M1	M2	M3	M4	M5	M6	M7	M8	M9	M10	M11	M12	M13	Initial units	Sustaining units
Flat beds	Workshop	2													2	2
Low beds	Workshop	1													1	1
Forklifts	Workshop	1													1	1
Tyre handlers	Workshop	1													1	

16.2.10. Explosives

Explosives requirements were calculated from the number of holes in Table 16-40, which were derived from the blast designs as discussed under Section 16.2.4.1 above.

The estimated annual requirements for bulk explosives are shown in Figure 16-42 and is determined by the average power factors of the blast designs. Bulk explosives are considered variable cost item with base rate of USD984.07 per tonne applied.

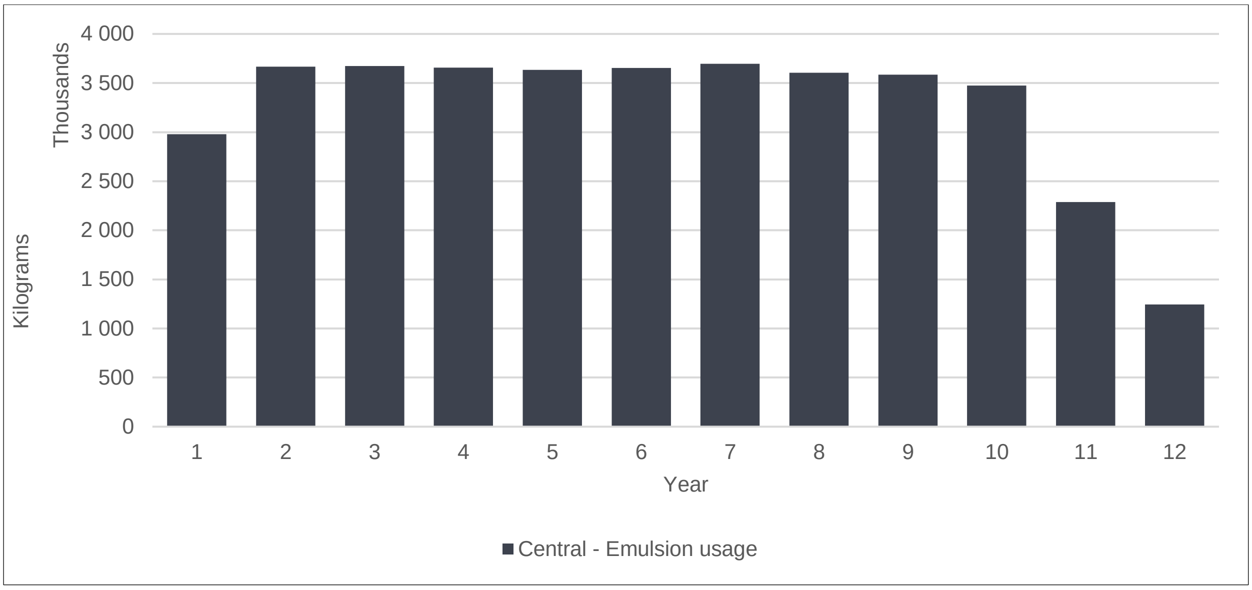

Figure 16-42: Annual bulk explosives requirements

Blasting accessory requirements are reflected in Table 16-40.

Table 16-40: Blasting accessory requirements

Explosives accessories	Unit	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
ED Axxis EDD 15m	ea.	13 733	16 894	16 945	16 877	16 792	16 930	17 117	16 734	16 586	16 095	10 656	5 875
Axxis 500 Surface Line	ea.	653	806	806	803	798	797	803	787	786	758	491	248
Primadet MS 15.0 m	ea.	3 433	4 223	4 236	4 219	4 198	4 233	4 279	4 183	4 147	4 024	2 664	1 469
Primadet EZ Trunkline 6.0 m	ea.	647	797	792	786	779	766	774	748	759	731	460	216
Riploc - Anchor stick, 600mm	ea.	1 620	1 890	2 058	2 502	2 574	4 031	4 859	4 566	3 942	3 147	3 841	5 573
Innopak SUPER 50 X 580	ea.	8 100	9 450	10 290	12 510	12 870	20 155	24 295	22 830	19 710	15 735	19 205	27 865
Gasbag	ea.	1 620	1 890	2 058	2 502	2 574	4 031	4 859	4 566	3 942	3 147	3 841	5 573
Primacord	m	17 893	20 846	22 715	27 586	28 376	44 421	53 500	50 293	43 395	34 701	42 321	61 386
Pentolite 150g booster	ea.	17 166	21 117	21 182	21 096	20 990	21 163	21 397	20 917	20 733	20 118	13 320	7 343
Primadet MS 3.0 m	ea.	1 620	1 890	2 058	2 502	2 574	4 031	4 859	4 566	3 942	3 147	3 841	5 573
Primadet lead-in-line 500m	ea.	105	126	128	130	131	145	153	148	142	131	103	91
IED 1.8 m 8D SA01 Copper Wire	ea.	105	126	128	130	131	145	153	148	142	131	103	91

The quantities were estimated based on an 80:20 split between electronic and pyrotechnic blasting, derived from the following:

- Electronic blasting
  - Each hole is primed with a 15 m electronic detonator, 150 g booster and then loaded with bulk explosives followed with stemming material. The electronic detonators are then tied up by surface line for electronic detonation

- Pyrotechnic blasting
  - Each hole is primed with a 15 m shocktube detonator, 150 g booster and then loaded with bulk explosives followed by stemming material. Rows are then tied up with 6 m trunk lines in accordance with the timing sequence and pattern. The blast block is then connected by means of Primadet lead-in line to a safe distance and initiated by means of an improvised explosive device ("IED").

The spacing between holes for presplit blasting makes electronic blasting cost prohibitive and since similar results can be obtained by means of conventional blasting the latter was used. In summary:

- Presplit blasting
  - Each hole is charged with five number 50 x 580 mm cartridge explosives. The cartridges are connected by means of Primacord and primed with a shocktube detonator suspended by means of ski-rope and an anchor stick. To prevent excessive air blast each hole is sealed by means of an airbag. The presplit is then connected by means of Primadet lead-in line to a safe distance and initiated by means of an IED.

A provision of USD30.30 per m³ of stemming material was made either from commercial sources or from onsite crushed material depending on availability. Estimated stemming material requirements are shown in Figure 16-43.

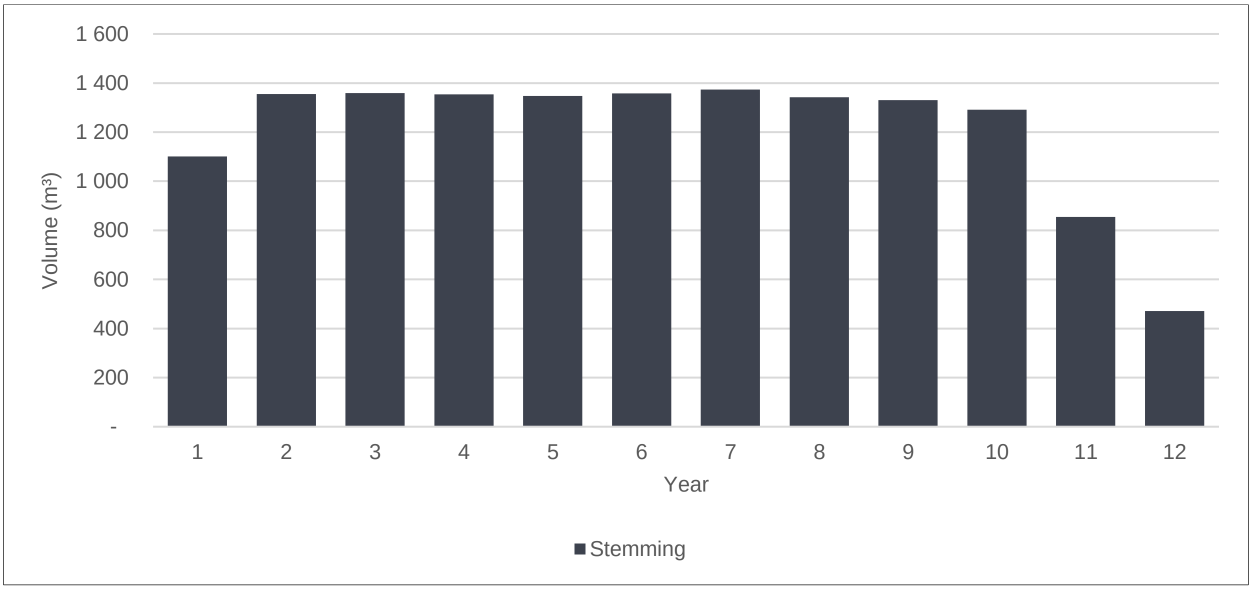

Figure 16-43: Annual stemming volumes required

16.2.11. Fuel – mining section

For details concerning fuel supply and logistics please refer to the engineering section of the report. Like explosives, fuel is considered a variable cost item. A base rate of USD0.97 per litre was applied in the cost model. The same fuel base rate was applied to LDVs, buses and other road-legal equipment which is permanently site based.

OEM guidelines with reference to similar projects were applied for average fuel consumption rates used in the costing as reflected in the Table 16-41.

Table 16-41: Equipment fuel consumptions

Equipment	Power [kW]	Load Factor [%]	Consumption[1] [L/hr]
Cat 395 Excavator	405	65	53
Cat 745 ADT	376	30	28
Cat D8 Dozer	271	45	30
Cat 330 Excavator with hammer	193	60	25
Cat 730 ADT water bowser	254	30	22
Cat 140 Grader	143	40	16
Cat 428 TLB	76	40	14
Cat 730 ADT diesel bowser	254	30	22
Sandvik DI650i drill rig	403		60 (2.1 litre per meter drilled)
Lighting plant with Kubota Z482	10		7
Water pump with Cat C15 engine	317		17
Komatsu WA600 FEL	374	50	51
LDV	110		13km / litre
40-Seater site bus	100		7km / litre
Lowbed	330		12
Flatbed	300		12
Forklift	50		3
Tyre handler	263		33

A fuel safety factor of 5 % was applied to cover against fuel overuse and theft.

In total it is estimated that 106.1 million litres of fuel is required over the duration of the mine with annual requirements of up to 10.5 million litres, as shown in Figure 16-44. The average fuel requirement is 1.22 litres per bcm mined. The slight increase in the fuel used per bcm mined year 11 and 12 is related to equipment placed on fixed monthly hours while the mining volume decreases.

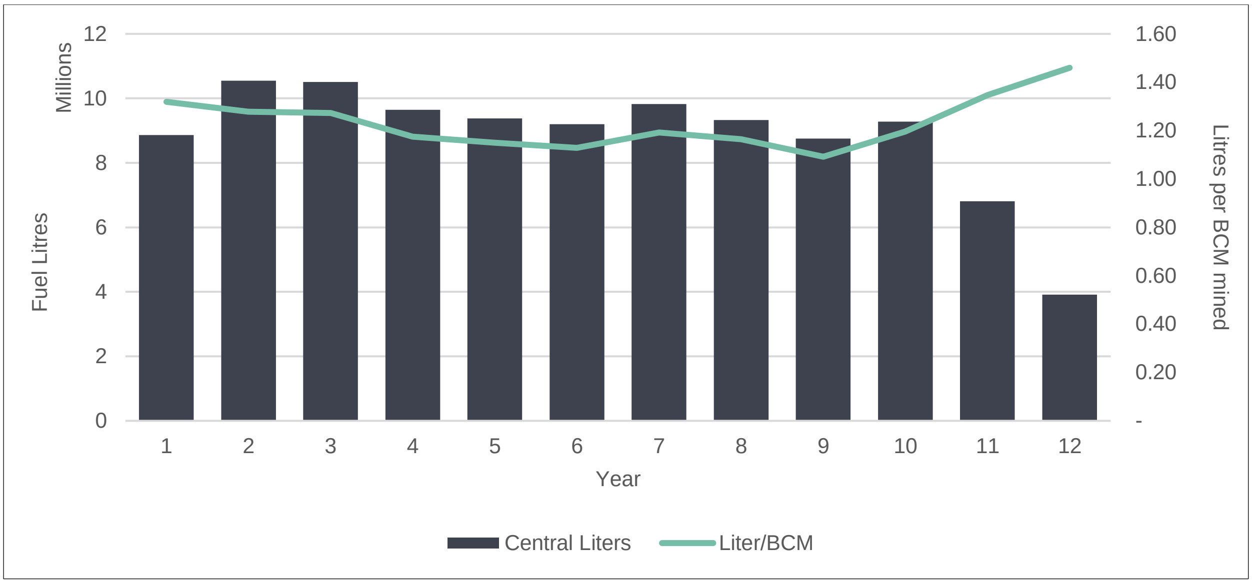

Figure 16-44: Total fuel required for mining

Figure 16-45 shows the positive correlation between haul trucks litres used per bcm mined and the average hauling distance.

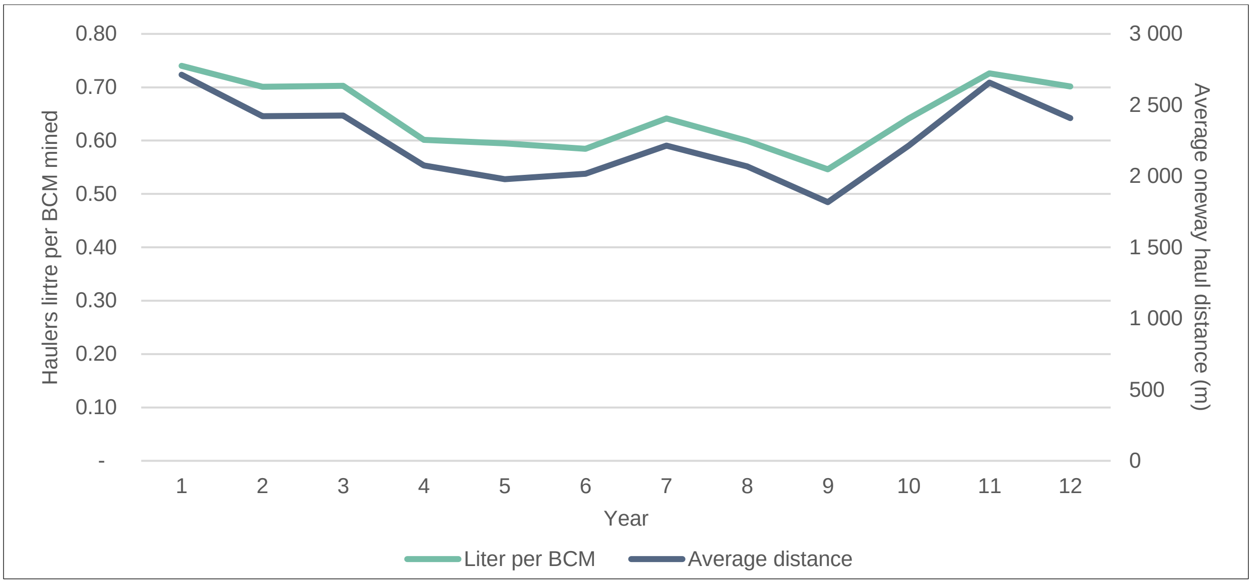

Figure 16-45: Correlation between haul truck fuel use per bcm and distance

16.2.12. Mining personnel

The mining operation is managed by four key departments reporting to the Mining Manager:

1. Technical services department
2. Drill and blast department
3. Production department
4. Engineering department

The organisational charts in Table 16-42 outline the operational structure of the project with roles responsible for the day-to-day activities of mining, including drilling, blasting, excavation and hauling. Job levels were allocated to each position in accordance with Table 16-42 taken from the report by Align Advisors: “Benchmark Salary Report, Zambian Mining Industry 2024”.

Table 16-42: Job categories for mining personnel

Level	Job category	Example positions
14	GM II	General Manager Operations - Large Site / Multiple Sites, Senior Executive, Senior VP
13	GM I	Country Manager, General Manager Operations - Small / Medium Site, Executive, VP
12	Senior Manager	Operations Manager, Large Manager Role
11	Manager II	Geology Manager, Mining Manager, Technical Services Manager, Mill / Process Manager, Engineering / Maintenance Manager
10	Manager I	Finance & Administration Manager, IT Manager, HR Manager, Environmental Manager, Health & Safety Manager, Loss Prevention / Security Manager, Supply & Logistics Manager
9	Superintendent II	Mining Superintendent, Maintenance Superintendent, Mill / Process Superintendent, Geology Superintendent, Chief Engineer, Chief Geologist
8	Superintendent I	Environmental Superintendent, Finance & Accounting Superintendent, Supply Superintendent, Health & Safety Superintendent, HR Superintendent, Chief Surveyor, Security Superintendent
7	Senior Professional / Supervisor High	Senior Metallurgist, Senior Process Engineer, Senior Mining Engineer, Senior Geologist General Foreman
6	Full Professional/ Supervisor II	Senior HR Officer, Senior IT Specialist, Mine Planning Engineer (Long Term), Project Geologist, Geotechnical Engineer, Metallurgist, Senior Maintenance Planner, Senior Accountant, Project Engineer Mining Senior Supervisor
5	Supervisor/ Full Professional	Mining Supervisor, Electrical Supervisor, Instrumentation Supervisor, Maintenance Supervisor, Supply & Logistics Supervisor, Health & Safety Supervisor, Control Room Supervisor Experienced Professional: IT Administrator, Mining Engineer, Mine Planning Engineer (Short Term), Drill and Blast Engineer, Mine Geologist, Exploration Geologist, Maintenance Planner, Training Officer, Human Resources Officer, Accountant
4	Professional, Snr Clerical Snr Skilled Worker, Team Leader	Office Manager, Executive Assistant to Site General Manager, First Chef, Buyer, Nurse, Graduate Engineer, Security Supervisor, Assistant Accountant, Help Desk Officer Operations Senior Leading Hand, Maintenance Planner, Training Officer
3	Clerical II Snr Operator/ Snr Technician	Payroll Officer, Administration Officer, Human Resources Assistant, Section Chef, Document Controller, Driver Supervisor Leading Hand, Senior Operator, Large Excavator / Jumbo Operator, Senior Tradesperson (Highly Skilled & Qualified), Senior Technician (Highly Skilled & Qualified)
2	Clerical I Operator II/ Technician I	Administration Assistant, Housekeeping Leading Hand, Nurse Assistant Crane Operator, Large Ancillary Operator (Dozer, Grader), Survey Technician, Qualified Maintenance Tradesperson
1	Operator I / Technician Assistant	Entry Level Skilled Operator, Small Ancillary (Roller, Compactor, Back Hoe), Dump Truck Driver, Technician Assistant, Lab Assistant Guard, Assistant Chef, Storeperson, Driver
0	Unskilled / Labourers	Cleaner, Gardener, Entry Level Labourer, Unskilled Spotter, Unskilled Sampler

16.2.12.1. Technical services department

The technical services department is overseen by the appointed technical services manager, and is responsible for:

- Mine planning and design develops short-term, medium-term, long-term mine plans. Designs pit layouts, haul roads, waste dumps and stockpile areas. Ensures mine designs comply with safety, environmental and economic requirements
- Surveying provides accurate topographical and volumetric data, monitors progress of excavation, stockpile volumes and waste dumps
- Geotechnical engineering assesses slope stability and designs safe pit walls and waste dumps. Monitors ground conditions and mitigates risk like landslides or failures
- Grade control monitors ore quality and ensures compliance with production targets. Manages selective mining of ore material and reduces ore dilution and waste contamination through precise planning
- Mineral Resource and Mineral Reserve estimation conducts analysis of geological data to estimate Mineral Resources, updates Mineral Resource models based on new exploration or production data, prepares reports for compliance with mining regulations and stakeholders.

The personnel reporting to the technical service manager are shown below in Figure 16-46.

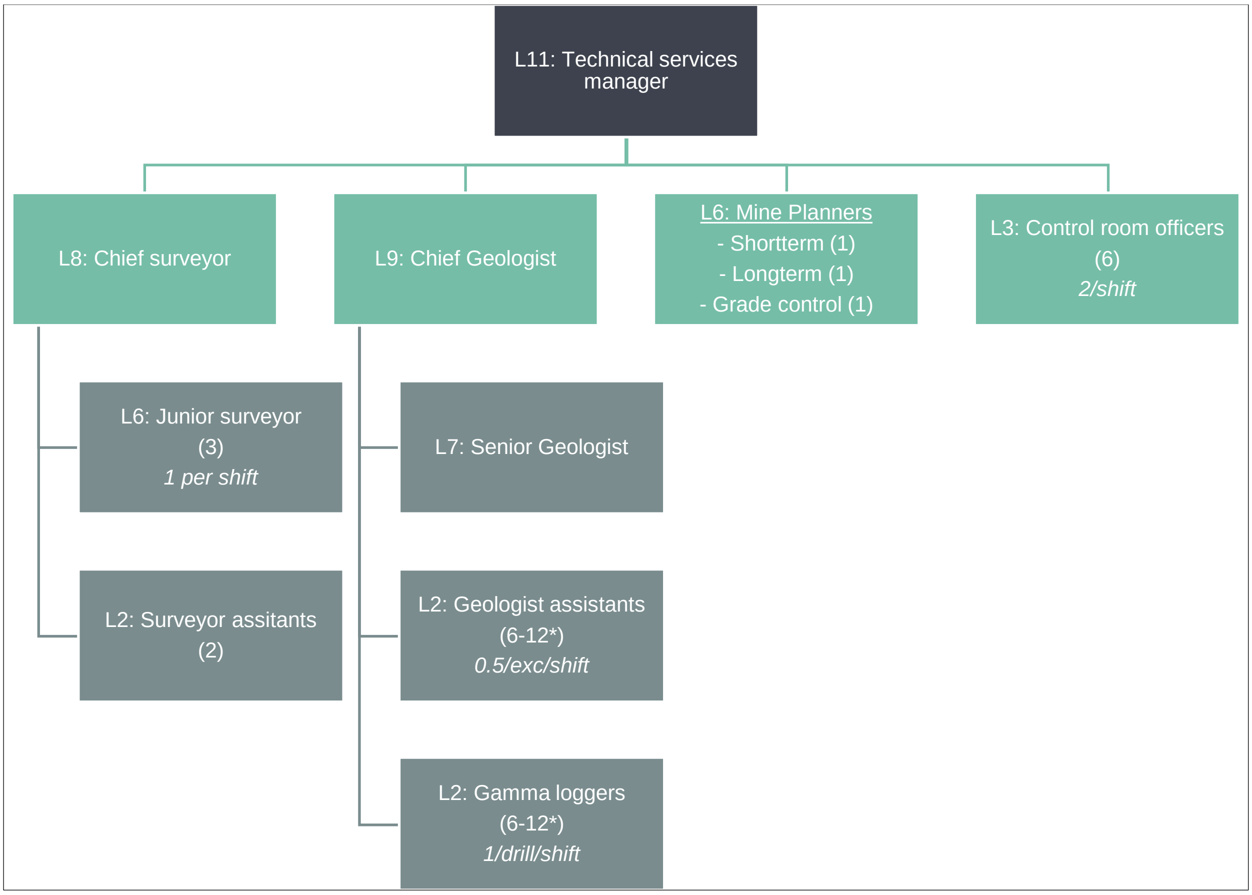

Figure 16-46: Technical services department organisational structure

16.2.12.2. Drill and blast department

The drill and blast department is a critical component of surface mining operations. Its primary role is to design and execute drilling and blasting activities to fragment the rock efficiently, enabling its removal and processing. Figure 16-47 shows the personnel allocated to the drill and blast department.

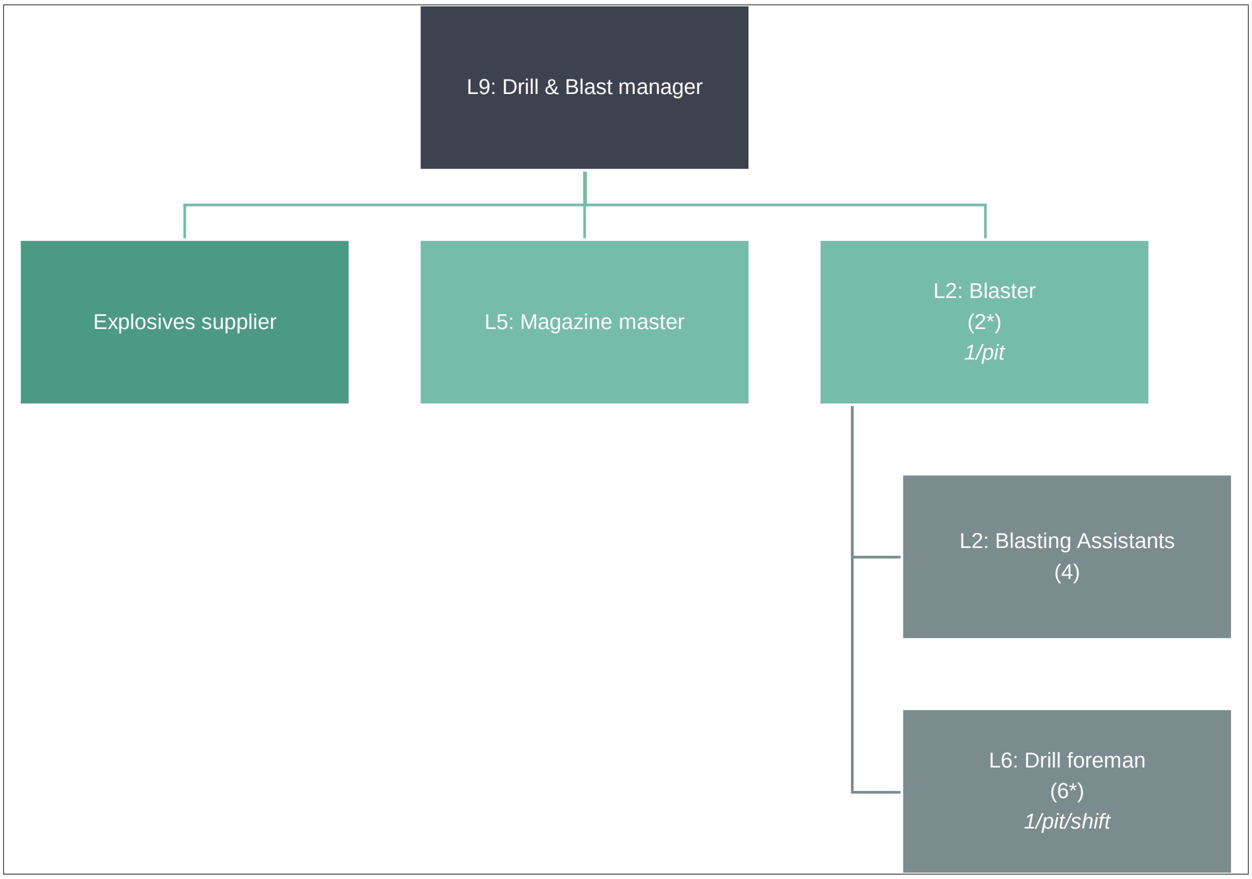

Figure 16-47: Drilling and blasting department organisational structure

As covered under Section 16.2.3 the selected explosives supplier will provide a “DTH” service. Included in the supplier’s fixed monthly fee is the following personnel:

- 1 x Site manager
- 5 x Mobile manufacturing unit (“MMU”) operators
- 6 x MMU assistants
- 1 x MMU mechanic
- 1 x Safety officer.

16.2.12.3. Production department

To assist with operator competency training, provision was made in the mining cost for the following personnel:

- Operational readiness training officer (1st 12 months)
- Full time mobile equipment training officers x 3
- Training clerk

In addition to the above, allowance was made for external off-site practical operator training through the National Council for Construction in Lusaka over a four-week period before project startup as well as further onsite operator training for five days and technician development training through the OEM suppliers during the startup period.

Figure 16-48 shows the production department which reports to the mining manager through the responsible production manager. The production manager is supported by one senior mining foreman, one deputy mining foreman on each shift overseeing both Muntanga and Dibbwi East, and one junior foreman for each shift shared between two load and haul teams.

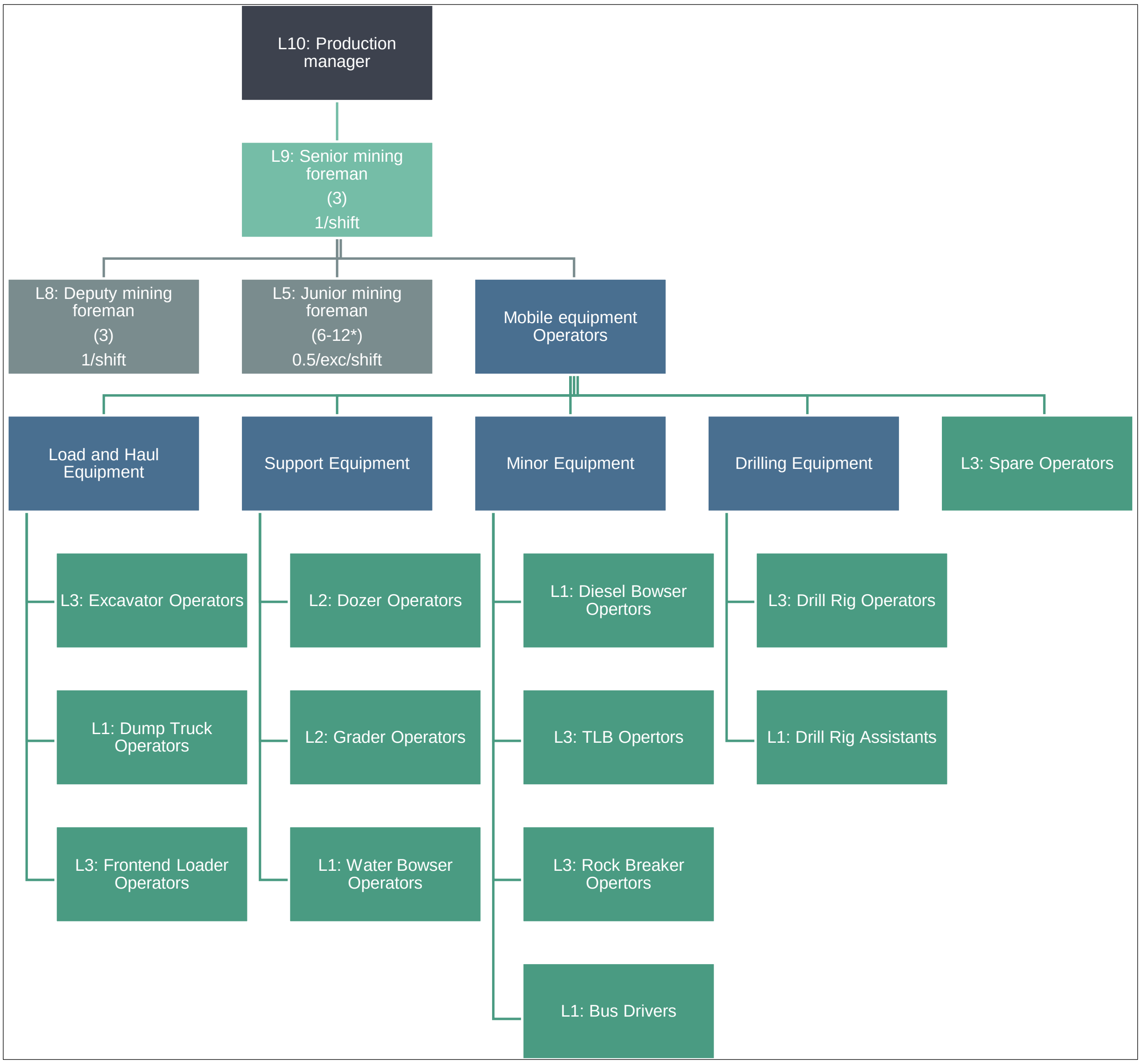

Figure 16-48: Production department organisational structure

16.2.12.4. Engineering department

Figure 16-49 shows the Engineering department organisational structure.

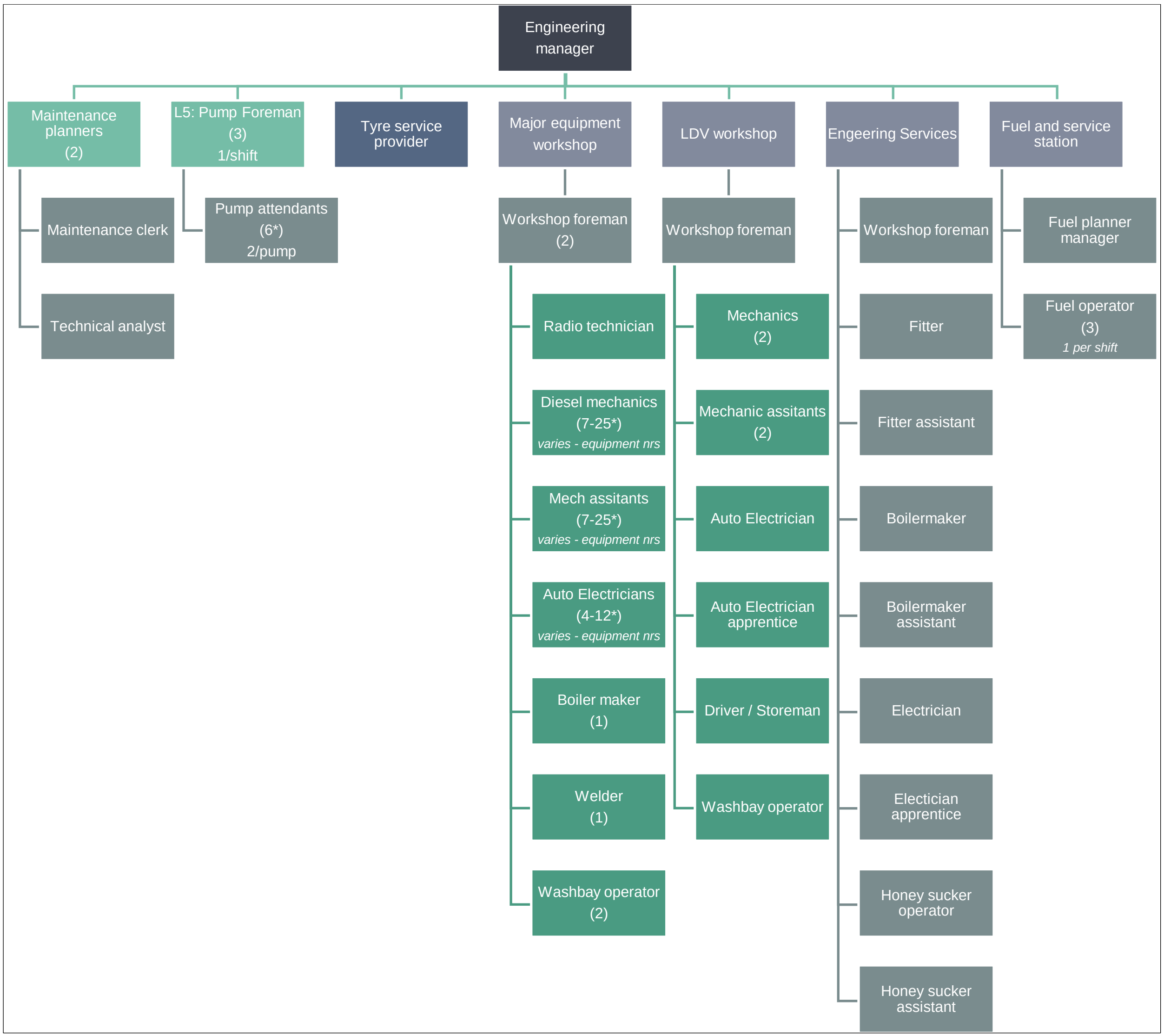

Figure 16-49: Engineering department organisational structure

Equipment maintenance will be performed by the owner on site at the mining maintenance facilities as described in the mining infrastructure section. A mining engineering manager will be responsible for legal compliance and management of all maintenance-related activities on site, reporting to the mine manager.

Maintenance activities consist of inspections, services and repairs for mining equipment, support equipment and infrastructure. Opportunity exists where mechanics will be able to service equipment during operational stoppages due to a two-shift roster for operations, which allows a four-hour break in each day. The staffing requirements per piece of equipment were calculated based on the maintenance activities and mechanic/ artisan available time indicated in the Table 16-43. Maintenance staff will be available during operation shifts as well as day shift.

Table 16-43: Maintenance activities for mobile equipment

Maintenance activity	Duration [hrs.]	Done by	Hours per equipment per year [hrs]
Daily inspections and top up	0.5	Shift	183
500 hr service intervals (1 per month)	8	Dayshift	96
7 500 hrs rebuild (1 week every 417 days)	56	Dayshift	49
Major Breakdowns (1 per year)	24	Dayshift	24
Sub total			**352**
Other maintenance (major component replacement, tyres, etc.)			51
Assume only 50 % of maintenance downtime needs mechanic attendance			403
Services hours			221
Shift Artisan available time/ ops stoppage	6	Shift	
Workshop artisan available time	8	Dayshift	

Artisan full time equivalent needs were calculated considering the number of pieces of equipment, maintenance activity type and wrench time available per artisan. All maintenance activities are planned by the maintenance planning department consisting of:

- Two maintenance planners
- Maintenance clerk, and
- Technical analyst.

Mining equipment inspections are executed during shift changes, stoppages and refuelling while services are scheduled according to equipment usage and OEM recommendations to maintain warranties and equipment integrity.

Trucks and other mobile equipment, fitted with tyres, are inspected at the service station, workshop or parking area by the artisans on shift, aligned with the mining operations. Track-mounted mobile equipment is inspected at the working front when not in operation. All equipment will be inspected daily as part of the proactive equipment maintenance strategy.

Defects that cannot be repaired during inspections are noted and communicated to supervisor and planning department for follow up work to be scheduled. Repairs are conducted at the major equipment workshop at the Central complex by the day shift artisans.

The mining equipment maintenance operates under the supervision of the workshop foreman, including the service station and wash bay. Mechanics and auto electrician full-time equivalents vary over the life of mine as the numbers are linked to the amount of equipment. A radio technician, boiler maker and welder would form part of the team to attend day-to-day maintenance activities.

Equipment major component change out and rebuilds are planned and will be executed by the day shift artisans with assistance from the shift and the engineering services artisans.

Two wash bay operators are included to ensure redundancy as all equipment is cleaned before being serviced or repaired.

The engineering services workshop attends all infrastructure maintenance under supervision of the engineering services foreman. This includes the pit de-watering pumps and pipelines, electrical supply network, lighting, buildings, potable water, raw water, sewage, fire systems and roads.

Pit dewatering pumps and pipelines are inspected by the engineering services fitter daily. The fitter, boilermaker and electrician are responsible to maintain the integrity of the pit dewatering systems. They will assist during major equipment rebuilds when required.

A honeysucker operator and assistant form part of the engineering services team. They will be responsible for collection of sewage from remote locations on the mine to be treated at the main sewage facility at the plant.

Service and support equipment (such as maintenance trucks, bowsers, light delivery vehicles (“LDVs”) and forklifts) is inspected once a week at the LDV workshop. The LDV workshop foreman will be responsible for all mobile equipment on the mine (excluding the mining equipment). Two mechanics and an auto electrician will attend to all service and support equipment for mining and the plant. A driver and wash bay operator forms part of the structure to ensure that equipment is collected and cleaned before attended to by the artisans.

The labour for tyre services forms part of the service contract and does not require client resources. Fuel is managed by the fuel service provider up to the service station. One foreman per shift will be responsible for planning the dispensing of fuel to the remote equipment. Pump attendants and bowser operators are responsible for the filling of equipment and will report to the foreman.

16.2.12.5. Labour rates

The monthly labour rates from the report from Align Advisors, who undertook a comparison of renumeration of mining operations across Zambia, were used with an additional allowance provided for severance, personal protective equipment ("PPE"), periodic medical examinations and mobile airtime as indicated in the Table 16-44.

Table 16-44: Labour cost to company (monthly)[1]

Level	Base salary	Housing allowance	Other allowances	Gross salary	Tax	Severance	PPE	Periodic medical	Air-time	Total CTC
13	12 800	2 304	16	15 120	605	2 133	23	5	8	17 894
12	10 400	1 872	16	12 288	492	1 733	23	5	8	14 549
11	8 000	1 600	16	9 616	385	1 333	23	5	8	11 370
10	5 600	1 288	16	6 904	276	933	23	5	8	8 149
9	3 800	950	16	4 766	191	633	23	5	8	5 626
8	2 200	594	16	2 810	112	367	23	5	8	3 325
7	1 244	336	16	1 596	64	207	23	5	8	1 903
6	916	256	16	1 188	48	153	23	5	8	1 424
5	696	195	16	907	36	116	23	5	8	1 095
4	528	158	16	702	28	88	23	5	8	854
3	412	124	16	552	22	69	23	5	-	670
2	344	103	16	463	19	57	23	5	-	567
1	296	89	16	401	16	49	23	5	-	494
0	236	71	16	323	13	39	23	5	-	403

1. Converted to USD from Zambian Kwacha (1USD:25ZMW)

The total mining labour requirement, grouped by department, is shown in Figure 16-50. There is a maximum of 482 staff in year 4 (excluding external service providers).

Mine management, consisting of the mining manager and heads of department (technical services, operations, engineering and drill and blast), remains consistent throughout the LOM, potentially rotating roles when required.

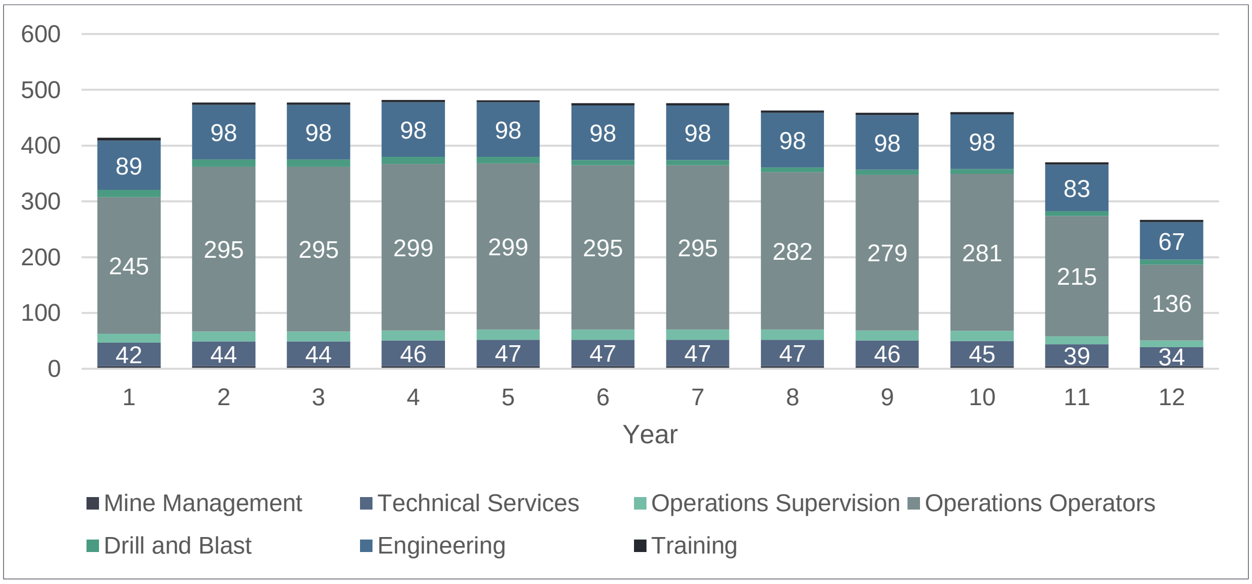

Figure 16-50: Labour histogram per department

Pareto analysis of the labour numbers shown in Figure 16-51 shows that employees are concentrated in grades 1 and 2, accounting for a substantial majority of the workforce. A small number of grades contribute to the majority of employment.

Grades 1, 2 and 3 are where most operators are classified, which is expected, as in surface mining projects operation of mobile equipment is normally the largest employment contributor.

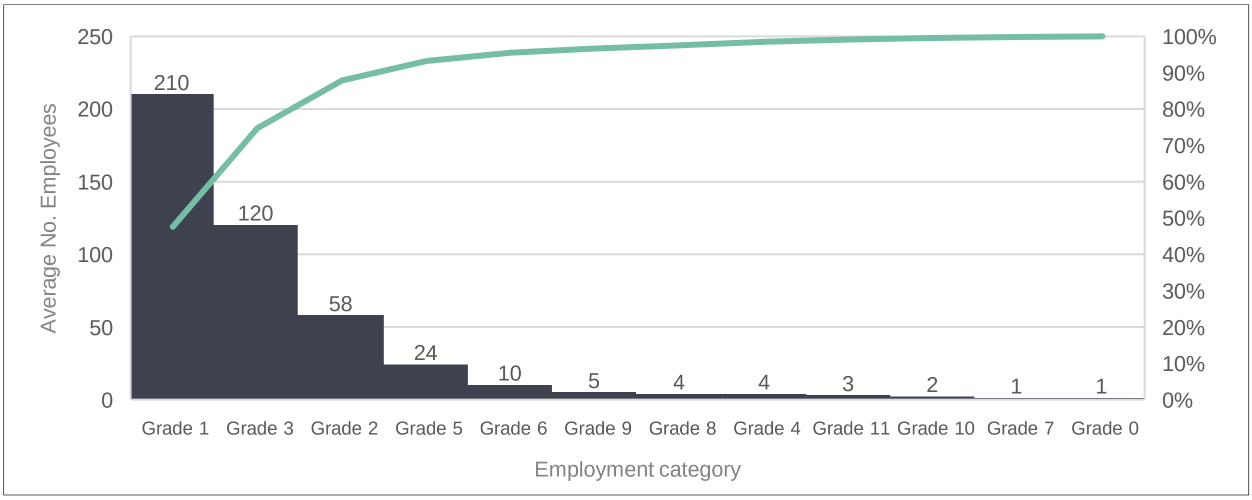

Figure 16-51: Pareto chart reflecting average labour number per employment category

A similar Pareto done on a total cost basis is shown in Figure 16-52, where in contrast with employee numbers, the cost contribution is more spread out across the different categories reflecting potential higher cost sensitivity in higher grade allocations, whereas lower grades are sensitive to labour rates with potential negative cost impacts if competent employees cannot be found and retained at the labour rates as indicated in Table 16-44.

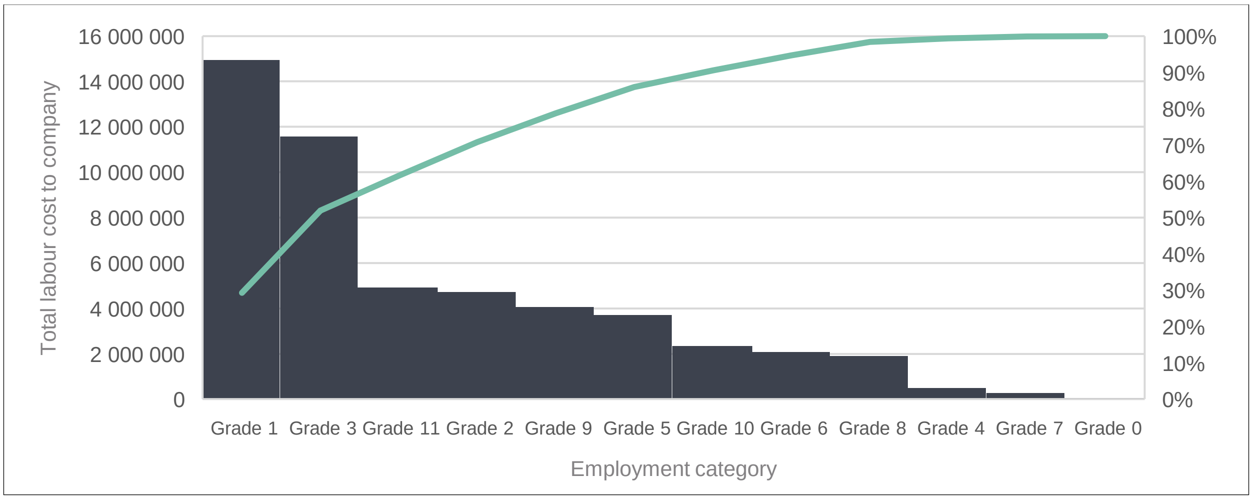

Figure 16-52: Pareto chart reflecting average labour cost per employment category

17. Project recovery methods

17.1. Introduction

The Central processing plant ("CPP") is designed to handle a total of 3.5Mtpa of ROM material sourced from the central Muntanga and Dibbwi East mining sites and, if mined in the future, sorted ore from the Dibbwi, Gwabi and Njame satellite mining sites. The mix of ore from the respective pits will vary over time.

Processing of the ROM ore to produce saleable U_3O_8 product takes place in three stages:

1. **Ore preparation:** ROM ore hauled from the pits is placed into the ROM tipping bin and enters three stages of crushing, before undergoing agglomeration in preparation for leaching
2. **Heap leach:** The agglomerated ore is placed on the HLF for leaching. The pregnant leach solution is pumped to the uranium recovery and purification plant, and the spent ore is placed on the spent ore dump
3. **Uranium recovery and purification:** Uranium recovery by IX followed by the recovery of U_3O_8 by eluting the uranium-loaded resin using a sulfuric acid solution. The eluate undergoes a nanofiltration process facilitating sulfuric acid recovery for recycling to the elution process. Following this, the concentrated solution is carefully dosed with hydrated lime and sodium hydroxide to neutralise residual acid and remove deleterious minerals such as iron, after which it is dosed with hydrogen peroxide, leading to uranyl peroxide precipitation. The precipitate is dewatered and calcined, and the final product and packed into drums as U_3O_8 or yellowcake.

The overall process flow sheet is shown in Figure 17-1.

The design is based on receiving 3.5 Mtpa of ore, with a steady-state average head grade from the central pits of 320 ppm U_3O_8. The processing plant is capable of producing 2.3 Mlb per annum of saleable U_3O_8 during steady-state operation.

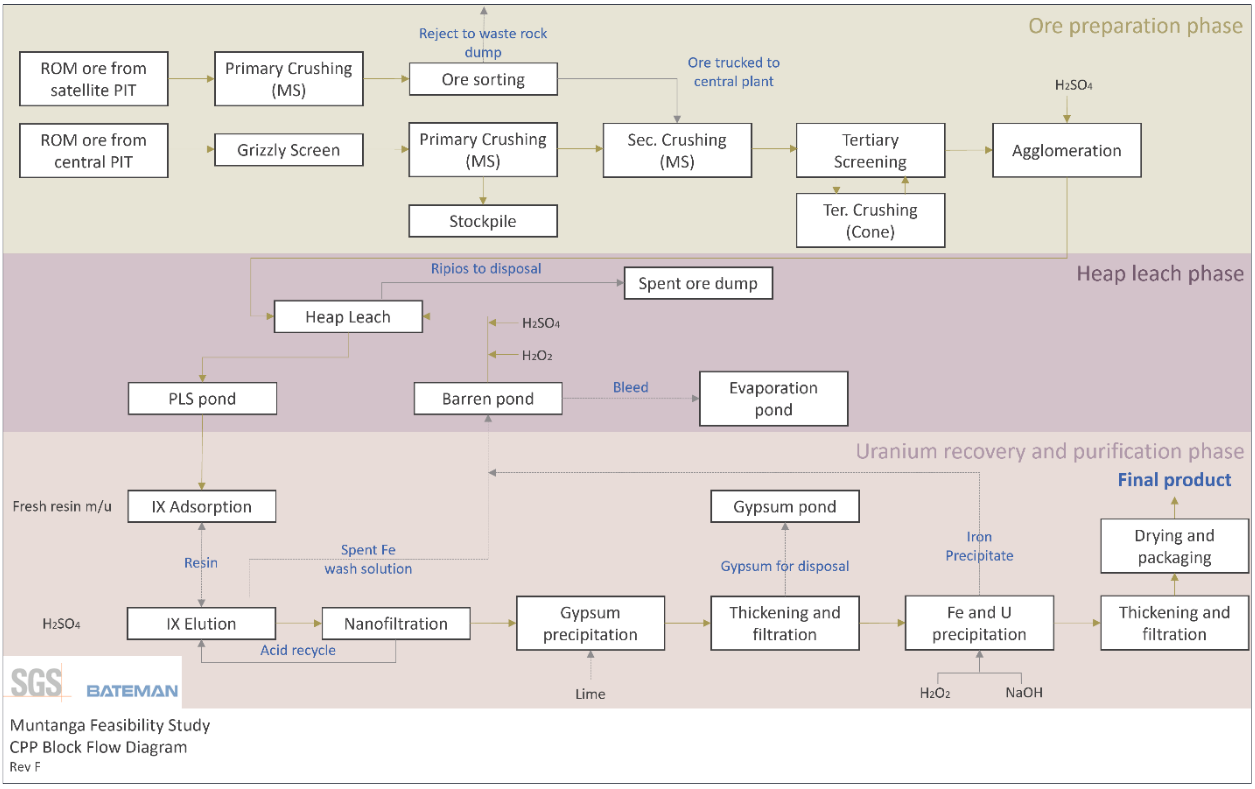

Figure 17-1: Block diagram of the CPP

17.2. Ore preparation

The ore preparation stage handles a total of 3.5 Mtpa of ROM material trucked from the central Muntanga and Dibbwi East mining sites, and if mined, radiometrically-sorted ore from the Dibbwi, Gwabi and Njame satellite mining sites. The mix of ore from the respective pits will vary over time.

The flow sheet for ore preparation encompasses a ROM tip, primary, secondary, and tertiary crushing stages, and an agglomerator.

17.2.1. Process design basis

A summary of key parameters is provided in Table 17-1.

Table 17-1: Summary of key parameters for ore preparation

Parameter	Unit	Value
Ore source		
Muntanga and Dibbwi East pits	tpa (max)	3 500 000
Dibbwi, Gwabi and Njame pits (after Rados sorting)*	tpa (max)	350 000
Combined ore throughput	tpa	3 500 000

*If mined

17.2.2. Process description

This section describes the ore preparation stage and should be read in conjunction with the following set of process flow diagrams ("PFDs"):

- M7610-C01003-P120-001-001Heap Leach & CIX Plant Primary Crushing - PFD
- M7610-C01003-P120-001-002Heap Leach & CIX Plant Stockpile - PFD
- M7610-C01013-P120-001-001Heap Leach & CIX Plant Secondary Crushing - PFD
- M7610-C01023-P120-001-001Heap Leach & CIX Plant Tertiary Crushing - PFD
- M7610-C01033-P120-001-001 Heap Leach & CIX Plant Agglomeration - PFD

17.2.2.1. Ore receiving and primary crushing

PFD Reference: C01003P120001001

Ore mined from the various central open-cast pits is hauled to the ROM (C0100-BN-001) feed bin using the mining haulage fleet.

The ore-receiving section of the plant has a single ROM bin (C0100-BN-001) which feeds onto the primary mineral sizer apron feeder (C0100-AF-001). Ore is fed into the ROM bin by direct tipping from the 60 tonnes ("t") mining trucks or by FEL servicing the stockpile. The bin will have a flat 750 mm static grizzly (C0100-SC-001) to retain oversize material in the feed. The rock breaker (C0100-RB-001) will be utilised to break down oversized material. The ROM bin is equipped with an apron feeder (C0100-AF-001) that draws ore from the bottom of the bin to feed into the primary mineral sizer (C0100-MS-001). The feed to the primary mineral sizer is controlled by varying the speed on the variable speed drive on the apron feeder.

A reduction ratio of 3:1 is anticipated within the primary mineral sizer targeting a p80 of 250 mm. The primary mineral sizer discharge conveyor (C0100-CV-001) is equipped with a metal detector (C0100-MD-001) and a belt magnet (C0100-MG-001) to pick up any metal pieces that would have escaped the static grizzly (C0100-SC-001) discharges onto the crushed ore stockpile. The magnet is installed to protect the secondary mineral sizer from large pieces of metal.

Process water is pumped into the dust suppression water tank (C0100-TK-001). The water is pumped through a water filter (C0100-FL-001) and then distributed to the ore transfer points, i.e. on top of static grizzly and belt feeder discharge chutes by using a dust suppression pump (C0100-PP-001). The dust suppression water sprays prevent the generation of excessive dust around the bins.

The ROM bins are situated in a bunded area comprising a slurry collection sump (C0100-SM-001) and a spillage pump (C0100-SP-001). The fine spillage is washed into the sump and pumped to the crushed ore stockpile. The bulk spillage is moved by mechanical equipment and reloaded onto the stockpiles.

Ore mined from the various satellite open-cast pits is hauled to the ROM (A0100-CH-001) feed chute using the mining haulage fleet and fed into the feed chute of the primary sizer station by direct tipping from the 45 t mining trucks (although up to 60 t trucks can be accommodated) or by FELs. The primary sizer station (A100-VP-001) will produce a product with a p80 of 110 mm and directly feed the pair of radiometric ore sorters in A100-VP-002. The ore sorting package produces rejects, which are conveyed onto the rejects stockpile (A100-SK-001) and sorted ore conveyed to the sorted ore stockpile (A100-SK-002). The stockpiles are reclaimed by FEL and transported to either the WRD in the case of the rejects or the central plant by a dedicated road haulage fleet in the case of the sorted ore.

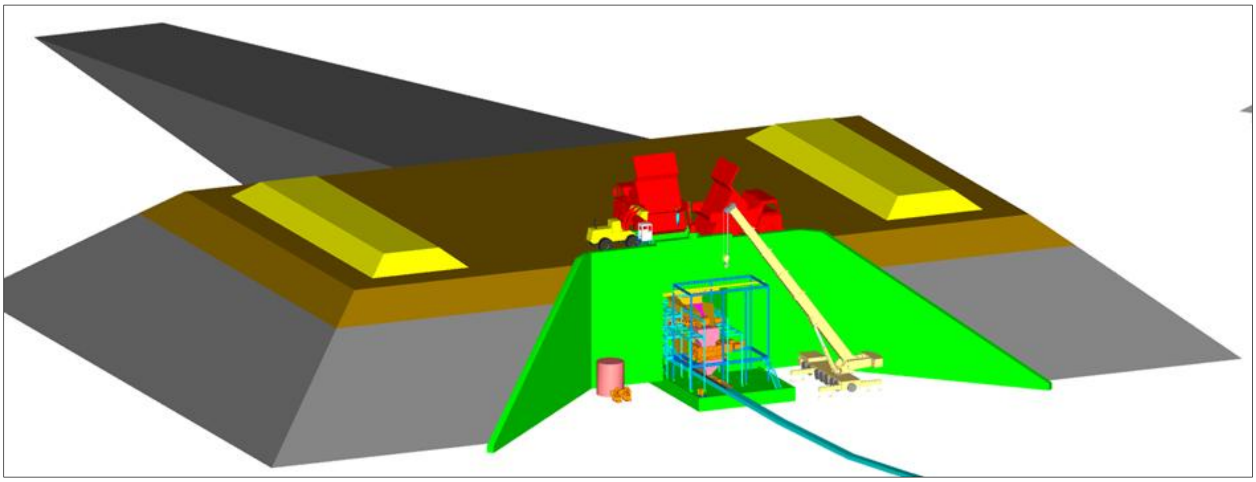

Figure 17-2: ROM ore tip

17.2.2.2. Crushed ore stockpiling

PFD Reference: C01003P120001002

The crushed ore derived from the primary crushing circuit is directed towards the stockpile feed conveyor (C0100-CV-001). Subsequently, this crushed ore is deposited onto the stockpile (C0100-SK-001) the crushed ore will be conveyed via stockpile apron feeders (C0102-AF-001/002/003/004) to the secondary crusher. The hydraulic power pack will be situated in a hydraulic room, equipped with a fire suppression system. Ventilation will be facilitated by extraction fans (C0102-FA-001/002) to ensure proper ventilation within the tunnel beneath the stockpile.

Process water is introduced into the dust suppression water tank (C0100-TK-002) and is distributed to ore transfer points via the dust suppression pump (C0100-PP-003). The water sprays from the dust suppression system are crucial in preventing the generation of excessive dust around the bins. Process water will be directed to wash points located at the head (C0100-WP-001) and tail (C0100-WP-002) of the conveyor belt.

The area incorporates a slurry collection sump (C0100-SM-002) and a spillage pump (C0102-SP-002). Fine spillage is directed into the sump and subsequently pumped to the heap leach stack, while mechanical equipment manages and reloads bulk spillage onto the crushed ore stockpile (C0100-SK-001).

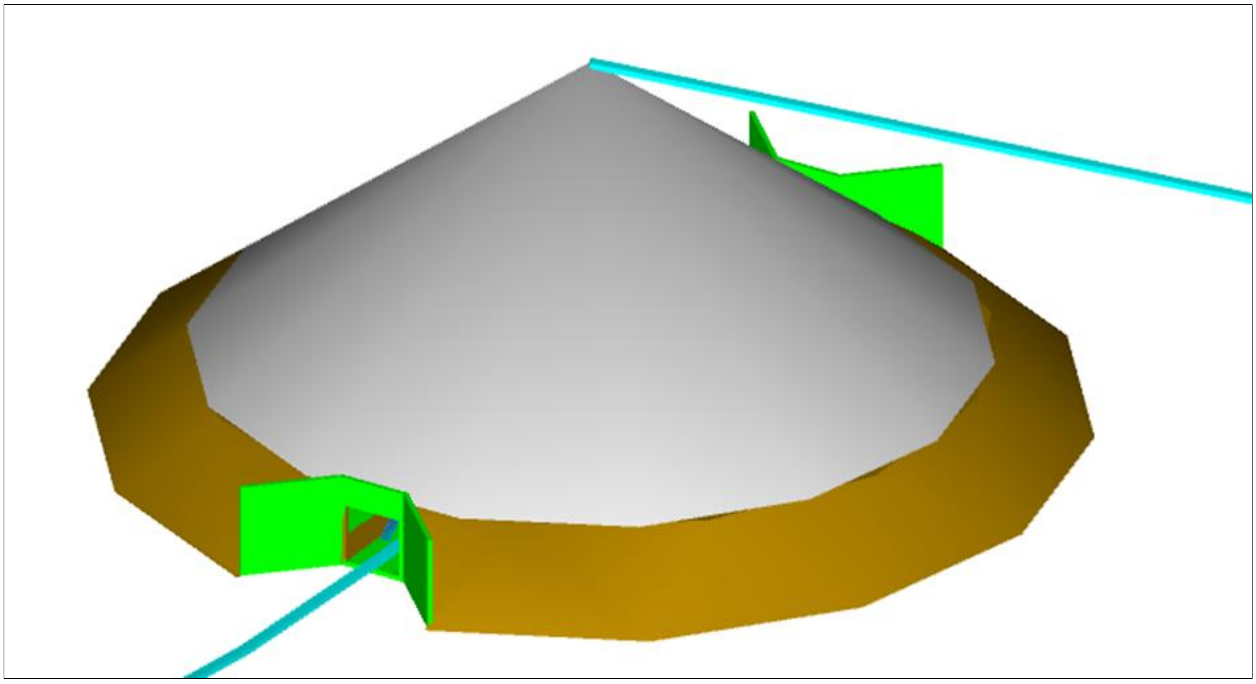

Figure 17-3: Crushed ore stockpile

17.2.2.3. Secondary crushing

PFD Reference: C01013P120001001

The crushed ore is directed via three apron feeders into a single feed bin (C0101-BN-001) feeding the secondary mineral sizer (C0101-MS-001) through the primary mineral sizer discharge conveyor. Equipped with a vibrating feeder (C0101-FD-001), the bin extracts ore from its base to supply the secondary mineral sizer. The speed of the apron feeder's variable speed drive controls the feed to the secondary mineral sizer. Ore trucked from the satellite pits is introduced to the process by direct tipping into the reload chute.

A reduction ratio of 3.33:1 is anticipated within the secondary mineral sizer targeting a p80 of 75 mm. The secondary mineral sizer discharge conveyor (C0101-CV-001) is fitted with a metal detector (C0101-MD-001) and a belt magnet (C0101-MG-001) designed to capture any large metal pieces that may have escaped, safeguarding the tertiary cone crusher from potential damage.

Process water is introduced into the dust suppression water tank (C0101-TK-001). After passing through a water filter (C0101-FL-001), the water is distributed to ore transfer points using the dust suppression pump (C0101-PP-001). The water sprays from the dust suppression system prevent the generation of excessive dust around the bins.

The feed bins are located within a contained area that includes a slurry collection sump (C0101-SM-001) and a spillage pump (C0101-SP-001). Fine spillage is directed into the sump and pumped to the crushed ore stockpile, while mechanical equipment handles and reloads bulk spillage back onto the plant feed stockpile.

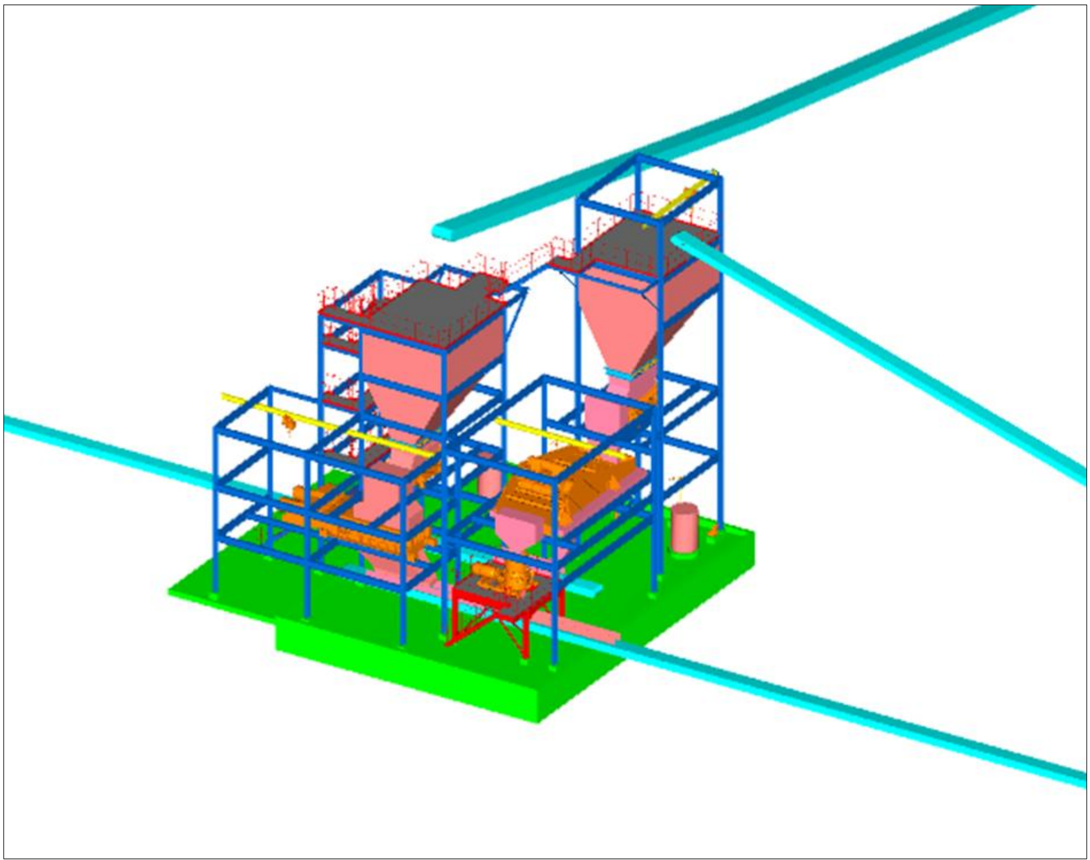

Figure 17-4: Crushers

17.2.2.4. Tertiary crushing

PFD Reference: C01023P120001001

Anticipating a reduction ratio of 3:1 within the tertiary crusher, aiming for a p80 of 25 mm, the tertiary crushing circuit will function as a closed circuit. The crushed ore generated by the secondary mineral sizer is directed into a singular feed bin (C0102-BN-001) to supply the tertiary cone crusher (C0102-CR-001). Subsequently, the ore is fed from the base of the tertiary feed bin to a vibrating screen (C0102-SC-001) via a vibrating feeder (C0102-FD-001). To mitigate over-grinding, the undersized material from the vibrating screen is directed straight to the product conveyor (C0102-CV-001). The oversize from the vibrating screen is channelled into a cone crusher (C0102-CR-001), and the resultant product from the cone crusher is conveyed back to the feed bin (C0102-BN-001) via two transfer conveyors (C0102-CV-004/005) and transfer tower (C0102-TT-001).

Process water is introduced into the dust suppression water tank (C0102-TK-001). Following filtration through a water filter (C0102-FL-001), the water is disseminated to ore transfer points utilizing the dust suppression pump (C0102-PP-001). The water sprays from the dust suppression system maintain the moisture content preventing the generation of excessive dust around the bins.

The feed bins are situated within a confined area that incorporates a slurry collection sump (C0102-SM-001) and a spillage pump (C0102-SP-001). Fine spillage is directed into the sump and subsequently pumped to the crushed ore stockpile (C0102-SK-001), while mechanical equipment manages and reloads bulk spillage back onto the plant feed stockpile.

17.2.2.5. Agglomeration

PFD Reference: C01033P120001001

Crushed ore from the stockpile is directed to the agglomerator (C0103-AD-001). The ore is introduced into a feed bin (C0103-BN-001), equipped with a base-mounted belt feeder (C0103-FD-003), through which the crushed ore is fed into the agglomerator. Simultaneously, concentrated sulfuric acid (98 %) is injected into the agglomerator, leading to the formation of an agglomerated product. Given the exothermic nature of the reaction within the agglomerator, the resulting product is initially conveyed onto a sacrificial conveyor (C0103-CV-003) to facilitate controlled cooling before being transferred to the heap leach feed conveyor (C013-CV-004).

The agglomerator is situated within a bunded area, incorporating a slurry collection sump (C0103-SM-002) and a spillage pump (C0103-SP-002). Fine spillage is directed into the sump and subsequently pumped back into the agglomerator feed chute (C0103-CH-002). In the event of acid spillages within the area, the common spillage pump (C0103-SP-002) directs them to the emergency pond (C0105-PD-005).

17.2.3.Basis of design

Design of the ore preparation stage was done in combination with the processing plant discussed in Section 17.4, and the mechanical, piping, civil, electrical, and C&I bases of design are presented in detail in Section17.4.3 to Section 17.4.7.

17.3. Heap leach pad design and operation

17.3.1.Introduction

Agglomerated material from the ore preparation stage will be delivered to a dynamic heap leach facility (“DHLF”) located between the two largest pits, Dibbwi East and Muntanga. Leached and rinsed ore will be moved from the DHLF to an engineered spent ore dump (“SOD”).

This report section details the design of the DHLF, associated solution corridor, solution ponds, gypsum ponds, and the SOD. This section should be read in connection with Figure 17-1 and Figure 17-5.

17.3.1.1. Scope of work

This section provides a feasibility level engineering design for the Muntanga Project DHLF. For this report, feasibility level design is defined as providing sufficient detail to allow for a ±15 % capital cost estimate.

The SOW for this report includes:

- Field investigation of the DHLF and SOD foundations and ponds including the gypsum pond, consisting of test pits, boreholes, sampling, and material testing
- Feasibility level design of the Muntanga DHLF, which includes a HLP, associated solution corridor, solution ponds, gypsum ponds, and the SOD
- Design of stormwater diversion channels
- Water balances using average monthly climate data
- Stability analysis
- Drawings for cost estimation purposes
- Quantity estimates and outline construction costs and
- Recommendations are presented at the end of the report to increase the level of cost accuracy.

17.3.1.2. Ground investigation

SRK ZA was appointed by SRK UK to undertake the geotechnical engineering investigations for the proposed infrastructure components (listed below) at the Dibbwi East and Muntanga, Dibbwi, Gwabi and Njame sites.

- Heap leach facilities (“HLF”)
- Processing plant
- Access road and drainage crossings (Dibbwi East and Muntanga site only, addressed in a separate report SRK Report No. 593297/3)
- Potential construction material (addressed in a separate report SRK Report No. 593297/4).

Fifty test pits (“TPs”) were excavated in total, with forty-two excavated in the Dibbwi East & Muntanga site and given the “DM” prefix (DMTP), and eight excavated in the Muntanga area and given the “MT” prefix (“MTTP”). The TPs were excavated to depths ranging from 0.45 m to 3.95 m below surface.

Fifteen PQ3-size rotary core DHs were drilled across the Dibbwi East and Muntanga sites, each to a target depth of 25 m. The TP and DH positions are shown in Table 18-5.

A fixed length dynamic cone penetrometer of 0.90 m was used to conduct testing at each TP. Where refusal was not encountered in the upper 0.90 m, the test was continued from within the TP at a depth of 0.90 m to 1.80 m or shallower refusal. Dynamic cone penetrometer blow values from a depth of 0.90 m to 1.20 m have been removed as this interval is considered disturbed ground like the seating blows for a SPT test.

A total of 78 SPT and 44 DPSH tests were conducted within the DHs at 1.5 m depth intervals.

Disturbed soil samples were collected from all TPs and DHs and representative samples were packaged and submitted to a soil and rock testing laboratory, respectively, for testing.

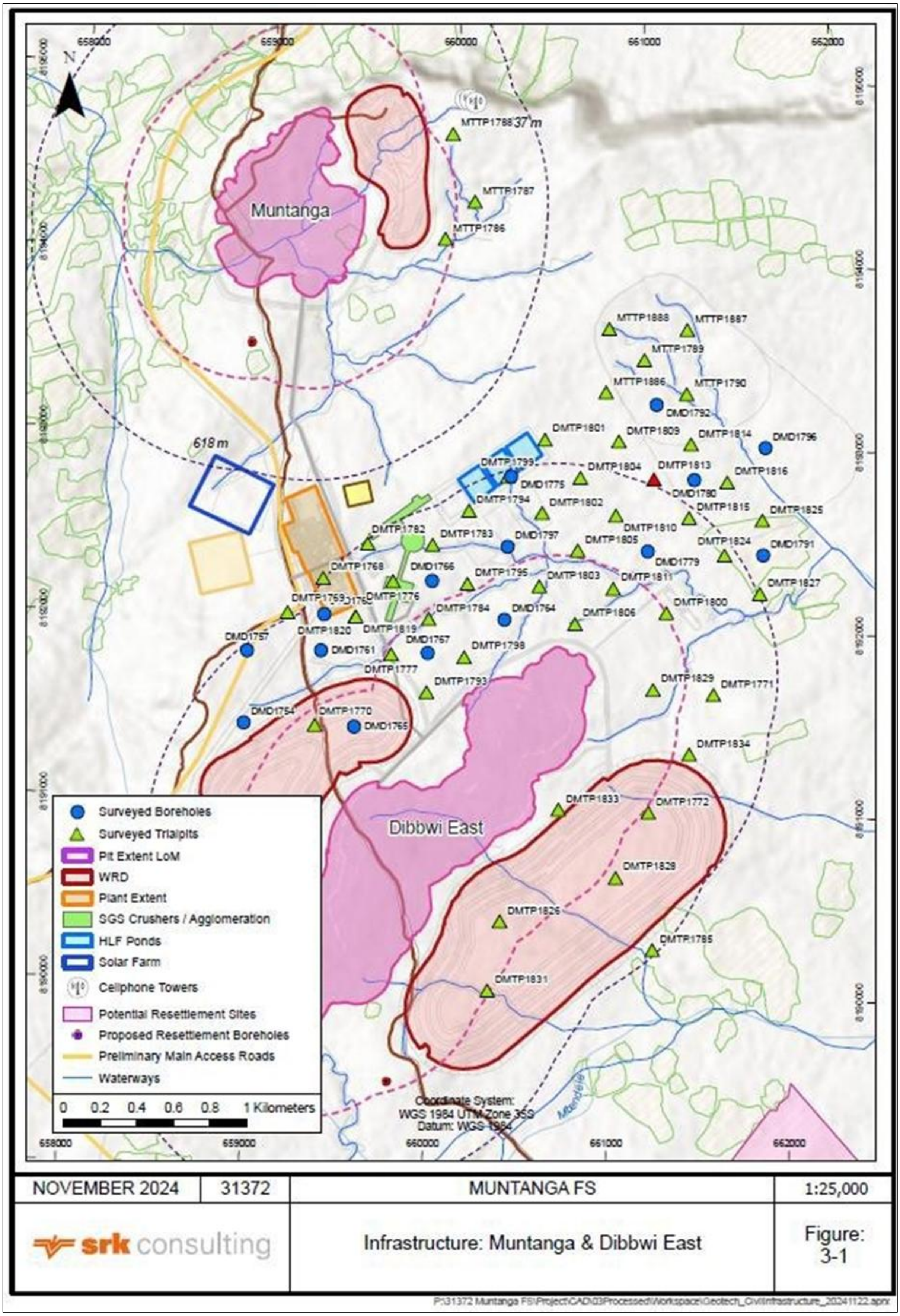

Figure 17-5: Trial pit and drill hole plan

17.3.1.3. Ground profile and foundation conditions

The investigation identified four main lithological units on the near-surface that will form the foundations for DHLF and SOD. Their nature and occurrence are as follows:

1. Topsoil
   - The ground profile is generally characterised by a surface layer of silty, fine- to medium-grained sand with roots to a maximum depth ranging from 0.02 m to 0.50 m below surface
2. Alluvium
   - Topsoil is generally underlain by dense to very dense silty and/or clayey sand, with or without gravel, or silty and/or clayey, sandy gravel with moderate to abundant roots profiled to depths between 0.02 m and 3.75 m, with the presence of roots decreasing with depth
3. Residual soil
   - The topsoil and/or alluvium horizons are occasionally underlain by dense to very dense silty sand or sandy gravel residual sandstone with moderate to trace roots to depths between 0.02 m and 3.20 m. The residual horizon is considered discontinuous in its lateral extent.
4. Bedrock
   - The topsoil and/or alluvium and/or residual soil horizons, in 15 of the 31 TP, are underlain by sandstone, with only two TP, DMTP1866 and DMTP1874 in the Dibbwi East area, intersecting highly to moderately weathered, massive, low to medium strength (similarly soft rock to medium hard rock) mudstone from 0.75 m to 2.40 m and 0.96 m to 1.66 m, respectively
   - Sandstone bedrock was intersected to depths between 0.17 m to 3.66 m and was generally described as highly weathered, massive, low to medium strength (similarly soft rock to medium hard rock)
   - The DH across the site intersected rock at depths typically ranging from 1.15 m to 2.45 m below surface
   - No subsurface seepage of water was noted during the excavation of TP and the water table was not intersected during DH drilling.

17.3.1.4. Geotechnical laboratory testing

17.3.1.4.1. Index testing

A total of 18 samples were submitted for index testing to two laboratories: Bilvin, a Zambian soil testing laboratory and a South African soil testing laboratory, VGLS. The samples consisted of :

- Nine alluvium-disturbed soil samples (Table 17-2)
- Six residual disturbed soil samples (Table 17-3) and
- Three excavated bedrock samples (Table 17-4).

The index testing included:

- Particle size distribution ("PSD")
- Atterberg limits
- Optimum moisture content ("OMC")
- Standard and modified proctor maximum dry density ("MDD")
- California bearing ration ("CBR")
- Swell, and unified soil classification system ("USCS") classification.

From the test work, the following conclusions can be made:

- The USCS classification ranges between SC and SM: alluvium was classified as SM, SP and SC; residual and bedrock as SC to SM
- Based on the grading modulus, it is not expected that the material would be problematic due to reworking; however, it should be noted that the percentage of fines less than 0.15 mm is over 60 % at the upper end, indicating that 'silt behaviour' may occur
- A high percentage of silt leads to moisture-sensitive behaviour, causing poor compaction, an increase in voids, increased settlement and reduced remoulded strength.

It is important to note that the Proctor maximum dry density and OMC, and by extension the saturated CBR, results reported are based on strictly controlled conditions in the soil testing laboratory. Conditions during construction will be more variable, particularly the control of the moisture content.

A total of ten tests were recorded for the UCS with strain gauge results. These are summarised in Table 17-5. The '50 % UCS' ranges between 1.3 MPa and 17.8 MPa, with an average of 3.3 MPa. The results, at 50 % UCS, indicate that the hardness of the sandstone bedrock can best be described as very soft rock to soft rock with a 'layer or zone' of medium hard rock from approximately 5.0 m to 7.0 m below surface.

Table 17-2: Alluvium index testing results for Dibbwi East

TP ID		DMTP1770	DMTP1833	DMTP1824	DMTP1802	DMTP1816	DMTP1802	DMTP1801	DMTP1824	DMTP1825
Depth from (m)		**0.25**	**1.5**	**2.5**	**0.75**	**0.15**	**0.2**	**0.9**	**0.3**	**1.8**
Depth to (m)		**0.8**	**2.3**	**2.7**	**1.75**	**2.4**	**0.75**	**1.75**	**2.5**	**3**
Weighted PI		*0*	*0*	*0*	*5*	*13*	*12*	*1*	*0*	*0*
Weighted LL		*14*	*13*	*22*	*16*	*27*	*23*	*0*	*0*	*0*
Weighted LS		*0*	*0*	*0*	*3*	*6*	*6*	*0*	*0*	*0*
OMC		10.6	10.9	12.3	10.3	13	11.8	7.7	9.8	9
MDD (Std Proctor, MDD)									1 972	1 724
MDD (Mod Proctor, MDD)		1 934	1 990	1 861	1 985	1 828	1 930	2 034		
at CBR (%)	93	5	11	5	6	5	10	9		
	95	10	16	11	12	7	12	11		
	98	17	23	15	18	9	14	15		
	100						14	23		
Swell (100 % MDD)							0	0		
COTO (2020)		<G9	G8	<G9	<G9	<G9	G9	G9		
USCS (ASTM)		SM	SM	SM	SC	SC	SC	SM	SM	SP

Table 17-3: Residual soil index testing results

TP ID		DMTP1795	DMTP1814	DMTP1834	DMTP1795	DMTP1801	DMTP1810
Depth from (m)		0.2	0.3	1.9	1.2	2.45	0.8
Depth to (m)		1.2	0.95	2.8	1.4	3.05	1.5
Origin		Residual sandstone	Residual sandstone	Residual sandstone	Residual sandstone	Residual sandstone	Residual sandstone
Weighted PI		*11*	*10*	*0*	*12*	*4*	*24*
Weighted LL		*24*	*22*	*14*	*24*	*12*	*43*
Weighted LS		*4*	*3*	*0*	*6*	*2*	*12*
OMC		12.5	15.3	12	13.9	10	18.1
MDD (Std Proctor, MDD)					1 821	1 981	1 700
MDD (Mod Proctor, MDD)		1 882	1 842	1 979			
at CBR (%)	93	6	6	4			
	95	9	9	12			
	98	12	14	19			
	100						
Swell (100 % MDD)							
COTO (2020)		<G9	<G9	<G9			
USCS (ASTM)		SC	SC	SM	SC	SC-SM	CL

Table 17-4: Excavated bedrock index testing results

TP ID		DMTP1783	DMTP1820	DMTP1830
Depth from (m)		0.3	0.65	0.35
Depth to (m)		0.8	1.15	0.8
Origin		Excavated sandstone	Excavated sandstone	Excavated sandstone
Weighted PI		*0*	*5*	*8*
Weighted LL		*13*	*17*	*23*
Weighted LS		*0*	*3*	*0*
OMC		10.5	12.1	11
MDD (Std Proctor, MDD)				
MDD (Mod Proctor, MDD)		1 830	1 879	1 873
at CBR (%)	93	3	4	5
	95	6	12	14
	98	10	13	16
COTO (2020)		<G9	<G9	<G9
USCS (ASTM)		SM	SC	SC

Table 17-5: Bedrock UCS, E, v, axial strain and failure type summary

Sample ID	Mid-point depth	Rock type	Density	Strength [UCS]		Tangent elastic modulus @ 50% UCS	Secant elastic modulus @ 50% UCS	Poisson's ratio tangent @ 50% UCS	Poisson's ratio secant @ 50% UCS	Linear axial strain at failure	Failure code
				Failure	50%						
	[m]		[g/cm³]	[MPa]	[MPa]	[GPa]	(GPa)			[mm/mm]	
DMD1754_001	3.66	Sandstone	1.90	8.7	4.3	0.6	0.5	0.23	0.13	0.015619	XB
DMD1754_002	7.90	Sandstone	2.12	6.0	3.0	1.1	0.7	0.40	0.30	0.006970	XB
DMD1757_003	5.41	Sandstone	2.14	27.4	13.7	9.8	6.5	0.44	0.17	0.003650	XB
DMD1760_002	6.40	Sandstone	2.09	35.6	17.8	14.5	16.3	0.14	0.12	0.002916	XA
DMD1761_001	1.28	Sandstone	1.76	5.3	2.6	0.4	0.3	0.24	0.14	0.016768	XB
DMD1766_003	9.69	Sandstone	2.08	8.2	4.1	0.7	0.6	0.31	0.20	0.011378	XB
DMD1767_002	2.69	Sandstone	2.10	10.7	5.4	3.4	1.2	0.59	0.21	0.006097	XB
DMD1779_001	0.70	Sandstone	2.00	6.7	3.3	0.4	0.4	0.21	0.21	0.014892	XB
DMD1779_002	4.26	Sandstone	1.74	4.6	2.3	1.1	0.6	0.47	0.14	0.006543	0B
DMD1792_004	8.55	Sandstone	2.03	2.6	1.3	0.3	0.3	0.37	0.37	0.007870	0B

Notes:

- XA – Single sliding shear
- XB – Complete cone development
- 0B – Multiple discontinuities

17.3.2.Dynamic heap leach facility design

17.3.2.1. Design criteria

The design criteria for the facility are summarised in Table 17-6.

Table 17-6: Dynamic HLF design criteria

Parameter	Design criteria
Total tonnage	3.5 Mtpa
Material size	38mm minus
Maximum stacked ore height	10 m
Nominal angle of repose	2.5H:1V
Average in-place ore dry density	1 600 kg/m^3
24-hour storm event	100 year
Maximum head on geomembrane liner	300 mm
Solution collection methodology	Gravity
Minimum acceptable static factor of safety (“FoS”)	1.3

17.3.2.2. Pad location and layout

The location of the Muntanga DHLF is illustrated in Figure 17-6. The optimum location was determined from several parameters, listed below in order of priority:

- **Pad slope alignment:** The pads were orientated in such a way that the required 1˚ foundation slope aligns with the natural topography as much as possible and the associated solution ponds are downslope. The individual pads have a slight inclination to ensure that any collection fluid reports to a single point for ease of collection. This will aid in drainage of the leachate to the ponds with no requirement to pump uphill. Furthermore, this reduces the earthworks required for the pad foundations and limits the cut-and-fill quantities
- **Proximity to the plant:** The DHLF was located to reduce the transportation distance and cost from the pad to the plant. This reduces the amount of piping and pumping required
- **Standoff distance from the pits**: A radius of 500 m from the edge of the pit slopes was determined and the pads have been located outside of this boundary. This ensures minimal impact from any pit operations such as blasting that may affect stability of the HL infrastructure as well as limiting static and mobile plant interaction
- **Existing infrastructure:** The location of the pads was positioned to avoid any existing roads or planned infrastructure relating to the mine.

Based on the criteria listed above, the preferred location of the DHLF is directly northeast of the plant and crusher with the crest of the pad aligned parallel to the existing ridgeline. The associated ponds are located southeast of the pad at a lower elevation to promote passive drainage.

The DHLF has been designed as a large pad with a 1˚ slope to the southeast and three dividing berms to create four smaller paddocks. Each paddock has a base inclined to promote drainage to a single point. This will allow for continuous leaching as the process cycles through the paddocks. While one paddock is being leached, the others will undergo construction of the ore stack ready for the next leaching cycle, rinsing of the leached material before it is transported to the SOD and offloading of the spent ore to the SOD. The paddocks have a 0.5 m thick drainage gravel layer for solution collection that is intended to remain in situ beneath the ore through the life of the mine. In this drainage layer, there will be pipework arranged in a herringbone formation to collect the solution and transfer it to the drainage point where a small pond will exist. From here, the solution will then be routed to either the pregnant or the barren pond depending on whether the pad is in a leaching or rinse cycle.

A 1 m high safety berm runs around the perimeter of the pad, into which the liner will be tied to mitigate against loss of containment. On the northwestern and northeastern perimeters, a 1 m deep interception trench will be excavated outside the footprint of the pads to catch surface runoff running downhill and divert it to the stormwater ponds which have been designed to contain a 1-in-100-year storm event for seven days. A 2 m deep trench exists within the footprint to run the solution pipework that takes fluid from the barren and rinse ponds to the distribution system.

Table 17-7 provides the parameters that dictate the size requirements of the DHLF.

Table 17-7: Dynamic leach facility parameters

Total annual throughput [tonnes]	Annual throughput per pad [tonnes]	Ore density [kg/m^3]	Leach time [days]	Rinse time [days]	Height of stack [m]
3 500 000	875 000	1 600	25	25	10

The facility sizing was based on the annual ore throughput, geochemical leach cycle, rinse cycle, and stack height. These parameters were derived from leach and rinse column tests discussed in Section 13.14. The pads are slightly oversized to accommodate any changes in the volume of production once mining commences. Finally, the DHLF has been sized to allow sufficient space and standoff for the ore stacking and reclamation systems with their associated conveyor/ stacker systems. These have been oriented to deliver and collect ore in relation to the crusher and SOD, respectively.

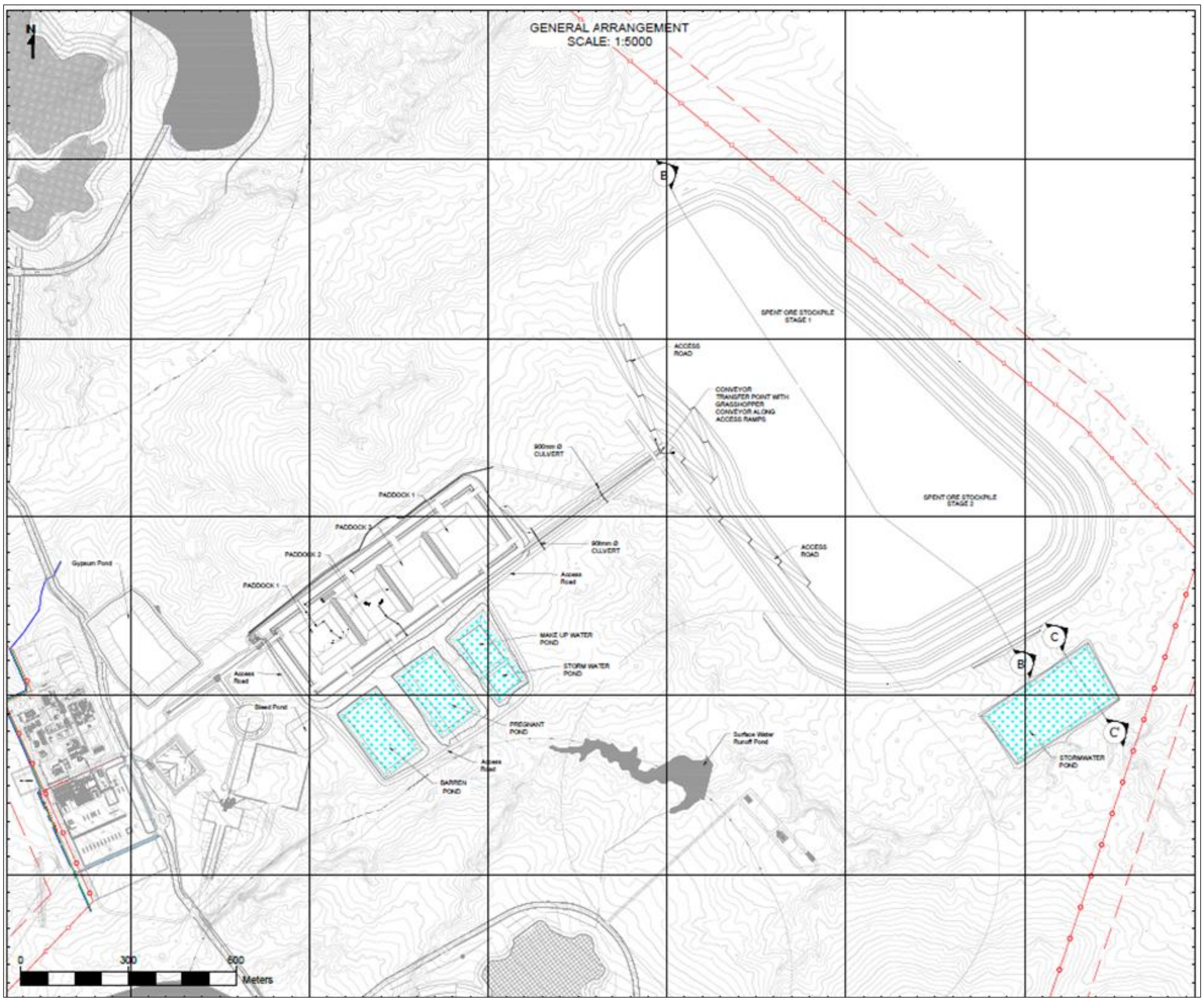

Figure 17-6: General map of DHLF

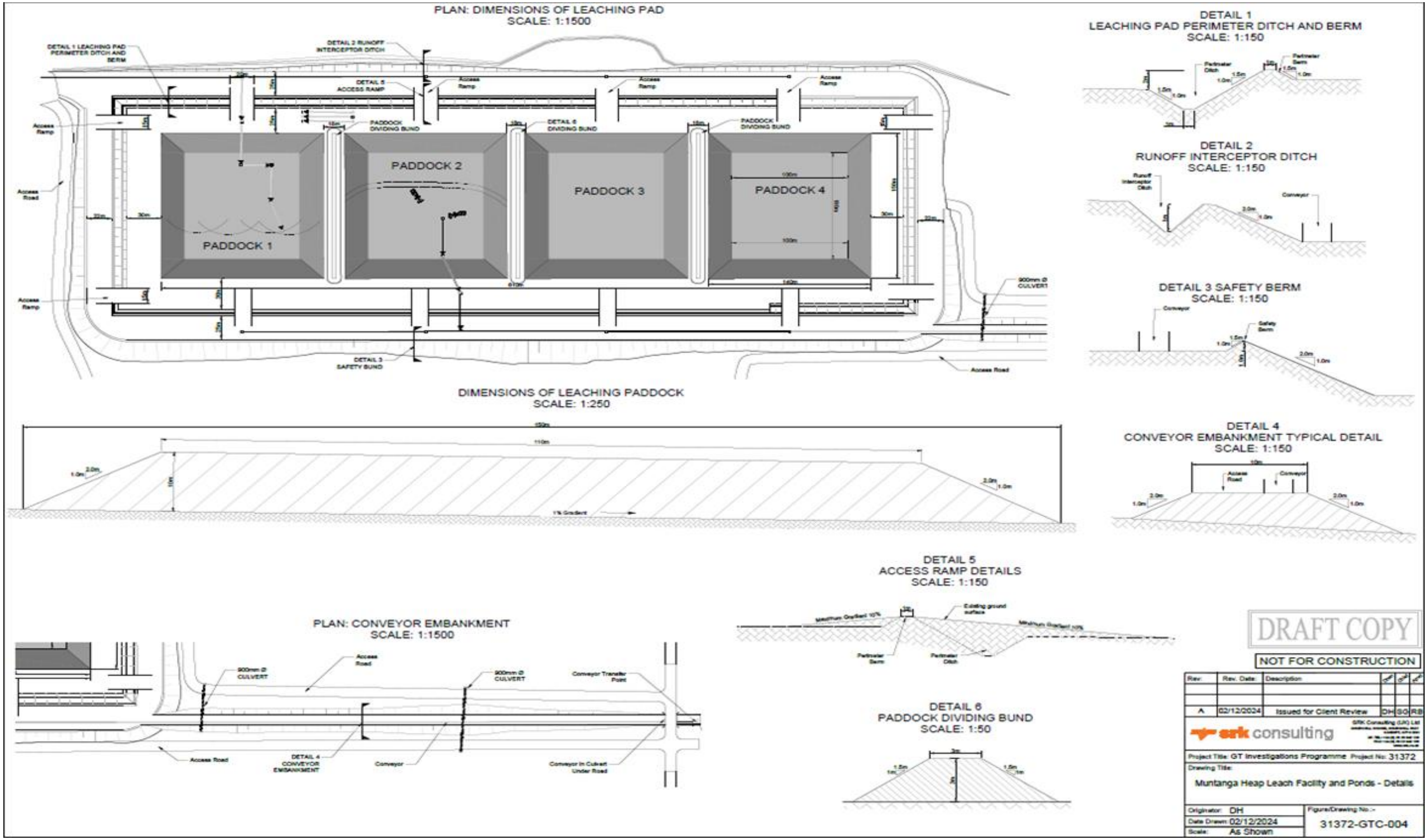

Figure 17-7: GTC004 General paddock arrangement with interception ditches and storm diversion berms

17.3.2.3. Heap leach pad liner system

The liner configuration has been selected to minimise leach solution losses which mainly would be due to defects in the geomembrane manufacturing process or damage from the installation and construction. Two liner configurations were considered:

1. Geosynthetic clay liner with geomembrane membrane overlaying drainage layer overlaying geomembrane (synthetic double liner)
   - This option would include preparing the in-situ surface and placing a geosynthetic clay liner (“GCL”), then two layers of geomembrane separated by a synthetic geodrain layer. This double-liner system offers the advantages of being able to monitor the entire DHLF area for any leakage from the upper geomembrane, and not being reliant on sourcing sufficient natural low-permeability materials. From a cost and schedule perspective, the drainage layer would consist of a synthetic geonet drainage material.
2. Natural clay liner with geomembrane membrane overlaying drainage layer overlaying geomembrane (synthetic double liner)
   - This option would include preparing the in-situ surface and placing natural clay soils with sufficiently low permeability (< $1x10^{-9}$ m/s), then two layers of geomembrane separated by a geodrain layer. This solution is heavily reliant on either sourcing an on-site clay material or importation of materials.

Given the permeability of the natural superficial materials and bedrock and the potency of the liquor, it is not considered appropriate to consider a single-liner system for this site.

Considering the costs and schedule associated with manufacturing a natural clay liner, SRK recommends that a double liner system with GCL is used for the DHLF footprint shown in Figure 17-8 (from bottom to top) comprises:

- Prepared native soil
- Internally reinforced geosynthetic clay liner
- 2 mm single-sided textured high-density polyethylene (“HDPE”) geomembrane
- 4 mm single-layered, 3D geodrain consisting of extruded strands
- 2 mm HDPE geomembrane synthetic liner
- 125 g/m^2 smooth geotextile for protection and
- Gravel drainage layer.

This configuration will minimise seepage losses and allow for leak detection via the geodrain layer.

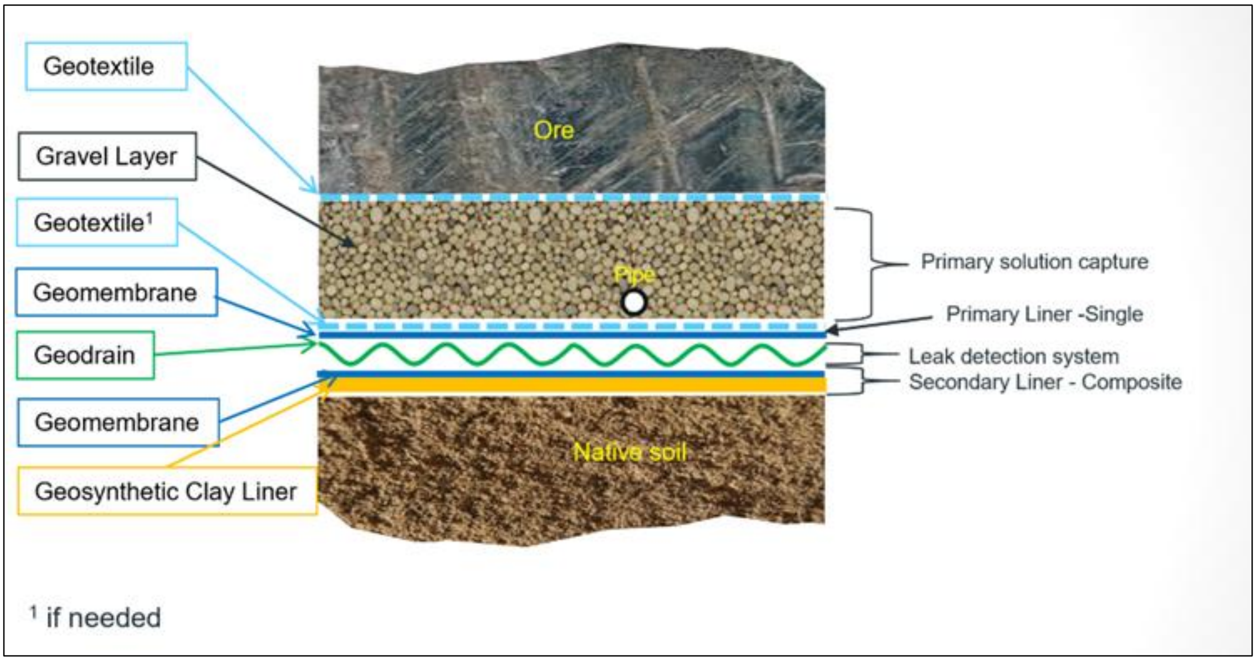

Figure 17-8: Double liner with composite GCL and Geodrain leak detection

17.3.2.4. Heap leach pad solution collection system

This section must be read in conjunction with Figure 17-9 to Figure 17-11. Solutions will be collected in drain ponds that will be situated 4 m from the base of the heap with open channels collecting solutions and feeding the ponds. A 250 mm diameter pipeline will extend from the drain pond to the PLS pond. For rinsing solutions, a separate 250 mm diameter pipeline will feed to the makeup water pond. The ponds will be positioned in the SE corner (pads 1 and 3) or SW corner (pads 2 and 4) of the heaps (GTC-005). The drain ponds will have a capacity of 2 640 m^3. Leach solution application across the heaps will be at a rate of 10 L/h/m^2.

17.3.2.5. Stability analysis

17.3.2.5.1. Methodology

Stability for static and pseudo-static cases was assessed to identify the FoS for the proposed dynamic HLF to ensure the stability of the design. The critical section was chosen based on a surface through the ultimate DHLF configuration, aligned with the maximum pad gradient.

To evaluate the critical safety factor, limit-equilibrium analysis for the proposed heap configuration was used. A geotechnical model was created, and the critical sections were modelled using Slide by Rocscience, Inc. Slide is a two-dimensional ("2D") limit equilibrium slope stability analysis program that evaluates slope stability using various slice methods. Spencer and Morgenstern Price/ GLE method was selected for the analysis as it satisfies both force and moment static equilibrium.

17.3.2.5.2. Configuration and material parameters

The material parameters used in the stability analyses are summarised in Table 17-8. These parameters were based on a combination of completed geotechnical test work, liner manufacturer-provided values, and SRK experience with similar materials. One of the aims of the analysis was to identify the minimum interface shear strength of the HDPE to allow for a FoS of 1.3. This will then inform any future liner selection at detailed design. Analysis has been undertaken for a range of Ru values to establish the influence of varying pore pressure on the DHLF and understand what unacceptable values may be. The liner was included along the base of the SOD as a weak layer. It was predicted that this would likely govern the basal failure surface rather than the foundation strength. The critical section running parallel to the 1˚ slope was adopted for the stability analyses including the berm and trench surrounding the pad perimeter.

Table 17-8: Material parameters for the HLP stability analyses

Material	Unit weight [kN/m^3]	Cohesion [c' in kPa]	Angle of Friction [φ' in degrees]	Ru
Ore	18	0	35	0.1 - 0.3
Waste Rock	20	0	37	0
HDPE	17	To be determined by analysis		0
Foundation	20	1	35	0

17.3.2.5.3. Stability analysis results

Using the material parameters and configuration summarised in Section 17.3.2.5.2, SRK has calculated that to meet a minimum FoS of 1.3 with a Ru value of 0.1, the HDPE interface requires a minimum φ' of 23o or 14o and c' of 10 kPa if there is a small amount of cohesion for the liner. These values should be specified in construction specifications. With increased pore pressure (Ru=0.3) a FoS of 1.03 was achieved. A Ru of 0.3 represents a large head of liquid on the base of the facility and was assessed to check the FoS in a worst-case scenario if the facility stops operating as planned. The base case of peak strength and a Ru of 0.1 is shown in Figure 17-9.

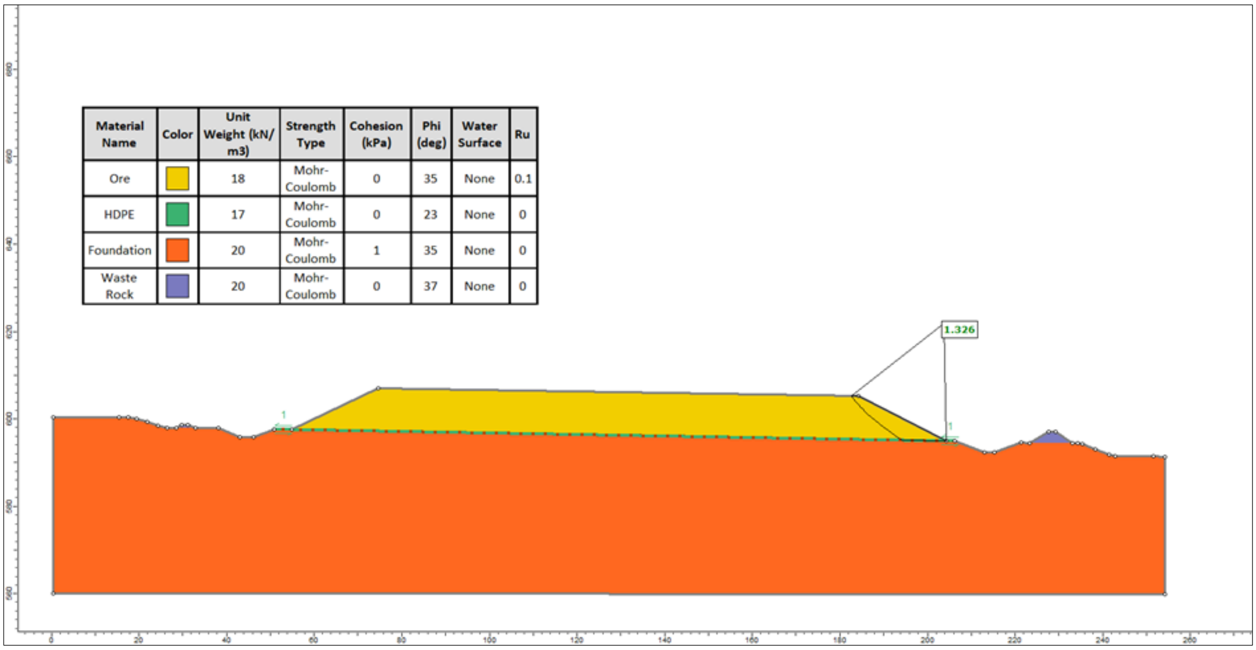

Figure 17-9: Base case HLP stability analysis

17.3.2.6. Pseudo static analysis

There is currently significant uncertainty regarding the seismicity of the site because Lake Kariba nearby induces seismic activity that could pose a risk to the facility. Rather than undertaking deformation analysis using finite element modelling, limit equilibrium analysis has been used to determine the horizontal pseudo-static seismic coefficient at which point failure occurs (the limiting FoS of 1.1 is reached). The critical seismic coefficient is the minimum horizontal load (Ky) required to reach the limiting FoS across all possible failure surfaces.

Pseudo-static analysis was undertaken using peak strength and Ru = 0.1 with the same material parameters applied for the static stability analyses, summarised in Table 17-8.

17.3.2.6.1. Pseudo-static analysis results

Currently, only data from the 1 in 475 annual exceedance probability (“AEP”) earthquake are available, with a peak ground acceleration (“PGA”) of approximately 0.2 g. According to ANCOLD (2019), this is appropriate for the operating basis earthquake (“OBE”), but not the safety evaluation earthquake (“SEE”) where the 1 in 10 000 AEP should be considered. Any future assessments should consider an earthquake magnitude associated with a larger return period, corresponding with the facility classification.

The resulting Ky was calculated at 0.074 g, as shown in Figure 17-10. This is approximately 40 % of the required PGA and therefore would not be considered stable in a 1 in 475 AEP event. However, this is for the worst-case scenario and, as previously stated, further work should be undertaken to establish stability and possible deformations as a consequence of larger magnitude events, as limit equilibrium analysis does not appear appropriate for this facility.

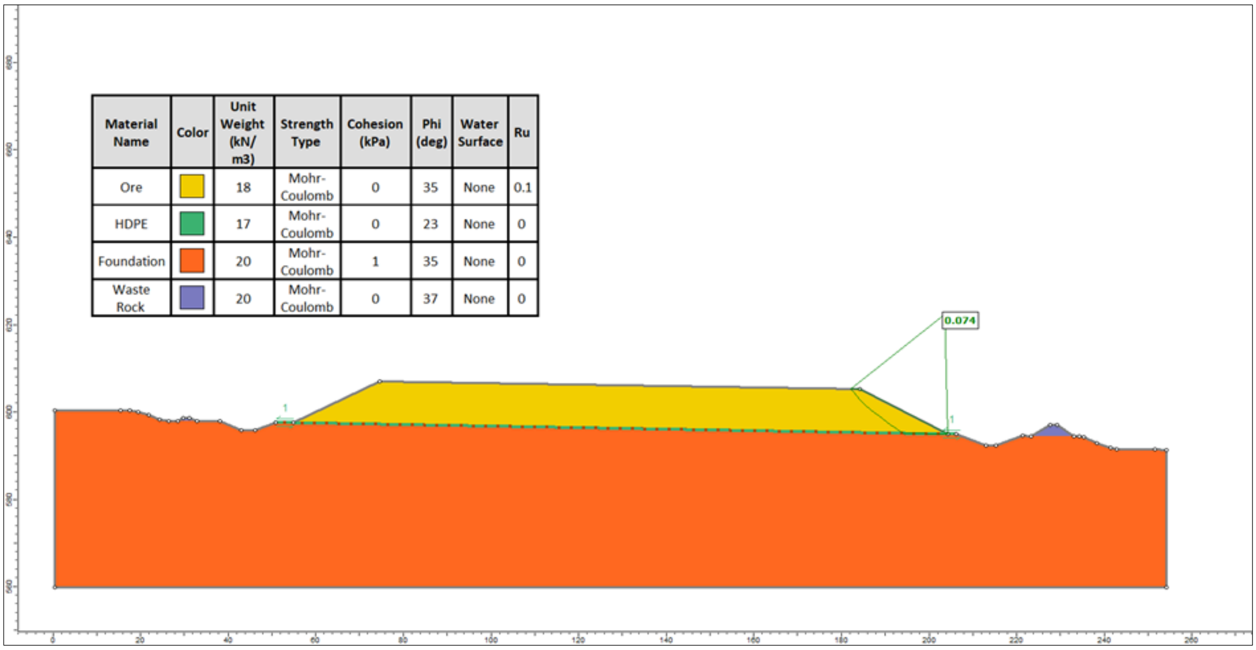

Figure 17-10: DHLF pseudo-static stability analysis

17.3.3.Solution corridor

17.3.3.1. Design criteria

Leaching of uranium from the ore will be achieved using a network of pipes and distribution sprinkler heads to contact the acid solution with the uranium. The porous heaps will allow percolation of solution through the heap to be collected in pipes, laid out in a herringbone format, acting as French drains at the base of the facility that will feed into a collection pond dedicated at each pad. This in turn will feed to a PLS pond and then be sent to the process facility to recover uranium. The major criteria will be the system will be under pressure at various points and will contain acidic solutions.

Solutions will be applied and recovered from the pad at pressures up to 16 bar. Solutions in the main will be highly acidic and thus HDPE piping is essential. Three sizes have been selected based on flows to collect by gravity to the PLS pond:

1. 500 mm diameter HDPE: PE100 PN16, SANS 4427 for conveyance of process fluids to the recovery plant and recovered barren solutions and stormwater runoff
2. 250 mm diameter HDPE: PE100 PN16, SANS 4427 for distribution of solutions on and off the pads
3. 25 mm diameter HDPE: PE100 PN16, SANS 4427 for the sprinkler system on the pads.

17.3.3.2. Solution corridor piping system

A conceptual piping system for the DHLF and transfer to the recovery plant is shown in Figure 17-11 and Figure 17-12. The different coloured lines represent solutions of different chemistries: blue - transfer of PLS; red - barren solution from the plant to be used to leach the ore; pink - fresh water to rinse the pads post leaching; and green - stormwater that will be moved away from the facilities and used as makeup.

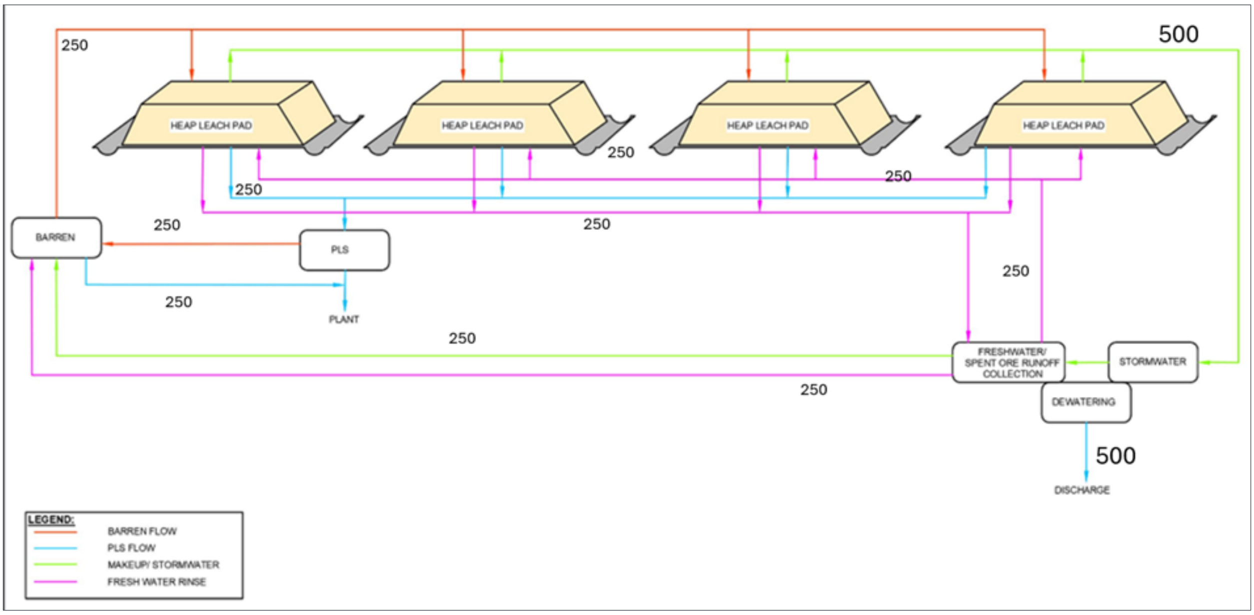

Figure 17-11: Layout of water flows for the DHLF

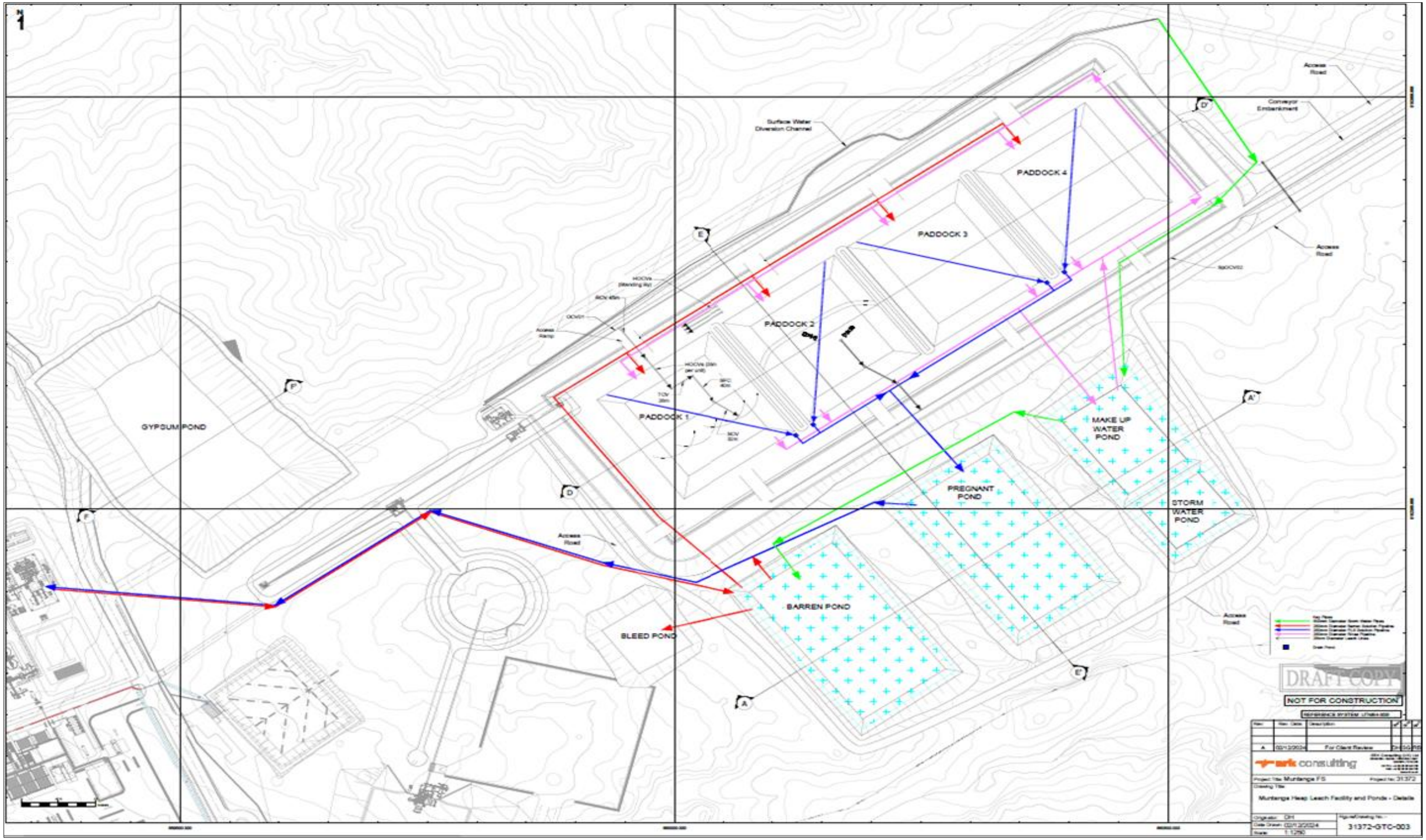

Figure 17-12: GTC003 general piping diagram

17.3.4.Solution ponds

The solution pond system made up of the barren, PLS, combined makeup and rinse, and stormwater ponds, was developed from the following mass balance system. An auxiliary bleed pond is included adjacent to the barren pond to allow for any overflow of barren solution. The barren solution is pumped from the barren pond onto the ore placed on one of the four pads, where it leaches through the ore material. The solution collected from the DHLF drains by gravity and is then diverted either to the barren pond to be recycled on the heap leach or to the PLS pond if the solution grade is high enough. From the PLS pond, the solution is pumped to the plant and treated, following which it is returned to the barren pond and the process is repeated.

Rinse water will be pumped to the makeup pond and then to the barren pond where hydrogen peroxide is added for use on the pad under leach. Stormwater run-off will be diverted to the stormwater pond, which is an extension of the makeup and rinse pond. The gypsum pond is part of a different system to the one described here.

17.3.4.1. Water balance model

The ponds associated with the DHLF have been designed and sized per the design criteria, including the PLS, barren, and makeup/ stormwater ponds, with the rinse pond modelled as a component of the stormwater dam. All ponds are located downstream of the DHLF to allow drainage to be gravity-driven.

The water balance was used to calculate the required volume for the barren, PLS, rinse, and stormwater ponds. Both the PLS and barren ponds were sized to provide seven days of storage based on the leach rate of the plant and DHLF, whereas the rinse pond was sized for three days of storage. The required storage in the bleed pond located west of the PLS and barren ponds was estimated at 20 % of the barren pond capacity. The stormwater dam was sized for a 1 000-year, 24-hour storm event plus operation volume and will remain dry unless excess flows are directed to the pond.

The water flow directions in the vicinity of the DHLP are outlined in Table 17-9. The water balance for the DHLF for dry conditions is presented in Figure 17-13 and the wet conditions are presented in Figure 17-14.

Table 17-9: Muntanga flow volumes to DHLP

Irrigation rate	10	$L/h/m^2$
Area of pad	10 800	m^2
Volume per day (Irrigation)	108 000	L/h
Volume per day (Irrigation)	108	m^3/h
Volume per day (Irrigation)	2 592	m^3/d
Volume per tonne	0.254	m^3/t
Testwork	0.343	m^3/t
Volume per day (Irrigation)	3 499	m^3/d
Use the test work volume per tonne		

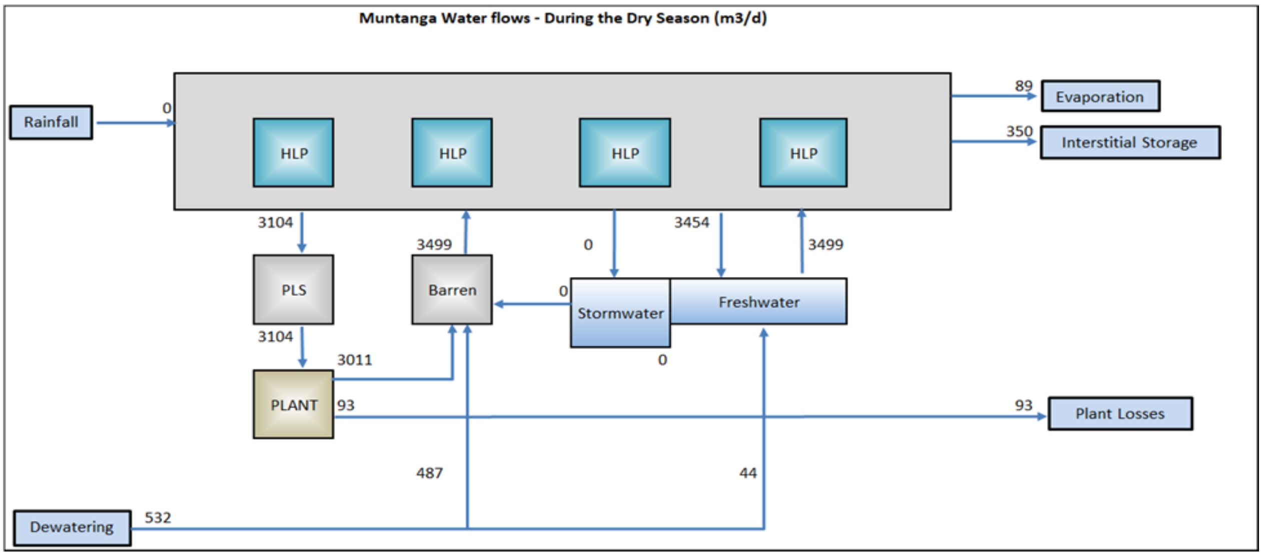

Figure 17-13: DHLF flows for Muntanga HLP under dry conditions

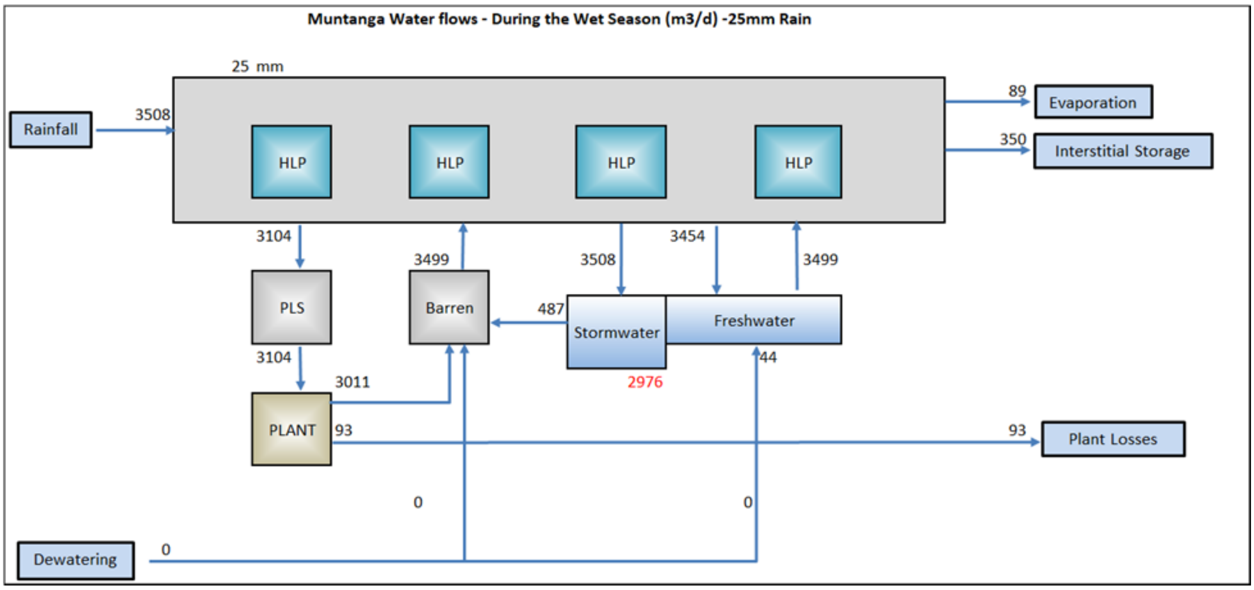

Figure 17-14: DHLF flows for Muntanga HLP under wet conditions (red text indicates change in storage in the stormwater dam)

17.3.4.2. Pond liner system

Two liner systems have been designed based on whether the pond is designed to store stormwater or process solution:

1. Ponds that will regularly contain solution and create a pressure head on the liner system, a double liner system will be installed
2. Ponds that will store stormwater or infrequently store process solution, a single liner system will be installed.

17.3.4.2.1. Barren, pregnant leach solution and bleed ponds

Both the barren and the PLS ponds will be used to regularly store solution, as will the bleed pond, producing a head on the liner. A double liner system has been designed by SRK, as has previously been discussed in Section 17.3 for the HLP, consisting of:

- Prepared subgrade
- Internally reinforced geosynthetic clay liner
- 2 mm single-sided textured HDPE geomembrane
- 125 g/m^2 smooth geotextile for protection
- 4 mm single-layered, three-dimensional geodrain consisting of extruded strands
- 2 mm HDPE geomembrane synthetic liner and
- 125 g/m^2 smooth geotextile for protection.

The in-situ surface will be properly prepared and a GCL placed, and then two layers of geomembrane separated by a synthetic geodrain layer. This dual-liner system provides the benefit of monitoring the entire DHLF area for any leaks from the upper geomembrane while eliminating the need to source adequate natural materials.

17.3.4.2.2. Makeup and stormwater

Based on storage requirements, it has been determined that the rinse and stormwater pond will be implemented with a single liner system, consisting of:

- Prepared subgrade
- Internally reinforced geosynthetic clay liner
- 2 mm single-sided textured HDPE geomembrane and
- 125 g/m^2 smooth geotextile for protection.

During operation, measures to reduce loss of solution through evaporation will need to be taken. This could include the use of ‘shade balls’, which are usually black polyethylene balls floated across the surface of ponds that prevent sunlight from evaporating the solution away.

17.3.4.2.3. Gypsum

The gypsum pond will be used to store excess gypsum from the plant The gypsum will be pumped as slurry material from the plant to the pond. It has been modelled as an excavated pond with a storage volume of approximately 150 000 m^3 with 3H:1V slopes and a 3 m high berm. Due to the nature of the material planned to be stored within the gypsum pond, a liner system is required. As with the other ponds, SRK has developed the liner design with the following aspects:

- Excavated and prepared subgrade
- Internally reinforced geosynthetic clay liner
- 2 mm single-sided textured HDPE geomembrane and
- 125 g/m^2 smooth geotextile for protection.

Gypsum is a hydrophobic substance that does not readily give up water, making obsolete conventional dewatering techniques, such as pond thickening. As such, the pond has been 'oversized' so that the thickness of the gypsum solution is minimised, and water will evaporate by the net negative atmospheric water balance. Given that the volume of gypsum is approximately 10 000 tpa, this should allow sufficient dewatering. During operation, it may become necessary to revise the gypsum management system because the pond is filling too quickly causing an excess thickness of gypsum or the evaporation is not sufficient. This may include measures such as dredging gypsum once dried and placing it in a separate lined facility.

17.3.5.Spent ore dump

17.3.5.1. Design criteria

The design criteria for the SOD are summarised in Table 17-10.

Table 17-10: SOD design criteria

Parameter	Design criteria
Total tonnage	54.2 Mt
Material size	38 mm minus
Maximum stacked ore height	60 m
Nominal angle of repose	3H:1V
Average in-place ore dry density	1 600 kg/m^3
Minimum acceptable static FoS	1.35
Minimum acceptable pseudo-static FoS	1.1

The SOD is to be developed in two stages, the size of which is governed by the rate of rise. The purpose of developing the facility in two stages is to minimise the potential influence of surface runoff on its stability and to spread the capital investment required before the commencement of mining. This is reflected in the following sections and discussed further in Section 17.3.7.5 which focuses on construction.

17.3.5.2. Spent ore dump location and layout

Following leaching, the spent ore must be removed from the DHLF and safely stored in another facility. The location and layout of the SOD is illustrated in Figure 17-15 and has been optimised based on several parameters:

- **Proximity to DHLF:** The distance of the SOD to the DHLF was considered both in terms of maintaining an appropriate standoff distance in case of failure and minimising the transport distance as much as is reasonable. A minimum distance of 250 m between the pad and the SOD has been maintained when considering location to ensure runout from a dump failure would not impact site personnel or the DHLF infrastructure
- **Distance from pit:** As with the DHLF, the SOD was located such that it did not extend closer than a 500 m radius from the pit slopes. This will ensure minimal impact from any blasting in pit that may affect stability of the heap leach infrastructure
- **Site boundary:** The SOD location was considered within the pre-determined site boundary
- **Watershed**: Due to the drainage below the SOD and the fluid collection required, the aim was to restrain the dump to a single watershed and limit the required collection channels and earthworks to ensure no uncontrolled seepage to the environment or inundation of the foundation materials that could create stability issues.

Using the criteria listed above, the optimum location of the SOD was determined to be in the valley east of the DHLF, approximately 500 m from the pad.

The SOD has been designed in such a way that the footprint was constrained to a single water catchment that requires management measures. The aim was to ensure all water from the SOD drains and is collected from the same location, reducing the risk of contamination of groundwater, aiding water management, and minimising the risk of water build-up at the base of the dump which could lead to stability issues. A flow map was used to provide the boundaries of the catchment (Figure 17-15). Whilst the distance from the DHLF exceeds the minimum standoff distance, thereby adding to the transport distance, the additional cost is considered minimal compared to the water management work that would be required if the dump spanned two catchment valleys.

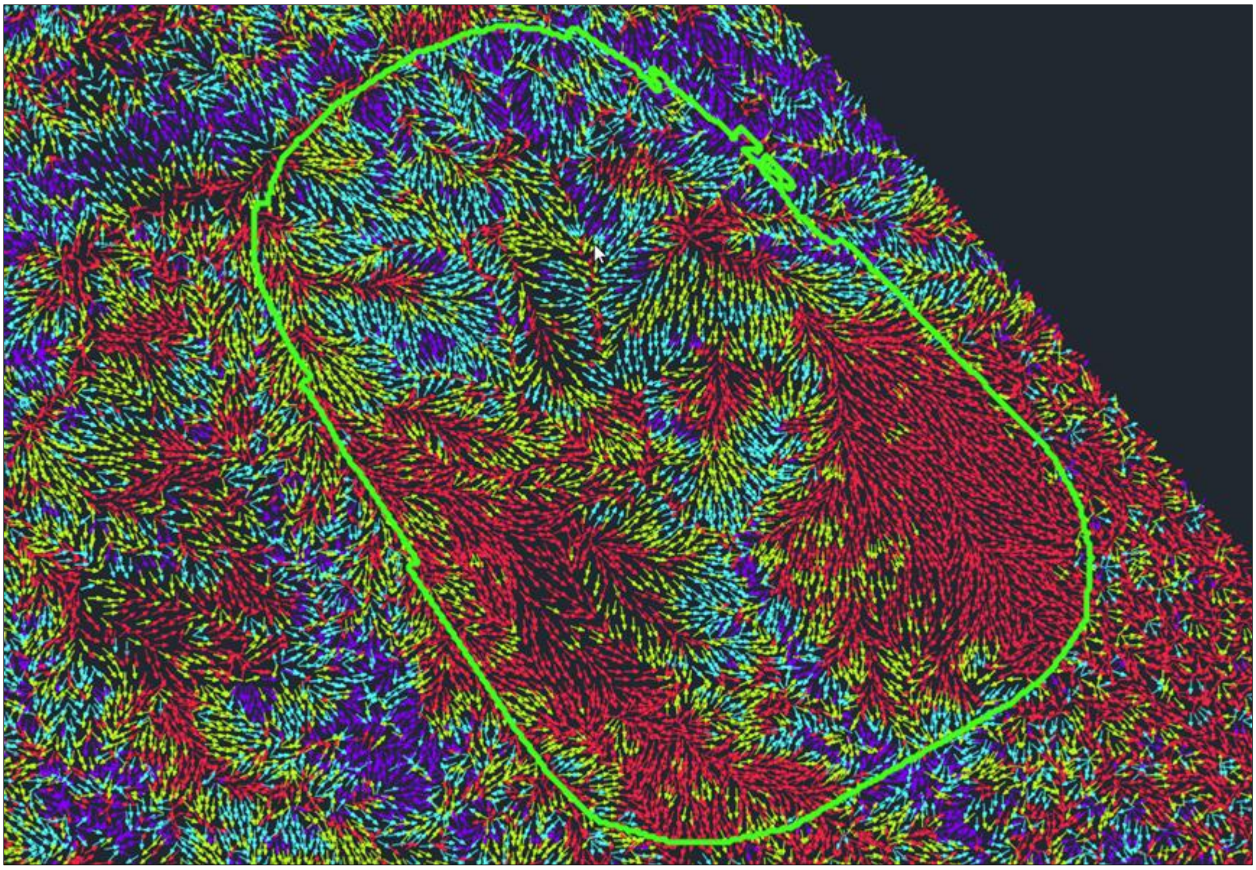

Figure 17-15: Map indicating direction of flow and associated catchment boundary (arrows depict direction and intensity of anticipated flow ranging from blue to red, where red is the most significant flow)

Two stages have been designed for the SOD, with the northern section being constructed first (Figure 17-16, SOD) to provide approximately 8 years of storage. The dump will then be expanded to the southern section for the remaining LOM. This staged approach will offset costs associated with foundation preparation and starter embankment construction for the SOD. A cross-section through the SOD illustrating the two stages of construction is presented in Figure 17-16.

A single stormwater collection pond associated with the SOD has been designed approximately 100 m downstream of the furthest extent. A drainage channel will be constructed to convey water from the first stage of the SOD to the stormwater collection pond, avoiding the requirement to construct two stormwater collection ponds for the different construction stages. A 0.5 m thick layer of waste rock has been included at the base to allow for drainage and dissipation of pore pressures around the foundations of the SOD.

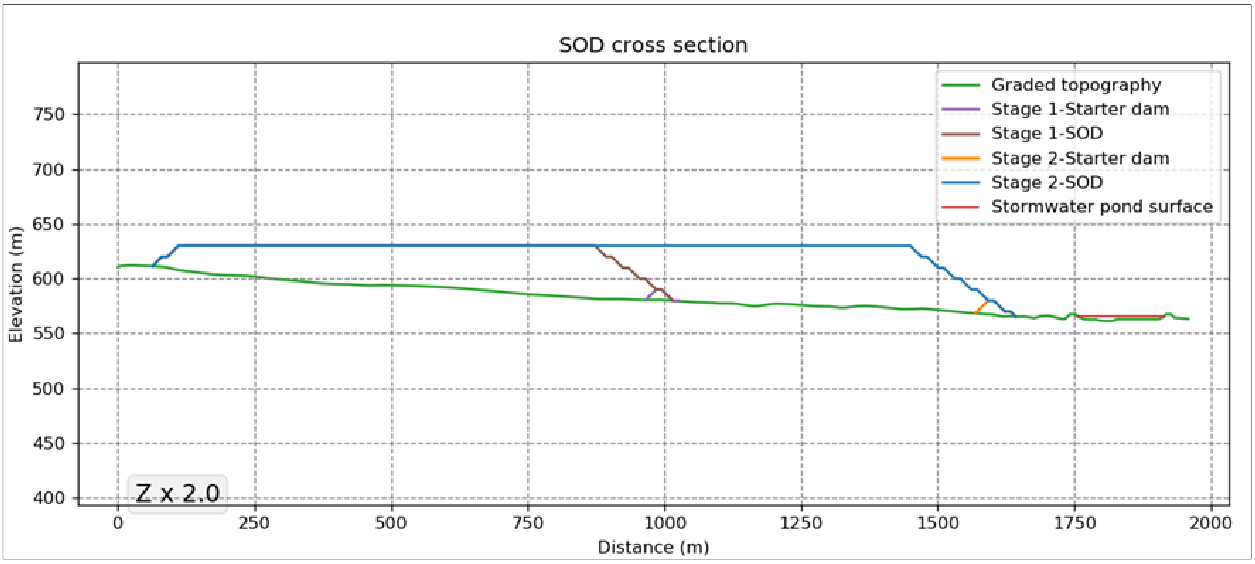

Figure 17-16: SOD cross-section

17.3.5.3. Stability analysis

17.3.5.3.1. Methodology

The static FoS for the ultimate, post-mining SOD configuration was assessed to ensure stability of the proposed design. The critical section was chosen based on the ultimate SOD configuration and aligned with the steepest gradient.

To evaluate the critical FoS, limit-equilibrium analysis was used. A geotechnical model was created, and the Slide 5.030 software from Rocscience, Inc. was used to simulate the critical sections. Slide is a two-dimensional limit equilibrium slope stability analysis tool that evaluates slope stability using various slice methods.

17.3.5.3.2. Configuration and material strength

The material parameters used in the stability analyses are summarised in Table 17-11. These parameters were based on a combination of completed test work as well as SRK's experience with similar materials for the spent ore materials. The critical section was selected trending northwest-southeast down the valley, intersecting the highest point of the SOD.

Table 17-11: Material parameters for SOD stability analyses

Material	Unit weight [kN/m^3] [t]	Cohesion [c' in kPa]	Angle of Friction [φ' in degrees]	Ru
Ore	18	0	43.5	0.1-0.3
Waste rock	20	0	37	0
Foundation	20	1	35	0

17.3.5.3.3. Stability analysis results

Using the material parameters and configuration summarised in Section 17.3.5.3.2, SRK has calculated a minimum FoS of more than 2.0 under normal operation conditions with a Ru of 0.1. An increase in pore pressures (Ru=0.3) gives a FoS of 1.98. An increased Ru of 0.3 represents a worst-case scenario with a build-up of the head at the base of the facility that is only likely to occur if the drainage layer is not working as designed The base case of maximum strength and a Ru of 0.1 is shown in Figure 17-17.

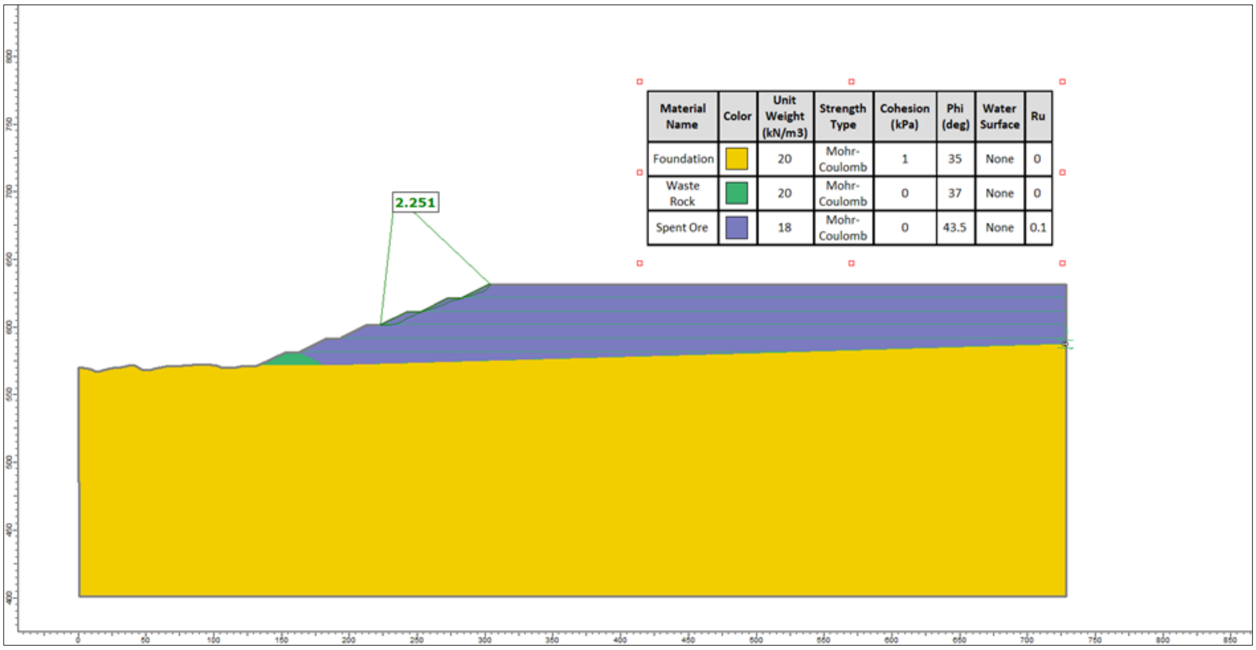

Figure 17-17: Base case SOD stability analysis

17.3.5.4. Pseudo-static analysis

As discussed in Section 17.3.7, limited seismic information is available currently for the site, hence the critical seismic coefficient has been calculated, as opposed to detailed deformation analysis. Any further study should consider a more detailed seismic assessment. The pseudo-static analysis was undertaken using the same material parameters as summarised in Table 17-11.

17.3.5.4.1. Pseudo-static analysis results

The resulting Ky was calculated ranging between 0.18 – 0.29 with Ru values between 0.1 - 0.3, as shown in Figure 17-18. This is more than half the PGA and hence would be considered stable for the 1 in 475 AEP event; however, as previously stated, further work should be undertaken to establish stability and possible deformations as a consequence of larger magnitude events.

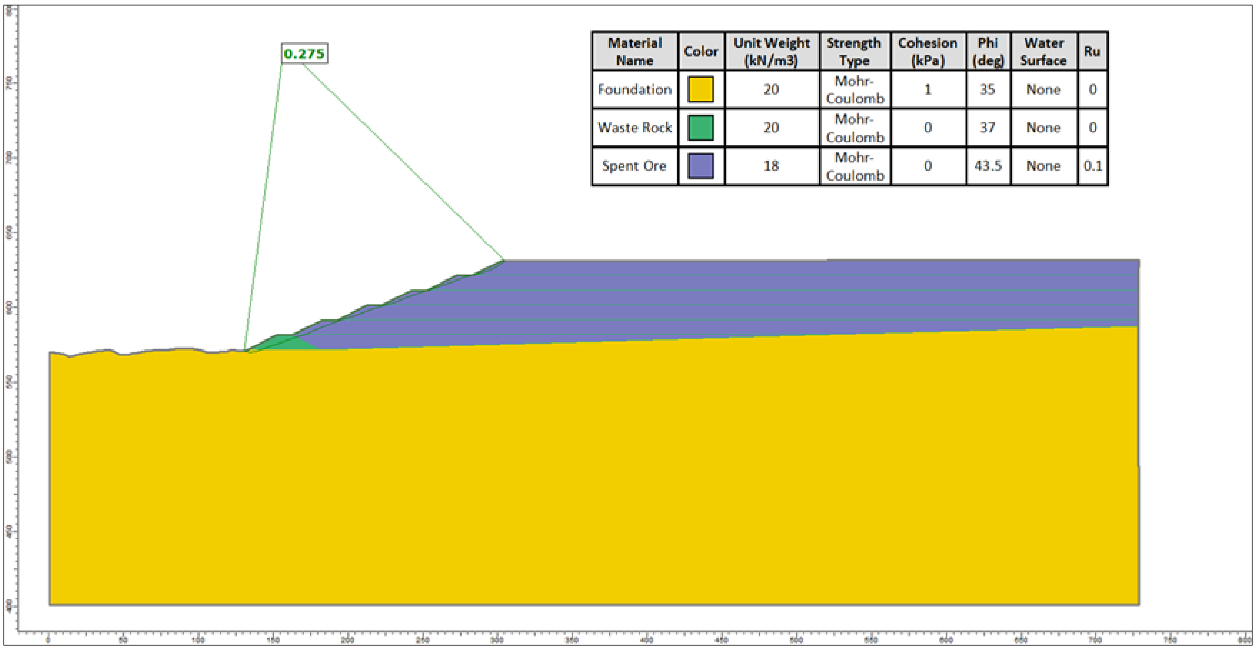

Figure 17-18: Pseudo static stability analysis of SOD with Ru 0.1

17.3.5.5. Water management

When considering the SOD location, surface water diversion was considered, the SOD is located at the top of the catchment area, hence minimal clean water diversion will be required. Drainage of water at the base of the SOD should be carefully considered. Earthworks should be undertaken across the footprint to ensure all water flow is directed properly downstream and there is no pore pressure buildup. This should include the installation of a 0.5 m thick waste rock drainage layer at the base.

The contact water will drain into the SOD stormwater pond located at the base of the SOD which has been sized according to the 1 000-year, 24-hour storm event.

17.3.6. Dynamic heap leach facility materials handling

17.3.6.1. Strategy

Muntanga is a heap leach operation utilising a dynamic “on-off” leach pad system for leaching ore with a separate storage area for spent ore, the SOD (Figure 17-19). The handling of material is a critical component of this operation and comprises:

- Crushing, agglomeration and delivery of agglomerated ore to the conveying transfer tower at the northwest corner of the “on-off” HLP area
- Conveying of agglomerated ore and stacking on the designated pad using a conveyor-stacking system comprising mobile grasshopper conveyors and radial stacker (termed “material handling system 1” or “MHS1”)
- The system will stack agglomerated ore to the correct height in a retreat stacking operation.
- The stacked ore will be leached and rinsed (as described in the relevant section of the FS report) before reclaiming operations can begin
- The spent ore will be reclaimed by a front-end loader (“FEL”) feeding to a grasshopper conveyor system linking to an overland conveyor for transport of spent ore to the SOD (termed “material handling system 2A” or “MHS2A”)
- At the SOD, a conveyor-stacking system comprising mobile grasshopper conveyors and a radial stacker will be used to deposit the spent ore (termed “material handling system 2B” or “MHS2B”). Deposition of the spent ore will be assisted by a dozer operation.

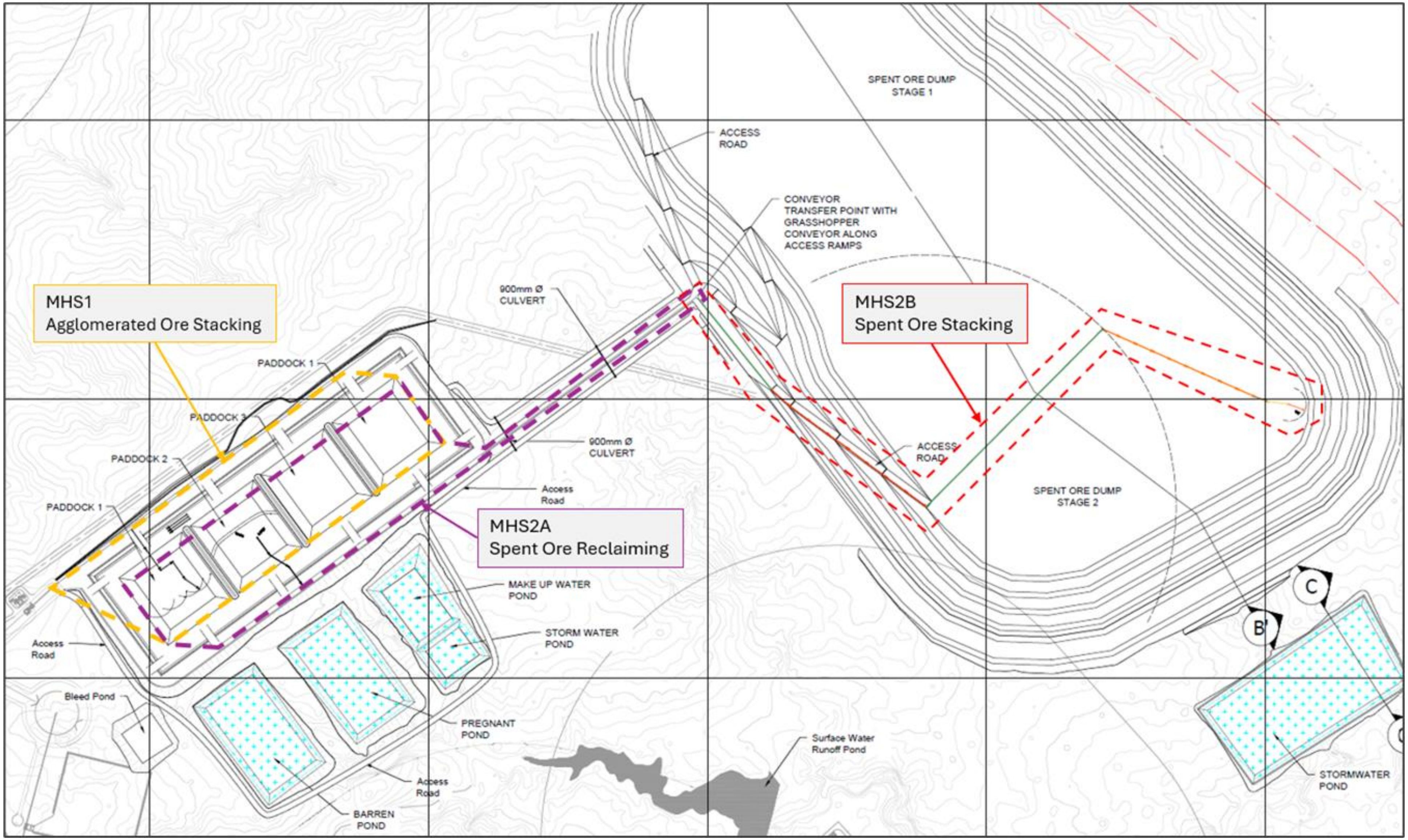

Figure 17-19: Layout showing the proposed stacking and reclaiming systems

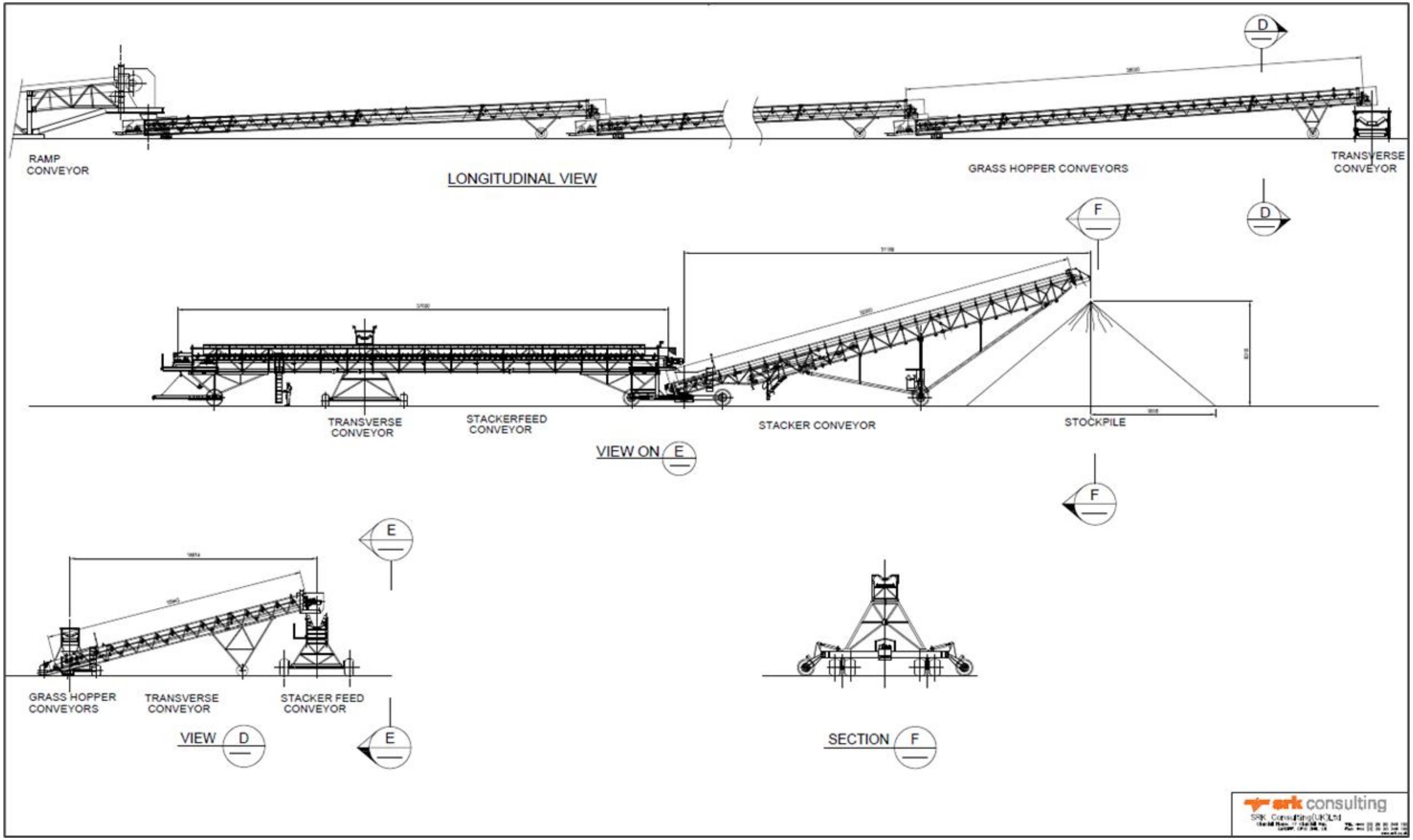

Figure 17-20: Proposed stacking equipment

17.3.6.2. Basis of design

17.3.6.2.1. Dynamic pads and equipment selection

A HL area with four separate paddocks was preferred to the circular "racecourse" style of dynamic pad to minimise pad area and capital costs. There was a preference for a stacking system comprising grasshopper conveyors and radial stackers over other options, such as tripper stacker and bucket wheel excavator reclaim arrangement due to the higher capital cost for the latter arrangement.

17.3.6.2.2. Design inputs

The basis of design inputs for this study is detailed in Table 17-12.

Table 17-12: Basis of design inputs for study

Item	Units	Value	Source / Notes
Battery limit with front end of plant	Name	See Drawing	SGS / FS team
Bulk density – Agglomerated ore	t/m^3	1.6	SGS
Angle of repose – Agglomerated ore	Degrees	38	SGS
Moisture in H/L feed – Agglomerated ore	MC %	10-12 %	SGS
Feed size (F80) – Agglomerated ore	mm	25 mm	SGS
Tonnes per year	Mtpa	3.5	GoviEx
Utilisation	%	75	SGS
Tonnes per hour delivered at battery limit	tph	532	SGS / GoviEx
Tonnes per pad	t	234,000	SGS / GoviEx
Number of pads	#	4	SGS / GoviEx
Leaching time	Days	25	SGS / GoviEx
Stacking height	m	10.1 m	SGS / GoviEx
Grid cost (per kWh)	USD/kWh	0.125	FS power study
Diesel (delivered to site) for FS	USD/L	0.97	GoviEx

17.3.6.3. Materials handling system 1: Agglomerated ore stacking

An illustration of the sequence for agglomerated ore stacking is shown in Figure 17-20 and described below.

17.3.6.3.1. Philosophy

- Four "side-by-side" leaching cells (or "leaching pads") will be developed, and leaching operations will rotate through the pads on a 25-day sequence, thus within 100 days, each leaching paddock will have the following operations: stacking of ore (25 days), leaching of ore (25 days), rinsing of leached ore (approximately 25 days), and reclaiming of rinsed ore (~25 days)
- Each pad will have 234 000 t of agglomerate ore and a basal footprint of 150 x 140 m
- The overall stacking philosophy is "retreat stacking" within each leaching pad.

17.3.6.3.2. Operation and equipment

- A ground-mounted fixed conveyor (OCV01) will be constructed running the length of the four pads and connecting to the front end of the plant at the battery limit to receive agglomerated ore
- A chain of mobile grasshopper conveyors (HOCV1-4) and radial stacker linked to the tripper conveyor with a ramp conveyor (RCV1) will extend the length of the pad and radially stack from side to side with the mobile conveyors being removed as the stacking front retreats closer to edge of the pad. Unused mobile conveyors are towed to a waiting area close to the next pad ready for deployment. Mobile conveyors are towed by a FEL (e.g. a WA500 size or similar)
- As retreat stacking in the stacking pad reaches completion, the final components of MHS1 are relocated to the next pad and commissioned for stacking operations
- The fixed conveyor is 900 mm wide running on the north side of the four-leaching pad. The conveyor will be provided with a motorised tripper car which will feed the ramp conveyor depending on which pad is being fed
- The ramp conveyor will feed the row of skid-mounted 900 mm wide x 45 m wheeled grasshopper conveyors; the grasshopper conveyors will discharge onto the 900 mm wide x 35 m radial stacker feed conveyor (SFC1) which will in turn feed the stacker conveyor (SCV1). The ramp, grasshopper, and transverse conveyors (TSV1) are manually positioned using the FEL

- The SFC1 is initially positioned by the FEL. When required to retreat, the stacker feeder wheel hydraulic drive is actuated manually. The stacker rotation is driven by a hydraulic drive operated manually
- The SCV-1 will discharge in an arc formation on the leaching cell. The stacker and its feed conveyor “retreat stacks” in 1 m or so steps for the full length of SFC01 during stacking thereby reducing the need for repositioning the grasshopper conveyors
- The mobile conveyors will be provided with interconnecting (plug and socket) electrical supply lines and suitable chocks to prevent the conveyors from rolling down-gradient.

The equipment is summarised in Table 17-13.

Table 17-13: Summary of stacking system MHS1

Agglomerate ore stacking system components	Total Units
Spine conveyor (OCV1)	1
Ramp conveyors (RCV1)	1
Grasshopper conveyors (HOCV1-4)	4
Transverse conveyor (TSV1)	1
Stacker-feed conveyor (SFC1)	1
Stacker (SCV1)	1
WA500 FEL	1

17.3.6.4. Materials handling system 2A: Spent ore reclaim

An illustration of the sequence for reclaiming leached (“spent”) ore is shown in Figure 17-21 and is described below.

17.3.6.4.1. Operation and equipment

- Spent ore is reclaimed from each leaching pad using two FELs. The FELs (a WA900 version or similar) will feed ore to a loading bin mounted on a grasshopper conveyor with a 1 500 mm wide belt which feeds a grasshopper conveyor system to a static loading bin on the overland conveyor (SpOCV02)
- The grasshopper conveyor system is estimated to be a maximum of four conveyor lengths (SPHOCV6 to 8) including the grasshopper with feed bin (SpHOC5Vb)
- SPOCV02 is a ground-mounted troughing belt conveyor, 900 mm wide running on the south of the four leaching pads
- The mobile conveyors will be provided with interconnecting (plug and socket) electrical supply lines and suitable chocks to prevent the conveyors from rolling down gradient
- Operations will be conducted like the MHS stacking system. The grasshopper conveyors will be stored until use and mobilised using the designated FEL.

The equipment is summarised in Table 17-14.

Table 17-14: MHS2A pad spent ore reclaim system

Spent ore stacking system components	Total Units
Front end loader (WA900)	2
Grasshopper conveyor (SpHOC5Vb) with mounted bin (leach pad)	1
Grasshopper conveyors (SpHOCV6-8) (leach pad)	3
Overland Conveyor (SpOCV02) with four static loading points	1

17.3.6.5. Materials handling system 2B: Spent ore stacking

An illustration of the sequence for reclaiming leached (“spent”) ore is shown in Figure 17-22 and is described below.

17.3.6.5.1. Philosophy and operations

- The overall stacking philosophy is “retreat stacking” within the SOD. The stacking system comprises ramp conveyors, grasshopper conveyors, and an identical stacking system to the on-off pads for ease of maintenance, training, operation and spares

- The stacking system connects to the fixed point at the end of SpOCV02. An off-load point is located at the end of SpOCV02 in case of maintenance or downtime on the stacking system
- To reach the far side of the SOD, a relocatable conveyor is included in the system and a second conveyor is needed for the southern area of the SOD along with four additional grasshopper conveyors
- The SOD is divided into operational areas, with deposition undertaken in sequence
- A D8 dozer is operated continuously to assist in moving material to the edges of the SOD footprint. Unused mobile conveyors are towed by the WA500 FEL or the D8 dozer to a waiting area close to the next pad ready for deployment
- The mobile conveyors will be provided with interconnecting (plug and socket) electrical supply lines and suitable chocks to prevent the conveyors from rolling down gradient.

17.3.6.5.2. Equipment

The equipment is summarised in Table 17-15.

Table 17-15: MHS2A Spent Ore Stacking System

Stacking System Components	Total Units
Overland conveyor (SpOCV02)	As per MHS2A - See Table 17-14
Ramp conveyors (SpRCV2-4)	3
Grasshopper conveyors (SpHOCV9-16)	9
Transverse conveyor (SpTSV2)	1
Stacker-feed conveyor (SpSFC2)	1
Stacker (SpSCV2)	1
Relocatable overland conveyor (SpROCV03)	1
Relocatable overland conveyor (SpROCV04) (Purchased in year 7)	1
Dozer (e.g. D8 size of similar)	1

17.3.6.6. Electrical

Power will be distributed by 11kV, 3-phase, 50 Hz distribution cables to the mini sub-stations. Substations will be provided with sufficient capacity at the following locations:

- Spine conveyor tail end
- Motorised tripper locations (skid-mounted substation).

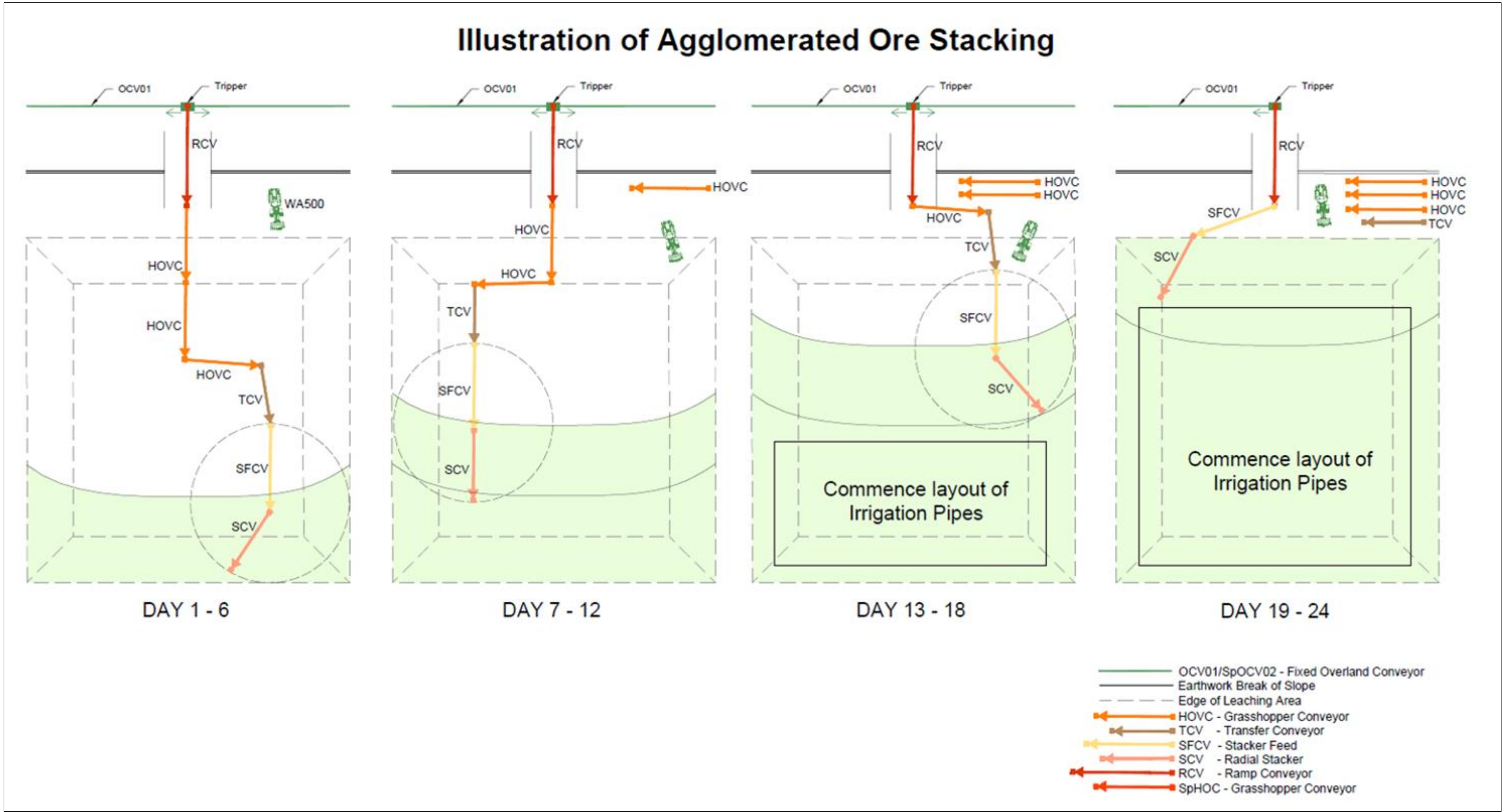

Figure 17-21: Schematic showing development of agglomerated ore stacking system

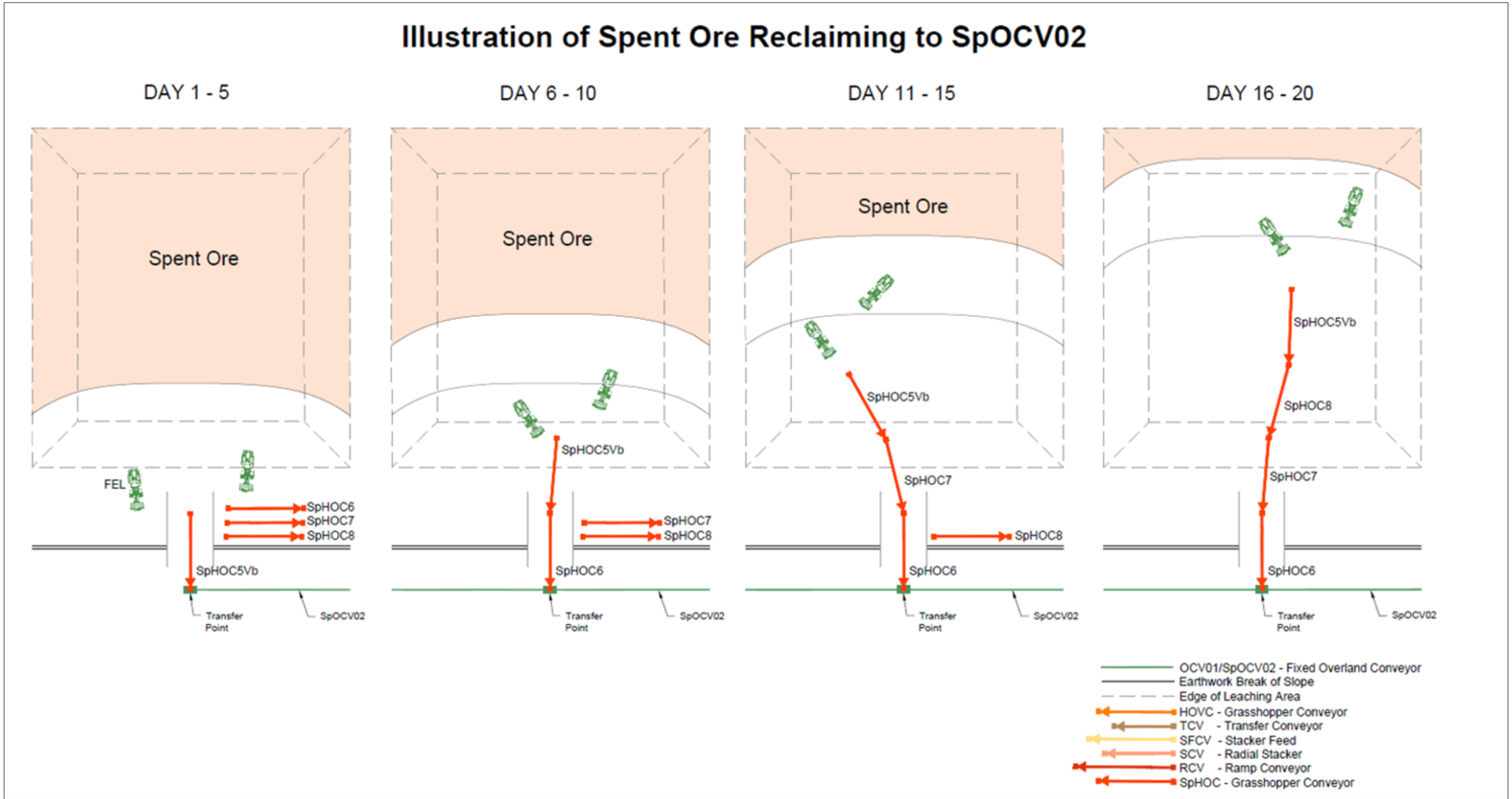

Figure 17-22: Schematic showing development of spent ore reclaiming system

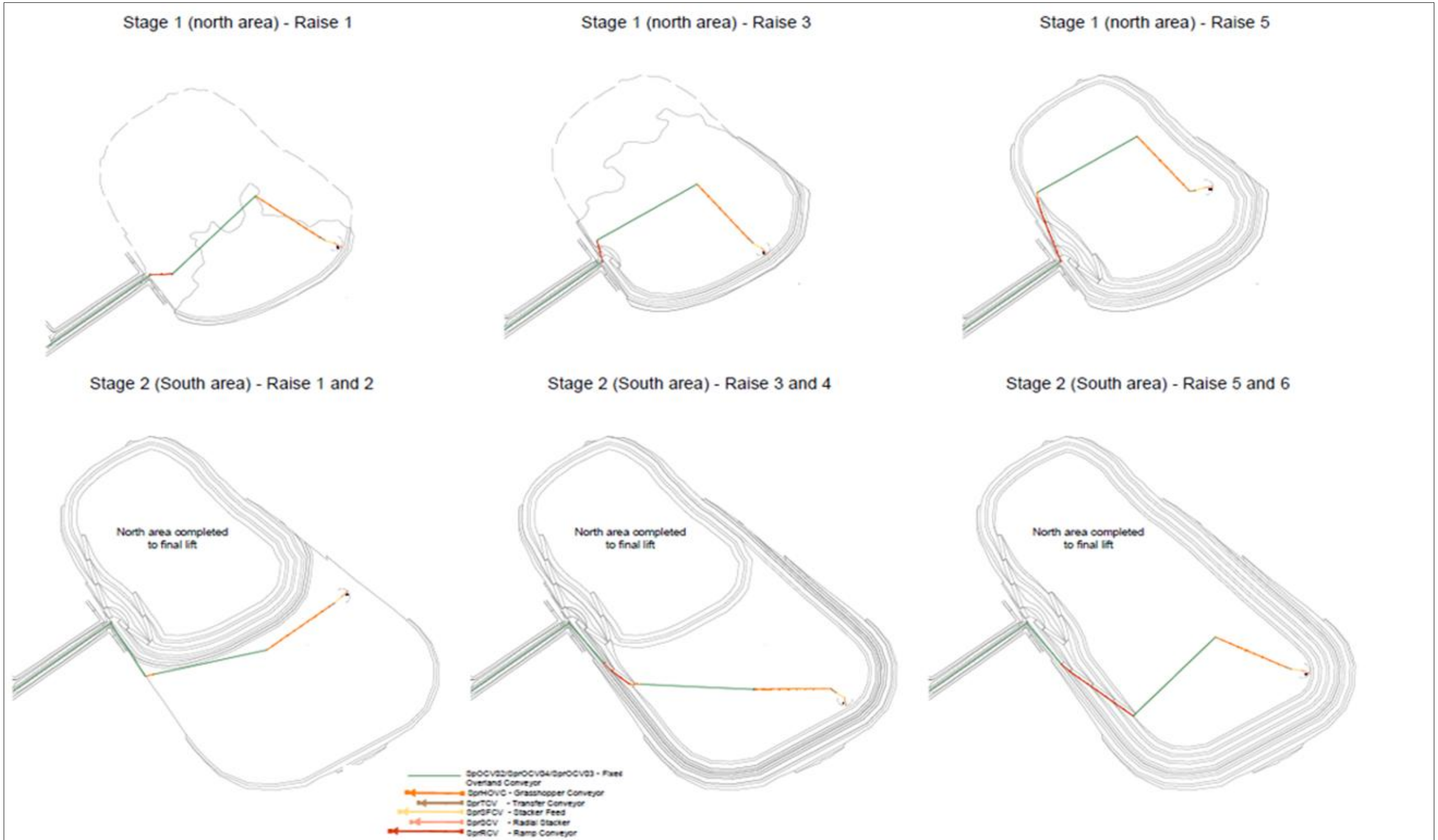

Figure 17-23: Illustration showing phasing of spent ore stacking system

Table 17-16: Stacking system (MHS1) with estimated parameters

Stacking system components	Units per system	Length [m]	Centres [m]	Capacity [tph]	Belt speed [m/s]	Installed power [kW]	Comments
OCV01/02	Overland conveyor/ Tripper wing	650 / 15	650	600	1.5	1 x 75 / 1 x 11	With motorised tripper Steel cord belt and tripper wing conveyor
RCV-01	Ramp conveyor	45	45	600	1.5	2 x 18.5	Fabric belt
HOCV01 to 04	*Grasshopper*	35	35	600	1.5	(4 of) 1 x 15	Fabric belt
TCV-01	Transverse	28	28	600	1.5	2 x 11	Fabric belt
SFCV-01	Stacker feed	39	39	600	1.5	1x 18.5	Fabric Belt
SCV-01	Radial stacker conveyor	32 / 9	32 / 9	600	1.5	1 x 41	Fabric belt. Extendable chute.

Table 17-17: Reclaiming System (MHS2) with estimated parameters

Stacking system components	Name	Length [m]	Centres [m]	Capacity [tph]	Belt speed [m/s]	Installed power [kW]	Comments
SpHOC5Vb	*Grasshopper (with FEL feed bin)*	35	35	600	1	2 x 30	1 500mm wide belt to handle FEL loading surges.
SpHOCV6-8	*Grasshopper conveyors*	35	35	600	1.5	(3 of) 1 x 15	Fabric belt; located on leach pad to reduce FEL tramming distance.
SpOCV02	Spine/ Overland conveyor	1,050	650	600	1.5	1 x 90	With transfer points for grasshoppers

Table 17-18: Reclaiming System (MHS2) with estimated parameters

Stacking system components	Name	Length [m]	Centres [m]	Capacity [tph]	Belt speed (m/s)	Installed power [kW]	Comments
SpRCV2-4	Ramp conveyors	45	45	600	1.5	(3 of) 2 x 18.5	Fabric belt
SpHOCV9-17	Grasshopper conveyors	35	35	600	1.5	(4 of) 1 x 15	Fabric belt
SpTSV2	Transverse conveyor	28	28	600	1.5	2 x 11	Fabric belt
SpSFC2	Stacker-feed conveyor	39	39	600	1.5	1x 18.5	Fabric belt
SpSCV2	Stacker	32 / 9	32 / 9	600	1.5	1 x 41	Fabric belt. Extendable chute.
SpReOCV03	Relocatable overland conveyor	450	450	600	1.5	1 x 55	Steel cord belt

17.3.6.7. SRK conclusions

Several trade-off studies have been undertaken to define the best option for the proposed stacking and reclaiming system. The proposed system is a standard, flexible, low-cost, low risk and well-known system and operation which is tried and tested on comparable operations.

17.3.6.8. Risks

17.3.6.8.1. Mobile equipment

As the project ramps up in year 1 and DHLP operations are developed and better understood, GoviEx may see a need for additional mobile equipment to support DHLP operations; specifically: an additional D8 dozer on SOD, a smaller D6 dozer to support on-off pad operations.

17.3.6.8.2. Trafficability

Geotechnical testing is required to confirm the design inputs for rubber wheels on the mobile grasshopper conveyors and radial stacker.

17.3.6.8.3. Spent ore stacking system

During detailed design scheduling of the SOD, the length of the relocatable conveyors should be revisited and optimised if necessary.

17.3.6.8.4. Front-end loader operations and solution systems

As operations progress, there may be a requirement to replace the cover material between the agglomerated ore and solution collection pipes in the event that spent ore clogs the matrix of the cover material and slows the rate of downward percolation of solutions.

17.3.6.8.5. Pad cyclicity

HLP material handling and irrigation pipe laying and take-up will be intensive. While rinsing may take less than 25 days, which means reclaiming operations could commence earlier than day 75 of the full 100-day cycle, a key time pressure will be the completion of stacking and commencement of irrigation for leaching.

17.3.6.9. Opportunities

During detailed design, there is the opportunity to review the alignment of the ROM pad, plant front end and on-off pads and switch the stacking system to the south side of the pad and the reclaim conveyor to the north side of the pads. This switch is perceived to have no material impact on capital cost or the proposed operational philosophy but would reduce the risk of FEL operations close to the solution reclaim systems, which are also on the south side.

17.3.6.10. Recommendations and next steps

Detailed earthworks scheduling of the SOD and design check against the equipment components of the spent ore stacking system.

17.3.7. Construction

17.3.7.1. Pad construction

The DHLF pad construction will include excavation and placement of compacted fill, foundation preparation, and installation of the liner system and solution collection system.

17.3.7.1.1. Excavation and foundation preparation

The pad foundation will be prepared before any placement of compacted fill. To achieve a 1˚ slope, approximately 110 000 m^3 of excavation will be required in the northwest area of the pad. In the remaining area, any vegetation will be removed. The entire pad foundation surface will be scarified and conditioned prior to placement of compacted fill and liner.

17.3.7.1.2. Compacted fill

Where excavation of material to a 1˚ slope is not required, compacted fill will be placed to achieve the required gradient for drainage. Compacted fill generated from excavation of the DHLF, and the associated ponds will be used to achieve the final surface demonstrated in the design drawings. Compaction should be achieved using mechanical methods such as a roller, whilst ensuring that the moisture content of the material is maintained close to optimum. Specific earthworks specifications will be required as part of the detailed design phase.

17.3.7.2. Liner system and drainage layer

The proposed liner system for the DHLF will consist of a double synthetic system, with a GCL, then two layers of HDPE geomembrane separated by a synthetic geodrain layer. The base grade must be sufficiently prepared to install the GCL. Installation of the liners requires specialist contractors with prior experience in installing composite liner systems on HLPs, such as those invited to tender as part of this FS. In addition, the integrity of the liner following installation should be checked by a QAQC engineer.

The drainage layer of 0.5 m thickness shall be placed on top of the liner consisting of crushed material from the pit box cut. Within this drainage layer, pipework in a herringbone formation to collect the solution should be placed. The details of this are shown in Figure 17-8.

17.3.7.3. Solution corridor construction

17.3.7.3.1. Foundation preparation

Foundation preparation of the ponds will include the removal of any vegetation and excavation to the required depth, ensuring sufficient depth and width for the pipework. Gravity-driven conveyance to the ponds from the DHLF. In some places, some dam construction will be required to give proper freeboard on the downstream slopes.

17.3.7.3.2. Solution corridor piping

A summary drawing showing the different aspects of the solution distribution and collection system is included in Figure 17-25. Solution movement around the heap is shown in GTC-003 to 005. Pipework that brings solutions to the ponds and from the PLS pond to the plant will utilise 500 mm diameter HDPE PE100 PN16, SANS 4427, plain-ended pipes that accommodate up to 16 Bar (250 PSI) water pressure.

Most pipes running to and from the pads and ponds are proposed to be 250 mm diameter HDPE PE100 PN16, SANS 4427, plain-ended pipes that accommodate up to 16 Bar (250 PSI) water pressure.

A lateral distribution line, also of 250 mm diameter HDPE will feed 13 lines on the heap comprised of 25 mm diameter HDPE PE100 PN16, SANS 4427, plain-ended pipes that accommodate up to 16 Bar (250 PSI) water pressure. The pipe will be laid and taken up using a spool feeder fixed to a low-profile vehicle and laid manually.

At equal intervals along this line, wobbler sprinkler heads will be utilised to distribute the acid. The proposed sprinkler head is the Senenger Xcel-wobbler CM which is acid resistant (Figure 17-24) as it is considered the most efficient in terms of volumes applied and coverage. The specifications of the wobblers are shown in Table 17-19.

To design the sprinkler arrangement, the Senenger WPSSPIv3 software was used. The results of the simulation are shown in Figure 17-25. From these calculations, a total of 808 units for the four pads are required: 182 complete units per pad, with approximately 20 spare units.

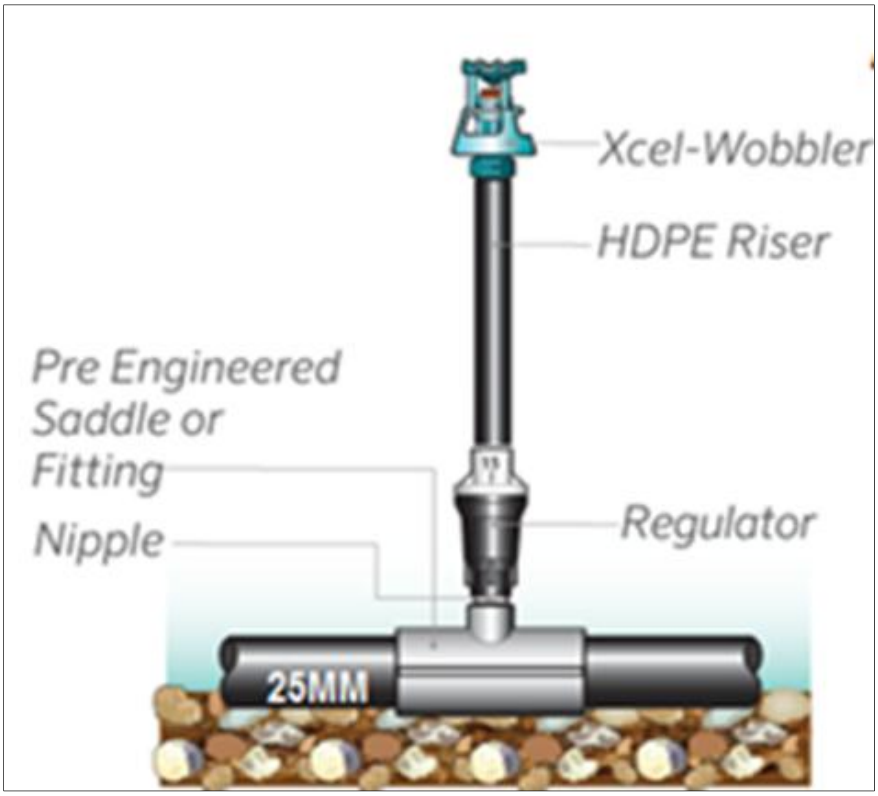

Figure 17-24: Xcel-Wobbler proposed on heap fed by 25mm HDPE pipe

Table 17-19: Acid resistant fittings for sprinkler heads

Fitting	Size	Chemical resistance
Xcel wobbler CM	20mm male	High
PRL20CMS regulator	20mm female	High
PP/SS clamp saddle	25mm x ¾''	Medium
HDPE plastic riser pipe	¾''x 850mm	High
Stainless steel nipple	20mm, ¾''	High
Stainless steel socket	20mm, ¾''	High

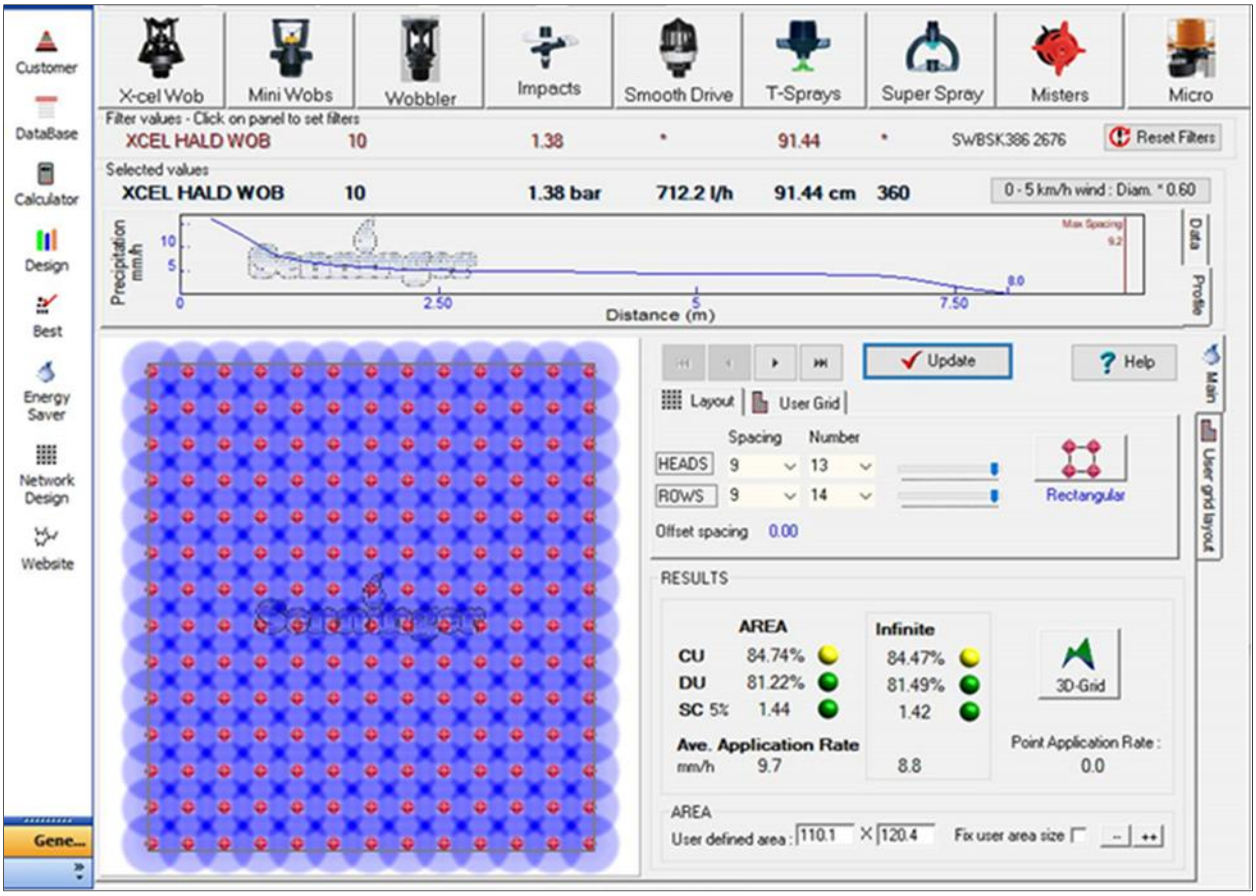

Figure 17-25: Solution distribution via 182 headers on active pad showing complete contact of ore with acid based on 9 heads on 13 lines

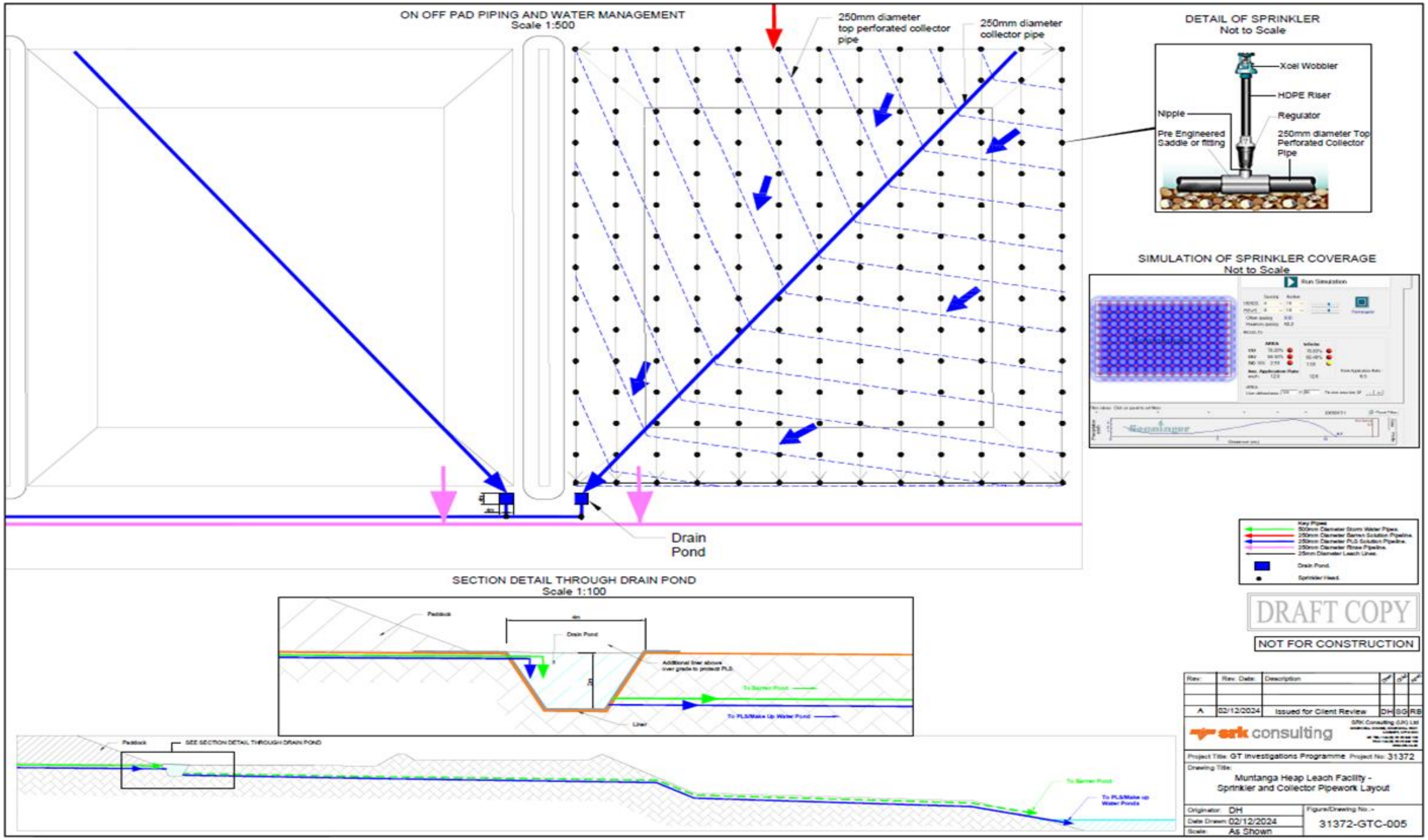

Figure 17-26: GTC005 solution collection system pipework

17.3.7.4. Pond construction

17.3.7.4.1. Foundation preparation

Foundation preparation of the ponds will include the removal of any vegetation and excavation to the required depth, ensuring gravity-driven conveyance to the ponds from the DHLF. In some places, some dam construction will be required to give proper freeboard on the downstream slopes.

17.3.7.4.2. Liner system

The slopes of the ponds have been set at 2.5H:1V to facilitate the installation of the liner system. The liner system proposed for the barren, bleed and PLS ponds is a double liner system as discussed for the DHLF. The liner proposed for the stormwater and rinse pond is a single-liner system with HDPE geomembrane as no solution will be stored in the pond.

17.3.7.5. Spent ore dump construction

The SOD construction will include excavation and placement of compacted fill, foundation preparation, construction of a starter embankment and installation of a drainage layer. At this stage, a liner has not been included because of preliminary geochemical test results. This will need to be confirmed at a detailed stage and is discussed further in Section 17.1.7.8.1.

17.3.7.5.1. Excavation and foundation preparation

The SOD foundation will be prepared before any placement of compacted fill. To place the drainage layer and ensure a general flow towards the centre of the dump, any vegetation will be removed. The entire area foundation surface will be scarified and conditioned prior to placement of compacted fill.

17.3.7.5.2. Compacted fill

Compacted fill will be placed to achieve the required gradient for drainage. Compacted fill generated from the excavation of the SOD and the associated SOD stormwater pond will be used to achieve the final surface demonstrated in the design drawings. Compaction should be achieved using mechanical methods such as a roller, whilst ensuring that the moisture content of the material is maintained close to optimum. Specific earthworks specifications will be required as part of the detailed design phase.

17.3.7.5.3. Drainage layer

The proposed drainage layer will consist of crushed waste rock derived from the pit box cuts. The drainage materials should be selected and placed to achieve a higher permeability than the spent ore. The drainage layer should be placed across the entirety of the SOD footprint at a minimum thickness of 0.5 m, but thicker in central drainage channel areas to be defined during the detailed design stage. Specific earthworks specifications will be required as part of the detailed design phase.

17.3.7.5.4. Starter embankment

The proposed starter embankment will consist of crushed waste rock derived from the pit box cuts. The embankment is 10 m in height with a 10 m crest width. The slopes shall be placed at a 1 in 2 angle. Material should be selected and placed to achieve geotechnical stability and allow drainage. Geotechnical laboratory testing and specific earthworks specifications will be required as part of the detailed design phase.

17.3.7.6. Quantities

The main cost items included in the DHLP cost estimate are:

- Soil and rock excavation for DHLP, ponds and diversion channels
- GCL and HDPE for the liner system
- Pipework and
- Monitoring installations (including vibrating wire piezometers and standpipes).

The most significant cost items from the estimated quantities include the excavation for ponds, earthworks for the liner system, and the HDPE and GCL for the DHLF. A basis for an estimate for each of the quantity items is summarised in Section 20.

17.3.7.7. Environmental considerations

17.3.7.7.1. Heap leach pad solution containment

Careful consideration should be given to the proper installation and maintenance of the liner system to reduce the likelihood of leakage. The solution may be lost through a geomembrane due to defects created during manufacturing, accidental punctures during installation, or defects in seams. Proper construction QA should be undertaken during liner installation.

Key factors that will impact the liner system performance and potential leakage rate in the event of a defect include:

- The size of the defect
- Solution head: the head of the solution refers to the vertical height at which the solution is anticipated to accumulate over the geomembrane, meaning the higher the solution lever, the greater the head. A higher head results in increased water pressure at the liner's surface, leading to more water being forced through any defects. To reduce the water level above the liner, a free-draining granular layer, a piping collection system, or a combination of both can be used. In this case, both are to be used
- Contact quality between the geosynthetic clay liner and the HDPE; for example, a geomembrane that has been installed with multiple waves or wrinkles will have poor contact conditions.

17.3.7.7.2. Dynamic heap leach facility closure

The DHLF will be closed and reclaimed as per the criteria set out in relevant local regulations and international standards, the main activities will include capping, regrading, landscaping, and covering with available soil.

17.3.7.7.3. Spent ore disposal

The SOD will be constructed in two phases, with the northern section being constructed to its maximum height in the first eight years; following this, the second stage will be constructed over the whole SOD footprint. This staged construction will allow for progressive rehabilitation before final closure. The SOD will be closed in line with relevant local regulations and international standards and will be reclaimed by regrading, landscaping and covering with available soil.

17.3.7.7.4. Gypsum pond breach

Process gypsum is a notoriously difficult substance to dispose of and has subsequently been the cause of several major failures worldwide because it is hydrophobic and stays in suspension with low geotechnical strength, as opposed to consolidating and giving up water for pumping. In addition, this gypsum may contain some leached metals and therefore may be a potential pollutant, although the current test work suggests low uranium concentrations. The gypsum pond has been designed to be below ground, to avoid dam failure and make the closure of a below ground structure simpler, and large to promote as much evaporation as possible. During operation, however, careful management of the gypsum is required and revisions to the process may be necessary. On closure, the liner will be cut, folded over and buried.

17.3.7.8. Recommendations

Based on the evaluations described above in this report, SRK recommends that the following work be done as part of the final design:

- The final design will require earthworks specifications which may need to be supplemented with test work to ensure that excavated and placed founding and drainage layers
- Detailed geochemical characterisation of the spent ore should be undertaken (including kinetic testing) to confirm that the material is suitable such that no liner is required for the SOD
- Previously work has been done to characterise the site; however, it was established that a more detailed seismic assessment of the site is required to set appropriate loading conditions in any modelling. Following a more detailed seismic hazard assessment, an update to the associated SOD stability analyses should be completed
- No station data are currently available for evaporation or rainfall at the site, hence only regional data could be used. The data used here should be compared to any data that becomes available on-site in the future to ensure appropriate assumptions are made.

17.4. Processing plant

The central processing plant ("CPP") handles a combined total of 3 500 000 tonnes per annum ("tpa") of ROM material sourced from the central Muntanga and Dibbwi East mining sites and sorted ore from the Dibbwi, Gwabi and Njame satellite mining sites. The mix of ore from the respective pits will vary over time.

The flow sheet encompasses primary (with ore sorting at satellites), secondary, and Tertiary crushing stages, heap leaching, and uranium recovery by Ion Exchange ("IX") followed by the recovery of U_3O_8 by eluting the uranium-loaded resin using a sulfuric acid solution. The eluate undergoes a nanofiltration process facilitating sulfuric acid recovery for recycling to the elution process. Following this, the concentrated solution is carefully dosed with hydrated lime and sodium hydroxide to neutralise residual acid and remove deleterious minerals such as Fe, after which it is dosed with hydrogen peroxide, leading to uranyl peroxide precipitation. The precipitate is dewatered, calcined and packed into drums as U_3O_8 or yellowcake.

The design is based on receiving 3.5Mtpa of ore, at a steady-state head grade from the central pits of 295 ppm U_3O_8.

17.4.1.Process design basis

A summary of key parameters is provided in Table 17-20

Table 17-20: Summary of key parameters

Parameter	Unit	Value
Ore source		
Muntanga and Dibbwi East pits	Mtpa (max)	3.5
Ore grade		
Dibbwi East	g/t U_30_8	295
Muntanga	g/t U_30_8	295
Uranium recovery (overall)		
Dibbwi East oxide	%	91.3
Dibbwi East primary	%	89.7
Muntanga	%	93.0

17.4.2.Process description

This section describes the processing facility and should be read in conjunction with the following set of process flow diagrams ("PFDs"):

- M7610-C01043-P120-001-001Heap Leach & CIX Plant Stacking & Ponds - PFD
- M7610-C01043-P120-001-002Heap Leach & CIX Plant Stacking & Ponds - PFD
- M7610-C02003-P120-001-001Heap Leach & CIX Reagents Sulfuric Acid Receipt & Storage - PFD
- M7610-C02013-P120-001-001Heap Leach & CIX Reagents Peroxide Storage - PFD
- M7610-C03003-P120-001-001Heap Leach Water & Services Raw Water - PFD
- M7610-C03013-P120-001-001Heap Leach Water & Services Process Water - PFD
- M7610-C03023-P120-001-001Heap Leach Water & Services Potable Water - PFD
- M7610-C03033-P120-001-001Heap Leach Water & Services Gland Service Water - PFD
- M7610-C03063-P120-001-001Heap Leach Water & Services Plant & Instrument Air - PFD
- M7610-C04003-P120-001-001Elution & Precipitation Plant PLS Clarifier - PFD
- M7610-C04003-P120-001-002Elution & Plant IX Adsorption Train 1 - PFD
- M7610-C04003-P120-001-006Elution & Precipitation Plant Safety Screens & Tank Farm - PFD
- M7610-C04013-P120-001-001Elution & Precipitation Plant Resin Regeneration - PFD
- M7610-C04023-P120-001-001Elution & Precipitation Plant Gypsum Precipitation - PFD
- M7610-C04023-P120-001-002Elution & Precipitation Plant Gypsum Filtration - PFD
- M7610-C04023-P120-001-003Elution & Precipitation Plant Uranyl Peroxide Precipitation - PFD
- M7610-C04033-P120-001-001Elution & Precipitation Plant Drying & Packaging - PFD
- M7610-C04053-P120-001-001Elution & Precipitation Plant Nano Filtration - PFD
- M7610-C05003-P120-001-001Elution Reagents Sulfuric Acid Offloading & Make-Up - PFD
- M7610-C05013-P120-001-001Elution Reagents Sodium Hydroxide Make-Up & Storage - PFD
- M7610-C05023-P120-001-001Elution Reagents Peroxide Storage - PFD
- M7610-C05033-P120-001-001Elution Reagents Milk of Lime Make-Up - PFD
- M7610-C05053-P120-001-001Common Infrastructure Fire Water - PFD
- M7610-C06003-P120-001-001Elution Plant Water & Services Demineralised Water -– PFD.

sulfuric

17.4.2.1. Heap leach and ponds

PFD Reference: C01043P120001001 and C01043P120001002

The agglomerated material is conveyed (C0104-CV-001) and stacked at a central heap pad (C0104-HL-001). The heap is irrigated with barren solution initially pumped via pump (C0104-PP-001) from the barren solution pond (C0104-PD-001). Sulfuric acid and hydrogen peroxide are added using inline mixers (C0104 MX 001/002) under pH and eH control, to enable the uranium leaching process. As the leach solution becomes enriched with uranium, it drains into the PLS pond (C0104-PD-002). The pregnant leach solution is then pumped (via pumps C0104-PP-003/004) to the central plant.

17.4.2.2. Ion exchange absorption and elution circuits

PFD Reference: C04003P120001001/1002/1004/1005

The pregnant leach solution ("PLS") is transferred from the PLS ponds via pumps (C0400-PP-003/004) to the PLS clarifier (C0400-CL-001). The underflow from the PLS clarifier is returned to the barren pond through the underflow pumps (C0400-PP-003/004), duty and standby. The overflow from the clarifier is directed to the PLS feed tank (C0400-TK-001). The PLS drain lines from the loaded resin transfer vessels also return to the PLS tank. Subsequently, the PLS is delivered by pumps (C0400-PP-005/006), duty and standby, to the NIMCIX adsorption column.

There is a single NIMCIX adsorption column (C0400-AC-001) and a single elution column (C0400-EC-001) at the central processing plant.

Resin is introduced at the top of the NIMCIX column (C0400-AC-001), while the PLS feed to the column flows upward from the bottom of the column. As the PLS progresses through the absorption column, uranium loads onto the resin in a counter-current process. The barren overflow from the NIMCIX adsorption column reports to the barren pond (C0104-PD-001) via a safety screen (C0400-SC-001) and the barren solution return tank (C0400-TK-003). The loaded resin is periodically removed from the base of the NIMCIX adsorption column into the loaded resin transfer vessel (C0400-TV-001). After draining barren from the vessel, a 10 g/l sulfuric acid solution is introduced to wash co-loaded iron from the resin. This solution is recycled back to the PLS feed tank. Eluate solution then pressurises the vessel, transferring the resin into the top to the elution columns (C0400-EC-001/002). The transfer vessels are equipped with a screen (C0400-FL-001/003) to retain resin while allowing the PLS and eluate solution used for resin movement to be drained and pumped back to the PLS pond and eluate tank via pumps (C0400-PP-009/015 and C0400-PP-007/017) respectively.

The loaded resin added into the elution columns (C0400-AC-001/002) is contacted with 100 g/L sulfuric acid (eluent) fed into the base of the elution columns. Similar to the absorption columns, a counter-current flow of resin and eluate occurs within the elution column. The resin is stripped of uranium, while the eluate becomes pregnant with uranium. The eluate solution exits the elution columns from the top and is screened (C0400-SC-002) to remove the entrained broken resin before being sent to the eluate tank (C0400-TK-002) for further processing. The stripped resin is periodically removed from the base of the elution columns into the eluted resin transfer vessel (C0400-TV-002). Barren solution is introduced to assist in transferring the eluted resin back into the absorption column (C0400-AC-001). The eluted resin transfer vessels are equipped with a screen (C0400-FL-002/004) to retain the resin during drainage of liquids, such as eluent and barren solution used for resin transfer, via pumps (C0400-PP-013/021 and C0400-PP-011/019) respectively.

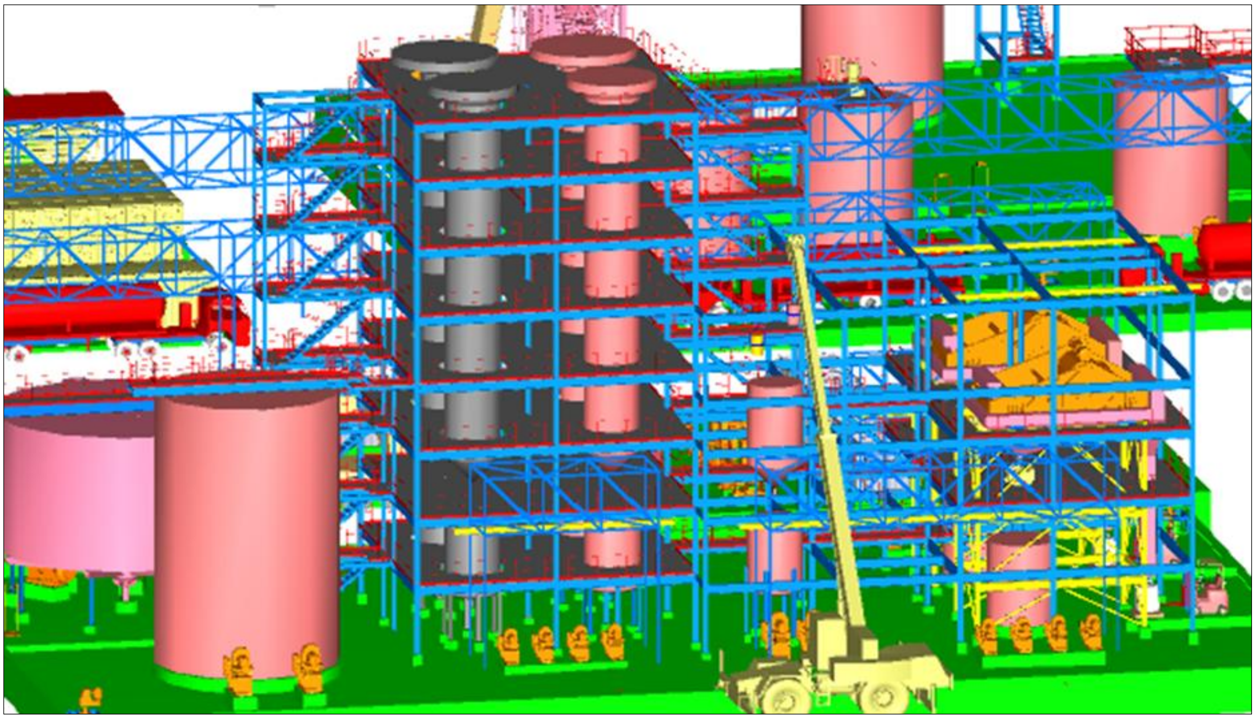

Figure 17-27: 3D Layout – IX absorption and elution circuit

17.4.2.3. Ion exchange resin regeneration circuit

PFD Reference: C04013P120001001

Every tenth batch of resin will be regenerated to remove silica poisoning. The regeneration cycle steps are described below:

17.4.2.3.1. Regeneration step #1: Filtered water wash

The regeneration process commences by transferring a batch of resin to the regeneration vessel (C0401-RV-001). Resin transferred from elution to regeneration will retain significant amounts of acid, necessitating a rinsing step before regeneration. The resin undergoes a wash with two bed volumes of filtered demineralised wash water, sourced from the demineralised wash water storage tank (C0600-TK-001). This water wash helps remove residual acid from the resin, preventing sudden pH changes that could lead to resin degradation due to osmotic shock. Additionally, using wash water for Thar 2 - Closure initial step facilitates the flushing of the common pipeline and pump shared among all regeneration solutions, preventing cross-contamination between regeneration cycles.

17.4.2.3.2. Regeneration step #2: Dilute sodium hydroxide wash

The next regeneration step involves washing the resin with a dilute sodium hydroxide solution (10 g/L to 25 g/L), stored in the weak sodium hydroxide storage tank (C0501-TK-001). This wash precedes the concentrated solution wash to prevent resin degradation from osmotic shock caused by abrupt pH changes. Spent sodium hydroxide is either returned to the weak sodium hydroxide storage tank or transferred to the gypsum disposal facility (to be confirmed) using the spent caustic pump (C0401-PP-006).

17.4.2.3.3. Regeneration step #3: Concentrated sodium hydroxide wash

Following the dilute sodium hydroxide wash, the resin undergoes a wash with two-bed volumes of concentrated sodium hydroxide solution (40 g/L) from the sodium hydroxide storage tank (C0600-TK-001) in the reagent make-up area. Spent sodium hydroxide is collected and transferred to the gypsum disposal facility using the spent caustic pump (C0401-PP-006).

17.4.2.3.4. Regeneration step #4: Filtered water wash

After the fresh caustic wash, the resin is rinsed of residual caustic in a water wash. The water wash is directed to the caustic effluent tank and transferred to the gypsum disposal facility using the spent caustic pump (C0401-PP-006). This post-regeneration water wash removes any remaining caustic from the resin before the sulfonation (acid wash) step, preventing degradation from osmotic shock.

17.4.2.3.5. Regeneration step #5: Dilute sulfuric acid wash

The final step in the regeneration cycle is the sulfonation of the resin. A dilute sulfuric acid (10 %) solution is transferred from the reagent make-up area and stored in the dilute sulfuric acid storage tank (C0500-TK-002). The dilute acid is pumped to the regeneration circuit. Spent acid effluent from the regeneration vessel is pumped to the barren pond to rejoin the main leach circuit.

17.4.2.4. Nanofiltration

PFD Reference: C04053P120001001

The final eluate from the IX Elution circuit undergoes treatment to recover sulfuric acid from the eluate whilst concentrating the uranium in the stream. This concentration aims to produce a liquor with reduced treatment costs for uranium precipitation.

To achieve this, the eluate stream is directed to a pre-filtration system, comprising one duty unit and one standby unit (C0405-FL-001/002), preventing membrane blinding by suspended solids. The filtrate is collected in the nano-filtration feed tank (C0405-TK-001).

Two separate membrane housing units (C0405-NF-001/002), each with 14 membranes, provide operational flexibility. Each unit is equipped with its dedicated feed pump (C0405-PP-002) and (C0405-PP-003), with a common standby pump (C0405-PP-001). The two resulting product streams post nano-filtration are a permeate stream with high acid concentration and low uranium concentration, collected in the permeate collection tank (C0405-TK-004), and a rejects stream with increased uranium concentration relative to its acidity, accumulated in the rejects collection tank (C0405-TK-003).

The permeate is pumped using duty/standby pumps (C0405-PP-008/009) to the PLS feed tank (C0400-TK-001) for reuse in the elution circuit, while the uranium-rich reject stream is pumped to the gypsum precipitation circuit via duty/standby pumps (C0405-PP-006/007).

A clean-in-place system is provided for periodic membrane cleaning, comprising a cleaning solution storage tank (C0405-TK-002) and two duty/ standby discharge pumps (C0405-PP-004/005). The cleaning solution discharge undergoes filtration through duty/standby units (C0405-FL-004/005) before being directed to the selected membrane for cleaning. The resultant cleaning solution will contain trace amounts of uranium and will be pumped into the PLS feed tank (C0400-TK-001).

Any potential spillage in this area is directed to the spillage pump (C0405-SP-001) via the sump (C0405-SM-001). Due to the high acidity of the eluate and permeate streams this area is also equipped with a safety shower (C0405-SH-002) if someone comes into contact with one of these streams.

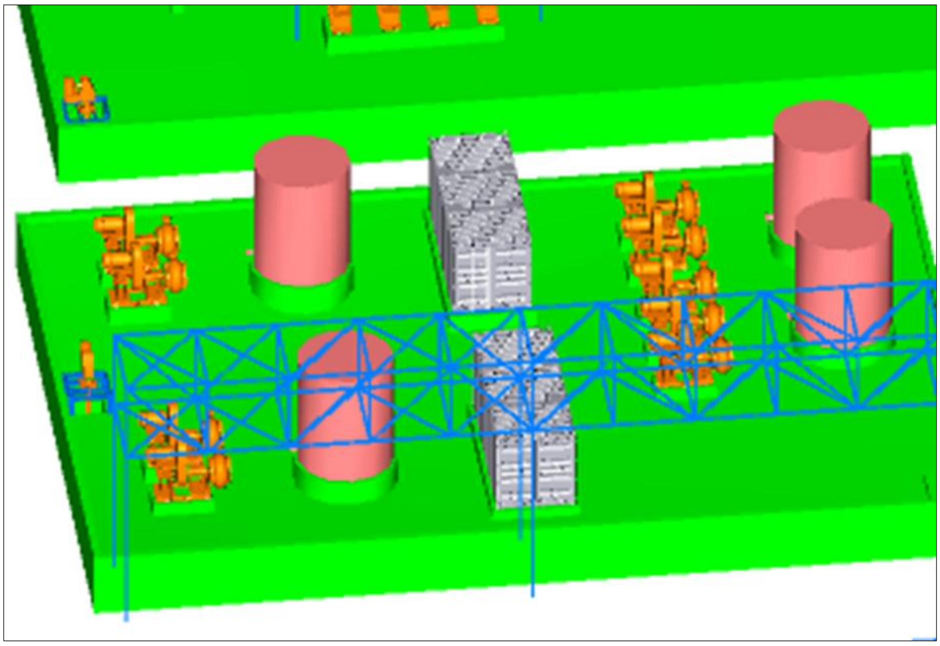

Figure 17-28: 3D Layout – nanofiltration circuit

17.4.2.5. Gypsum precipitation and filtration

PFD Reference: C04023P120001001 and C04023P120001002

The gypsum precipitation process involves six mechanically agitated tanks (C0402-TK-001/002/003/004/005/006) arranged in a series overflow configuration. The final eluate is transferred from the eluate tank (C0400-TK-004) to the gypsum precipitation circuit feed splitter (C0402-SL-001). The splitter directs the eluate to either of the first two precipitation tanks. Burnt lime slurry, utilised for precipitation, is pumped from the local day tank (C0402-TK-008) through a ring main system into any of the six precipitation tanks. Gypsum precipitation lime pumps (C0402-PP-003/004), operating as duty and standby, facilitate the movement of lime slurry through the ring main. A portion of the gypsum precipitate slurry is recycled from the last precipitation tank to the first tank using the gypsum precipitation seeding pump (C0402-PP-005), aiding crystal growth and precipitation rates. Bypass provisions are in place for any of the precipitation tanks.

The gypsum slurry formed in the precipitation tanks overflows into the gypsum thickener feed tank (C0402-TK-007). The slurry is pumped at a controlled rate from the thickener feed tank via duty and standby pumps (C0402-PP-001/002) into the gypsum thickener (C0402-TH-001). The underflow from the gypsum thickener is pumped to either of the two-duty gypsum plate and frame filters (C0402-FL-001/002). Gypsum filter feed pumps (C0402-PP-012/013), operating as duty and standby, feed gypsum slurry into the two filters, both equipped with variable speed drives. The filter feed rate depends on the filter system pressure controlled by the variable speed output.

To reduce uranium entrained in the filter cake, the cake is washed with a dilute sulfuric acid solution. Dilute sulfuric acid is transferred from the reagent make-up area, diluted in-line (via C0402-MI-001) with raw water to a concentration of 2 g/L, and stored in the dilute acid wash water tank (C0402-TK-009). The wash solution is then pumped to the respective filter using duty/standby dilute acid wash water pumps (C0402-PP-008/009). The wash solution, along with the gypsum thickener overflow, is collected in the gypsum removal filtrate collection tank (C0402-TK-010).

The gypsum filter cake from both filters is re-pulped in the re-pulp tank (C0402-TK-011) using a barren eluate solution. The re-pulped gypsum filter cake is transferred to the leach circuit via gypsum removal re-pulp slurry transfer pumps (C0402-PP-011/012), acting as duty and standby. If uranium co-precipitates with gypsum, repulping the gypsum cake prevents the loss of co-precipitated uranium. There is an option to discard the gypsum filter cake to the gypsum disposal facility circuit if the uranium grade is low.

The filtrate, gypsum thickener overflow, and wash solution from the gypsum filtration are collected in the filtrate collection tank (C0402-TK-010). The filtrate is pumped to the uranyl peroxide precipitation circuit to recover uranium as $UO_4.nH_2O$. Gypsum filtrate transfer pumps (C0402-PP-006/007), as duty and standby, facilitate the transfer to uranyl peroxide precipitation at a fixed controlled rate.

Spillage in the precipitation area is collected in the gypsum precipitation spillage sump (C0402-SM-001) and pumped back to the gypsum feed splitter (C0402-SL-001) using the gypsum precipitation spillage pump (C0402-SP-001).
A safety shower (C0402-SH-001) is provided due to the acidic environment and reagents utilised in this area.

17.4.2.6. Iron precipitation

PFD Reference: C04043P120001001

The filtrate obtained from the gypsum precipitation step undergoes a precipitation step to remove traces of iron that are introduced by the lime. This is done in a series of five agitated tanks (C0404-TK-001/2/3/4/5) using sodium hydroxide solution introduced under pH control from the dosing pumps (C0404-PP-006/7). The solution from the last tank is filtered using a filter press (C0404-FP-001) to remove the iron precipitate before proceeding to uranyl peroxide precipitation. Solids from the filter press are re-slurried (in tank C0404-TK-006) and pumped to the barren return tank (C0402-TK-017).

17.4.2.7. Uranyl peroxide precipitation and filtration

PFD Reference: C04023P120001003

The filtrate obtained from the iron precipitation step is pumped to the uranium peroxide precipitation circuit through the gypsum filtrate heat exchanger (C0402-HX-001). This hot water heat exchanger elevates the temperature of the diluted filtrate to 50ºC before precipitation. Uranyl peroxide is then precipitated from the heated filtrate in a sequence of four mechanically agitated tank reactors (C0402-TK-012/013/014/015) using hydrogen peroxide solution.

Concentrated hydrogen peroxide is transferred from the reagent make-up area to the uranium peroxide precipitation circuit, where it is inline-diluted (via dedicated mixers C0402-MI-002/003/004/005) to achieve the required concentration. Variable-speed hydrogen peroxide dosing pumps, dedicated to each reactor, are employed for precise dosing. Sodium hydroxide (caustic) solution is dosed into each precipitation tank at a controlled rate to maintain the pH during precipitation, with each tank equipped with a dedicated variable-speed caustic dosing pump.

A portion of the precipitate slurry is recirculated from the last precipitation tank to the first precipitation tank for seeding, facilitated by the final product seeding pump (C0402-PP-022). Seeding is carried out to enhance crystal growth and precipitation rates. Provisions are in place to bypass any of the tank reactors if necessary.

The resulting hydrated uranium peroxide slurry overflows into the final product thickener feed tank (C0402-TK-016) and is pumped to the uranyl peroxide wash circuit using final product thickener feed pumps (C0402-PP-020/021), operating as duty and standby.

The uranyl peroxide precipitate slurry, produced during product precipitation, contains trace amounts of acid and hydrogen peroxide that must be washed off and recovered before being pumped to final product handling. The uranyl precipitate washing is performed with water in a decantation process.

The uranyl peroxide precipitate is pumped into the uranyl peroxide product thickener (C0402-TH-002). The overflow, which contains trace amounts of acid and hydrogen peroxide, gravitates into the product thickener overflow tank (C0402-TK-017) and is pumped to the gypsum filtrate re-pulp tank (C0402-TK-010) using transfer pumps (C0402-PP-018/019), operating as duty and standby.

The underflow from the final product wash thickener is pumped to the final product handling area at a controlled rate and density using the final product wash thickener underflow pump set (C0402-PP-014/015), operating as duty and standby.

Spillage from this area is collected in the area spillage sump (C0402-SM-002) and pumped to the final product thickener using the spillage pump (C0402-SP-002).

The safety shower (C0402-SH-003) is provided for washing off any slurry and solution contact, ensuring safety in the acidic environment and handling of reagents.

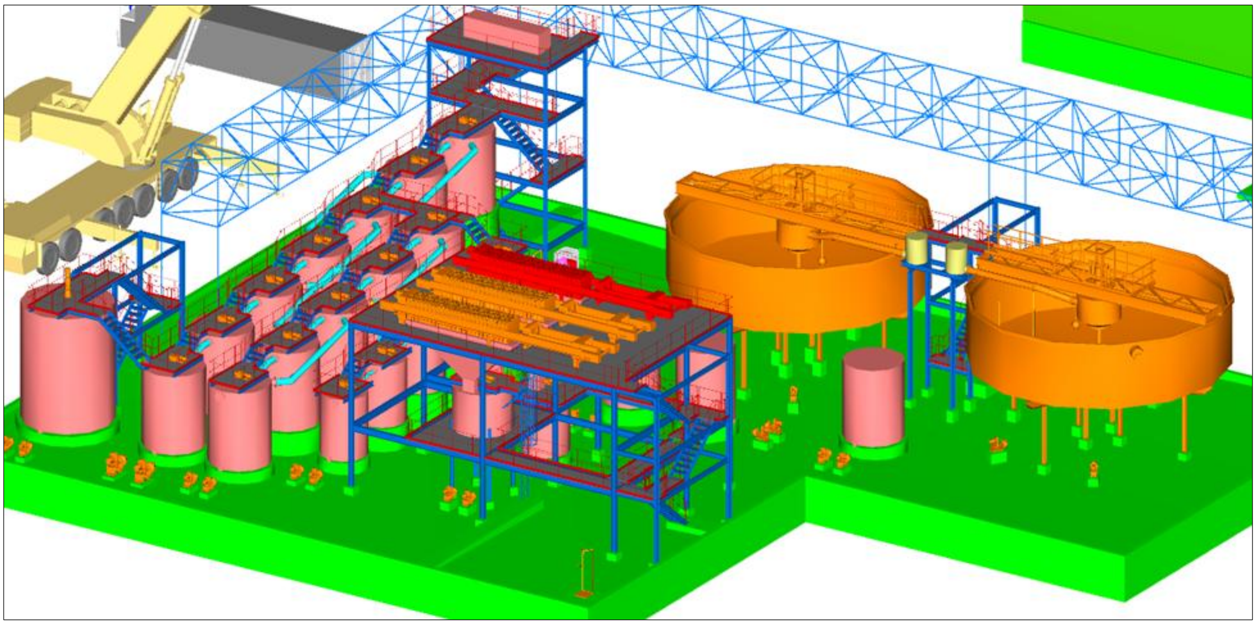

Figure 17-29: 3D Layout – gypsum precipitation circuit

17.4.2.8. Triuranium-octaoxide product handling

The uranyl peroxide product is transferred from the product thickener (C0402-TH-002) underflow to the centrifuge feed tank (C0403-TK-001) using thickener underflow pumps (C0402-PP-014/015). The thickener underflow is recycled back to the thickener feed until it reaches the required density, being transferred forward only if the density surpasses a set minimum.

The mechanically agitated centrifuge feed tank (C0403-TK-001, C0403-AG-001) primarily provides buffer storage capacity. The washed slurry is pumped from the centrifuge feed tank to the centrifuge (C0403-CF-001) using centrifuge feed pumps (C0403-PP-001/002/003), operating as duty and two standby pumps. The centrifuge separates most of the solution from the uranyl peroxide slurry, producing uranyl peroxide paste as the underflow. The solution separated from the slurry exits the centrifuge as the centrate, gravitating to the centrate surge tank (C0403-TK-004), and is recycled back to the uranyl peroxide precipitation tank (C0402-TK-016).

The solid discharge comprises yellowcake slurry with a paste-like consistency and a solids content of approximately 60 % to 70 % w/w. This yellowcake slurry is discharged into a hopper/chute connected to the uranyl peroxide screw feeder (C0403-SF-001), which transfers the material to the uranyl peroxide kiln dryer (C0403-KN-001).

The uranyl peroxide screw feeder conveys the centrifuge paste into the kiln heater (C0403-HE-001) at a controlled rate before feeding into the kiln (C0403-KN-001). The electrically heated kiln, with three heat zones, elevates the cake's temperature to approximately 750 °C, converting the uranyl peroxide into triuranium octoxide (U_3O_8) or yellowcake. The resulting yellowcake, with a moisture content of ≤ 1 % w/w, is discharged into the discharge conveyor (C0403-CV-002) and then deposited into the U3O8 storage hopper (C0403-HP-001) in the drumming plant.

The off-gas is drawn through the bag house located on the dryer, passing through a wet gas scrubber (C0403-SR-001) and a demist-packed column condenser (C0403-PC-001). The scrubbed gas is vented to the atmosphere using the ventilation fan (C0403-FA-004) through the stack (C0403-VS-004). The demist-packed column condenser (C0403-PC-001) condenses the steam and removes any entrained solids, with the condensed liquor pumped using pumps (C0403-PP-007) to the uranyl peroxide precipitation tank (C0402-TK-016).

Drumming of the final product, including sampling and lidding, is automated. To initiate a drum packing batch, six empty drums are manually placed onto the loading conveyor (C0403-CV-004). The drums are automatically conveyed to the filling conveyor (C0403-CV-005), positioning the drum under the dried U3O8 storage hopper (C0403-HP-001). Filling and sampling commence automatically, with the filled drum conveyed to the lidding conveyor (C0403-CV-007), where lids are fitted. The filled and lidded drums undergo automatic washing, drying, and weighing on conveyors (C0403-CV-008). The plant operator then attaches labels to the drums and loads them for dispatch to the packaged product storage area.

Any spillage in this area is collected in the refinery spillage sump (C0403-SM-001) and pumped to the final product thickener using the refinery spillage pumps (C0403-SP-001/002).

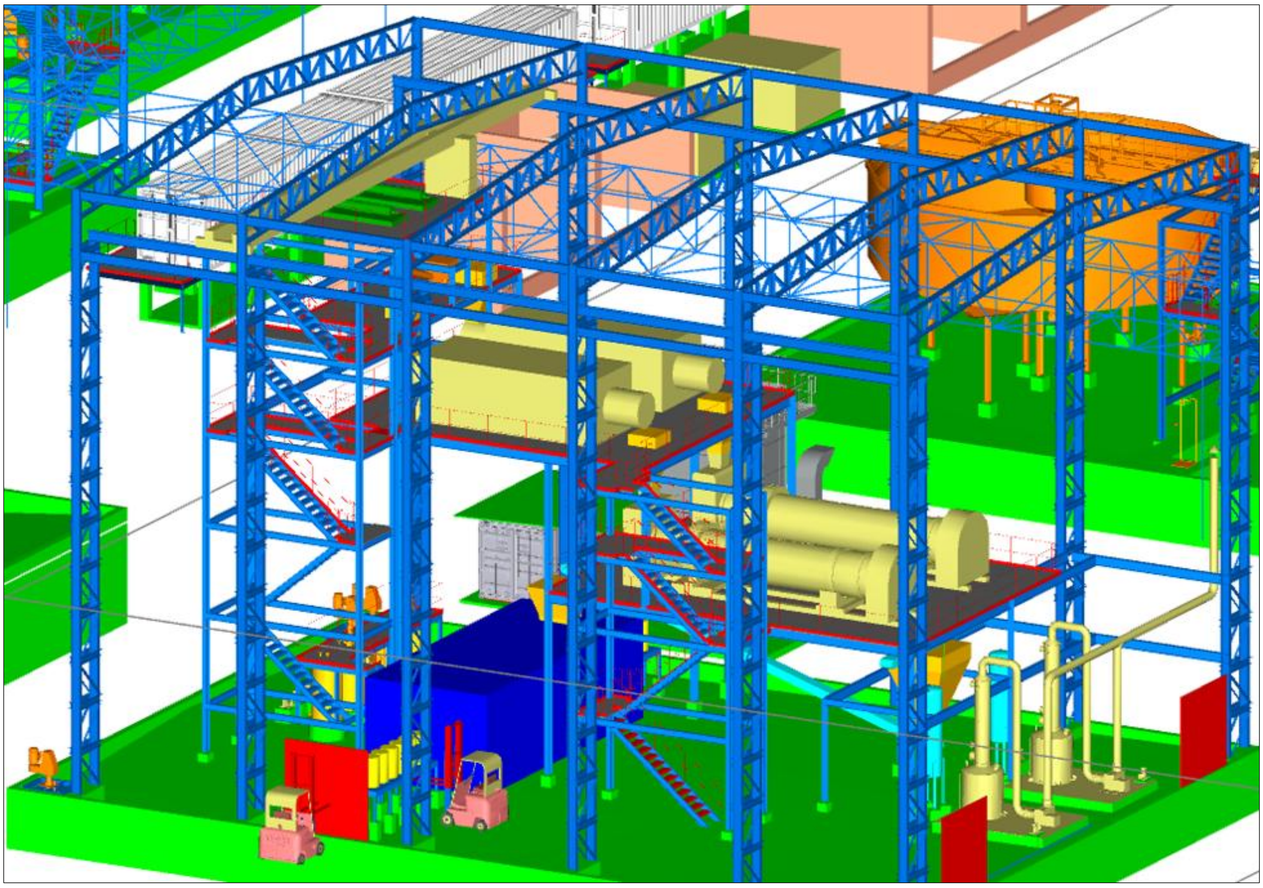

Figure 17-30: 3D Layout - U_30_8 product handling

17.4.2.9. Concentrated sulfuric acid

PFD Reference: C02003P120001001

The concentrated sulfuric acid used in the process is purchased and transported to the site. It is then transferred into two concentrated sulfuric acid storage tanks (C0200-TK-001/002) for bulk storage, using pumps (C0200-PP-001/002), with one serving as a duty pump and the other as a standby. The concentrated sulfuric acid is further transferred to the heap leach via transfer pumps (C0200-PP-005/006). On-demand, the acid is also pumped to the agglomerator (C0103-AD-001). The necessary acid for a minimum 24-hour leaching period is stored in the sulfuric acid storage tanks (C0200-TK-001/002). It should be noted that the acid demand varies between the different ore sources. The storage tanks currently offer between one and ten days of storage capacity, depending on the mix of ore being treated.

To prevent the concentrated acid from being diluted by moisture in the air, a sealed pot with a dryer (C0200-DR-001) is installed on the lid of each storage tank.

In the event of any concentrated acid spillage, it is collected in the concentrated sulfuric acid sumps (C0200-SM-001/002) and then pumped to the barren pond (C0104-PD-001) using the concentrated sulfuric acid spillage pumps (C200-SP-001/002).

Safety measures in this area include a safety shower (C0200-SH-001) and a wash point (C0200-WP-001), ensuring proper protocols for handling concentrated sulfuric acid and responding to any emergencies.

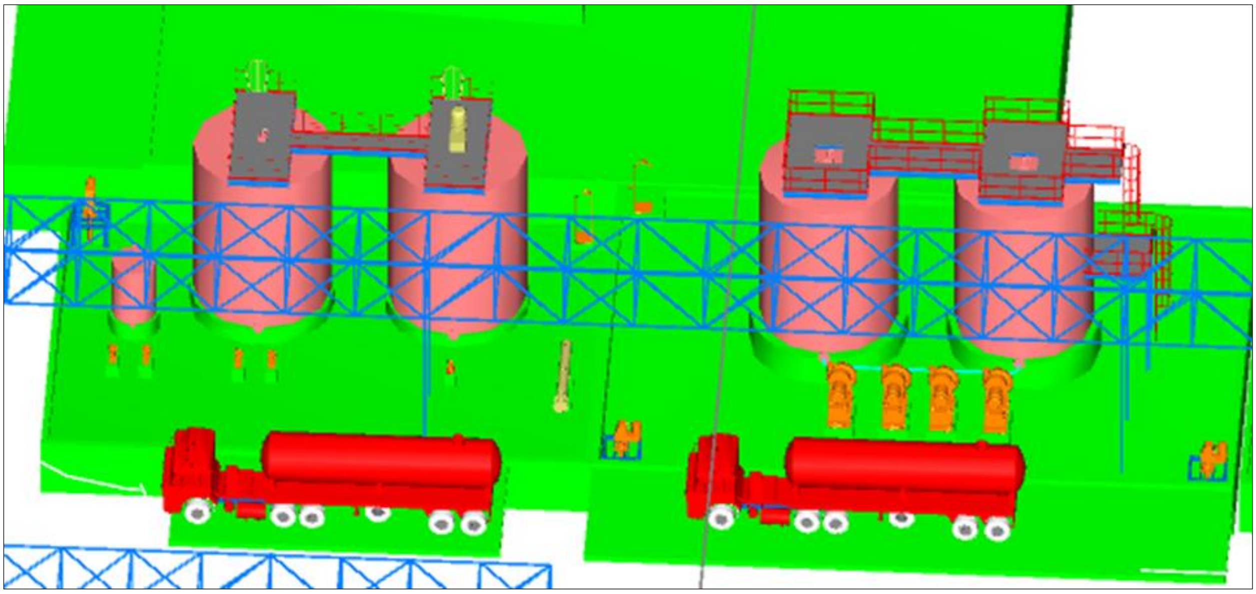

Figure 17-31: 3D Layout – concentrated sulfuric acid handling

17.4.2.10. Dilute sulfuric acid

PFD Reference: C05003P120001001

The elution processes require a one-molar sulfuric acid solution for use as an eluant. The one molar sulfuric acid is made up by diluting concentrated sulfuric acid with demineralised water in the dilute sulfuric acid make-up tank (C0500-TK-002).

Concentrated sulfuric acid is transferred from the holding. The automatic dilution water flow control valve opens to allow enough demineralised water into the dilute acid make-up tank. The dilution water and concentrated acid are ratio controlled to give one molar sulfuric acid in the make-up tank. The solution is continuously circulated through the dilute acid heat exchanger (C0500-HX-001) to remove the heat of dilution. Once a batch is completed it is transferred to the eluate tank (C0500-TK-003) from where it is transferred on demand to IX elution and resin regeneration using the dilute sulfuric acid distribution pumps (C0500-PP-004/005). These pumps will stop and start as required by the different circuits.

The iron wash solution makeup tank also receives dilute sulfuric acid from C0500-TK-002. Demineralised water is added under ratio control to produce a solution of 10 g/L sulfuric acid, which is pumped on demand to the loaded resin transfer vessel (c0400-TV-001).

Any dilute acid spillage is collected in the dilute sulfuric acid sump (C0500-SM-001) from where it is pumped to the barren pond (C0104-PD-001) using the dilute sulfuric acid spillage pump (C0500-SP-001).

A safety shower (C0500-SH-001) is provided for washing off any acid contact.

17.4.2.11. Sodium hydroxide makeup and distribution

PFD Reference: C05013P120001001

Sodium hydroxide is used in multiple parts of the process, including pH regulation in the uranyl peroxide precipitation process, neutralisation of sulfuric acid in the resin regeneration circuit, and use in the gas scrubber during final product handling.

The sodium hydroxide is delivered to the site as solid flakes in bulk bags. Upon arrival, it undergoes a batch process for makeup in the sodium hydroxide makeup tank (C0501-TK-001). The tank is filled with demineralised water to a predetermined level, and sodium hydroxide flakes are gradually added at a controlled rate, ensuring that the temperature remains within specified limits. The bags are lifted into the sodium hydroxide feed hopper-bag breaker (C0501-BB-001) using the sodium hydroxide makeup area hoist (C0501-HT-001). The bag breaker releases the flakes into the hopper, from where they are transferred to the makeup tank via the sodium hydroxide screw feeder (C0501-FD-001). The sodium hydroxide flakes are thoroughly mixed in the tank by the sodium hydroxide makeup agitator (C0500-AG-001) until completely dissolved.

The resulting sodium hydroxide solution is then transferred to various areas of application using sodium hydroxide transfer pumps (1310-PP-1994/1995), with one pump serving as the duty pump and the other as a standby.

In the event of spillage, a sodium hydroxide makeup area spillage sump (C0501-SM-001) collects the spill, and a spillage pump (C0501-SP-001) facilitates its return to the makeup tank (C0501-TK-001).

To ensure safety, a dedicated safety shower (C0501-SH-001) is installed in this area, providing an immediate response option in case of accidental contact with sodium hydroxide.

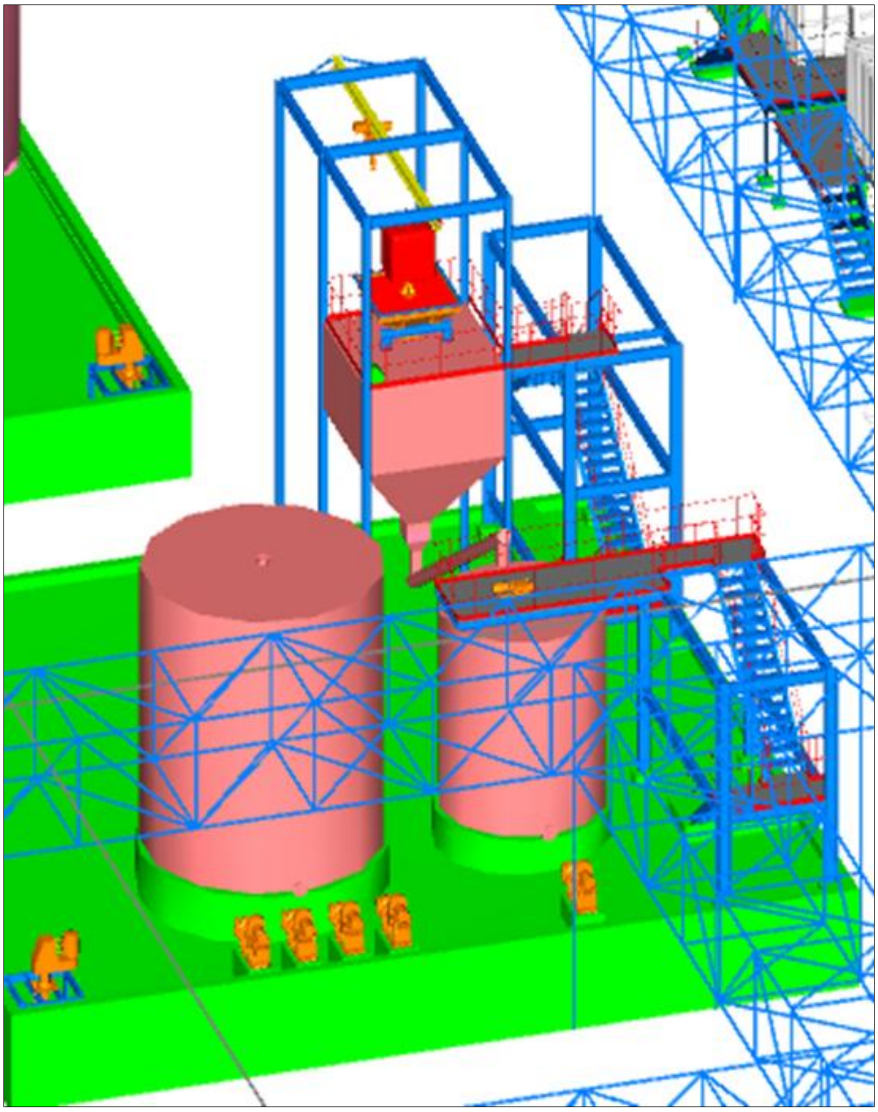

Figure 17-32: 3D Layout – sodium hydroxide make up

17.4.2.12. Hydrogen peroxide storage and distribution

PFD Reference: C05023P120001001

Hydrogen peroxide is a critical component in the precipitation of uranyl peroxide. It is delivered to the site as a 70 % (w/w) solution in bulk containers and stored in two hydrogen peroxide storage tanks (C0502-TK-001/002). The transfer from the delivery container to the storage tanks is facilitated by hydrogen peroxide offloading pumps (C0502-PP-001/002), with one pump designated as the duty pump and the other as a standby.

For the uranyl peroxide precipitation circuit, the 70 % (w/w) solution is further processed using two hydrogen peroxide dosing pumps (C0502-PP-003/004). Before introduction into the circuit, the 70 % (w/w) hydrogen peroxide is diluted with demineralised water to achieve a 20 % (w/w) hydrogen peroxide concentration. This dilution is performed using dedicated hydrogen peroxide dilution mixers (C0402-MI-002/003/004).

In the case of accidental contact with hydrogen peroxide, a safety shower (C0402-SH-003) is available for personnel to wash off the reagent promptly. This safety measure ensures a quick response to any potential exposure.

In the event of spillage, a hydrogen peroxide makeup area spillage sump (C0502-SM-001) collects the spill, and a spillage pump (C0502-SP-001) facilitates its return to the emergency pond (C0104-PD-005).

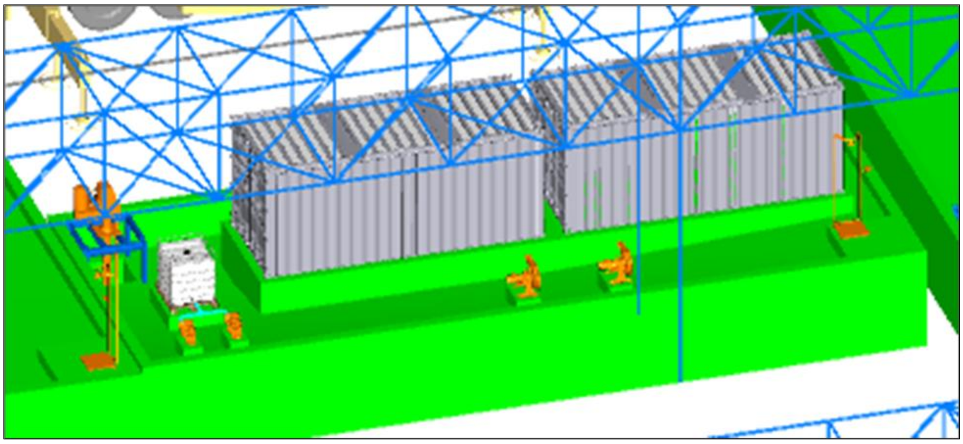

Figure 17-33: 3D Layout – hydrogen peroxide storage

17.4.2.13. Lime slaking and distribution

PFD Reference: C05033P120001001

Lime is delivered to the site in 1 t bulk bags and stored in a covered storage facility. The plant incorporates a bag off-handling facility, facilitating the unloading of bags from trucks into the storage area. The bags are then moved from the storage facility on a First-In-First-Out ("FIFO") basis to the lime-slaking plant.

The lime feed process begins with lifting the bags into the lime feed hopper-bag breaker (C0503-BB-001) using the sodium hydroxide makeup area hoist (C0503-HT-001). The bag breaker releases the dry lime into the mixing tank (C0503-TK-001), equipped with an agitator (C0503-AG-001). As lime is introduced to the slaking unit, process water is added in a ratio-controlled manner to ensure a consistent lime slurry density. The lime addition to the slaker unit is regulated based on temperature, maintaining it between 62 ºC and 72 ºC for optimal CaO utilisation.

The resulting slaked lime is then distributed to the gypsum precipitation circuit through the duty/standby slaked lime distribution pumps (C0503-PP-001/002).

To manage spillage, a lime-slaking area spillage sump (C0503-SM-001) is in place, collecting any spills that occur. The collected spillage is then pumped to the gypsum thickener using the lime-slaking area spillage pump (C0503-SP-001).

As a safety precaution, a safety shower (C0503-SH-001) is provided in this area to address any potential contact or exposure incidents.

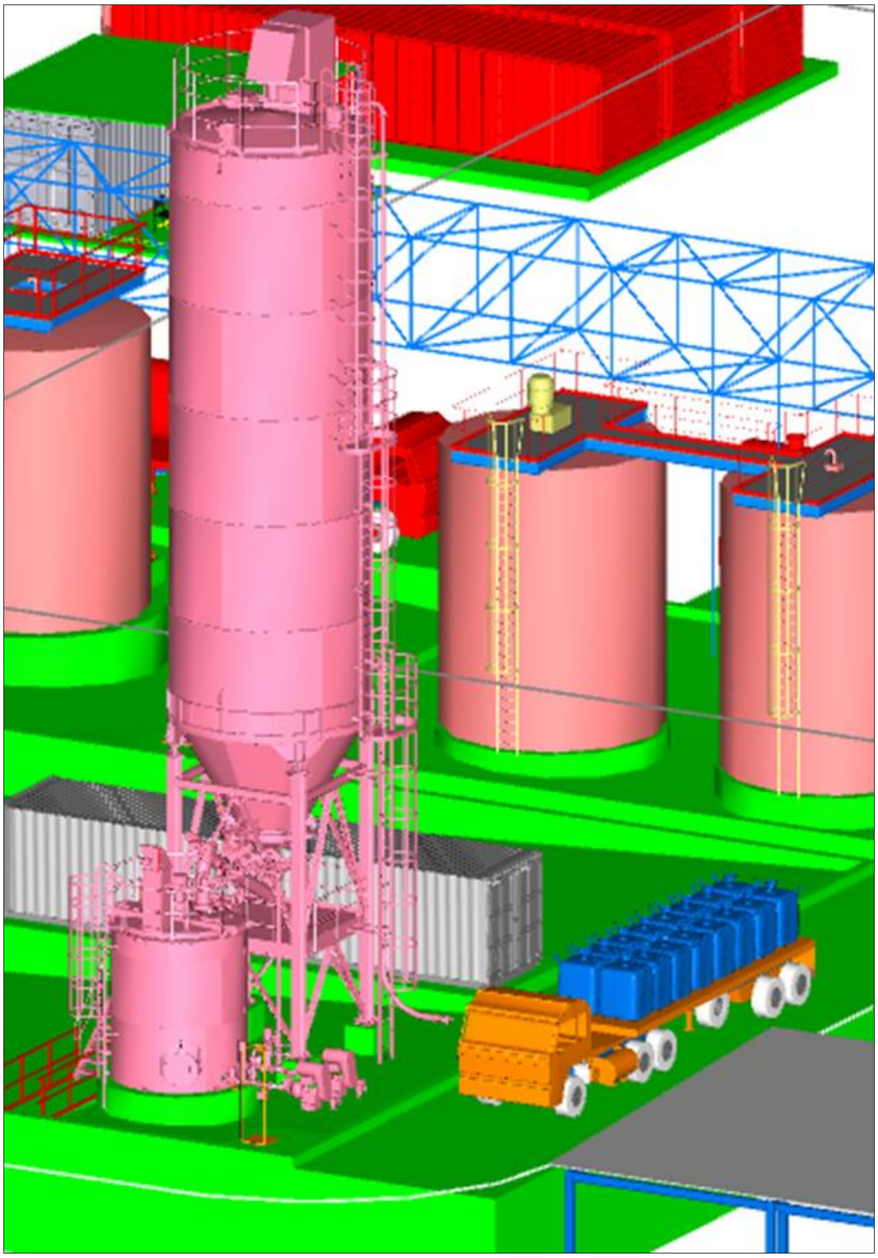

Figure 17-34: 3D Layout – lime slaking and distribution

17.4.2.14. Raw water and fire suppression water storage and distribution

PFD Reference: C03003P120001001 and C00053P120001001

Raw water from various sources is collected and stored in the raw water storage tank (C0300-TK-001). The raw water is pumped to either the process water storage tank (C0302-TK-001), gland seal water tank (C0303-TK-001) or potable water tank (C0301-TK-001) via pumps (C0300-PP-003/004), a duty and a standby.

The fire water system gets water supply from the raw water storage tank (C0300-TK-001). The fire water system consists of three fire water pumps: the electric fire water pump (C0005-PP-002), the diesel fire water pump (C0005-PP-003) and the jockey fire water pump (C0005-PP-001). The diesel required to drive the diesel-driven pump is gravity-fed from the diesel storage tank (C0005-TK-002). The jockey fire water pump (C0005-PP-001) keeps the fire water system pressurised. In the event of a fire, one of the main fire water pumps, electric or diesel is used to supply the fire water. The choice of the pump to be used depends on the availability of electrical power during the fire incident.

17.4.2.15. Demineralised water storage and distribution

PFD Reference: C06003P120001001

The demineralised water plant processes raw water to produce the demineralised water required for the elution and regeneration reagent make-up, packaging and precipitation processes. The demineralised water produced is stored in the demineralised water storage tank (C0600-TK-001) and is pumped to the acid tanks on demand using the demineralised water supply pumps (C0600-PP-001/002), a duty and a standby.

17.4.2.16. Gland service water storage and distribution

PFD Reference: C03033P120001001

Raw water pumped from the raw water storage tank (C0300-TK-001) is filtered in a sand filtration plant (C0303-FL-001/002), to remove excess solids that may be contained in the water. The filtered water is distributed to the various plant areas through a pressure-controlled ring main system using the filtered water supply pumps (C0303-PP-001/002), a duty and a standby.

Gland seal water spillage is pumped back into the gland seal storage tank (C0303-TK-002) by the gland seal area spillage pump (C0303-SP-001) after being collected in the gland seal area spillage sump (C0303-SM-001).

17.4.2.17. Process water storage and distribution

PFD Reference: C03013P120001001

The process water, vital for various aspects of the operation, is stored in the process water storage tank (C0301-TK-001). To maintain an adequate supply, the process water storage tank is periodically topped up with raw water, which is pumped from the raw water storage tank (C0300-TK-001). The distribution of process water to different areas is managed through a pressure control ring main, facilitated by the gland seal water supply pumps (C0301-PP-001/002), operating in a duty and standby configuration.

In the event of process water spillage, a system is in place to collect and manage it. The spillage is directed to the process water area spillage sump (C0301-SM-001), and from there, it is pumped back to the raw water storage dam using the process water area spillage pump (C0301-SP-001). This process helps maintain a controlled and efficient water management system within the facility.

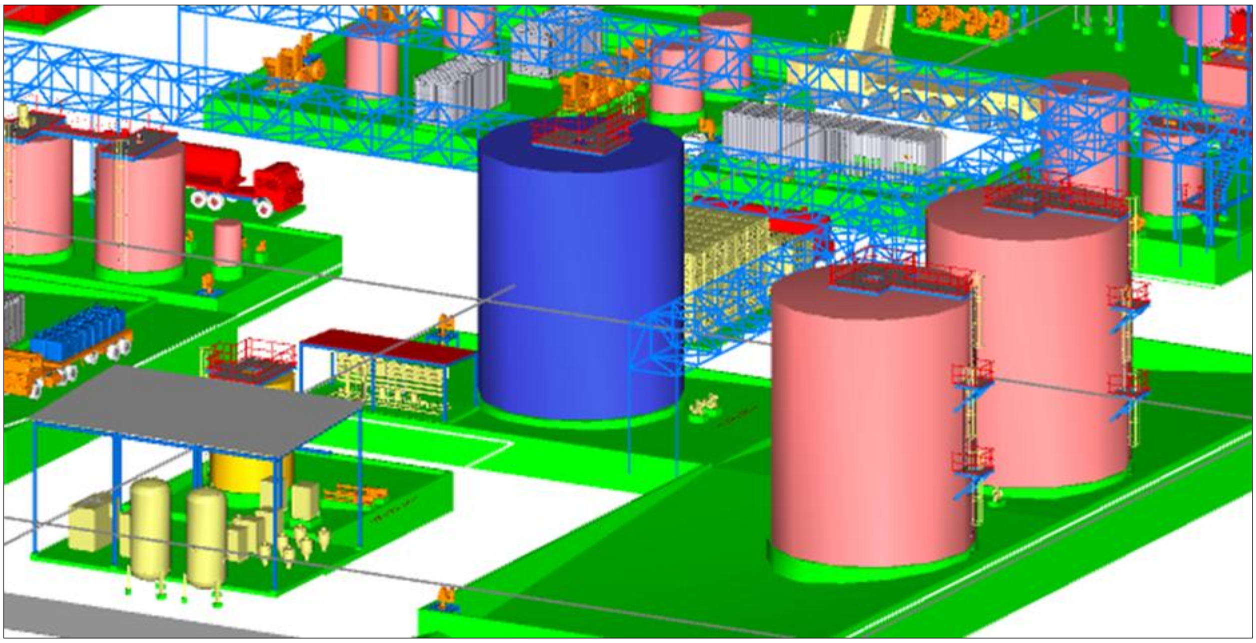

Figure 17-35: 3D Layout – water storage and distribution

17.4.2.18. Compressed air reticulation

PFD Reference: C03063P120001001

The plant relies on three types of compressed air: instrument air, filtration air, and general-purpose plant air. Dedicated compressors manage each category, with (C0306-CP-003) serving as the instrument air compressor and (C0306-CP-001) handling filtration and general plant air. (C0306-CP-002/0024) are standby compressors for instrument air and filtration/general plant air, respectively. Instrument air undergoes filtration through (C0306-FL-001/002/003) to remove contaminants, and the filtered air then passes through dryers (C0306-DR-001/002) to eliminate excess moisture. The treated instrument air is distributed to various plant areas from the instrument air receiver. Filtration and general-purpose plant air are delivered through the plant air receiver (C0306-AR-001) and distributed across the plant. Both types of compressed air are stored and distributed at a pressure of 10 bars to meet diverse operational needs.

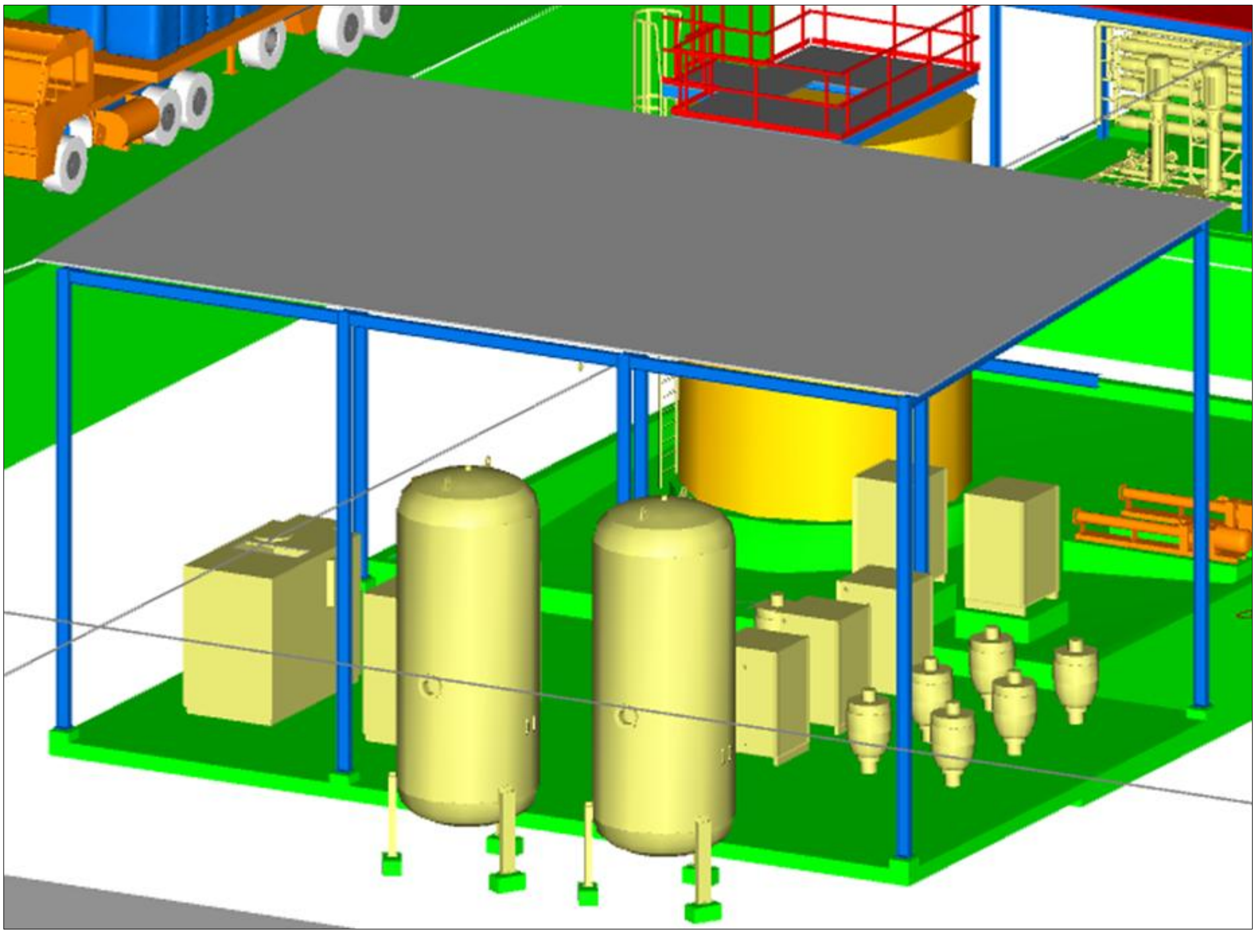

Figure 17-36: 3D Layout – compressed air

17.4.3.Mechanical engineering basis of design and estimation

This section details the basis of design and estimation for the mechanical equipment required for the FS. Note that this section also covers the mechanical engineering basis of design for the ore preparation stage detailed in Section 17.2 as it was part of the same scope of work.

Table 17-21: Key parameters

Client (Principal)	***GoviEx***
Site Location	*Zambia*
Project Type	*Major Greenfields*
Project Scope	*Indirect Field Costs (Man-hours and Consultants) and Direct Field Costs (Equipment Supply and Installation)*
Type of Estimate	*Definitive Feasibility Study Phase*
Estimate Accuracy	*DFS (Class 2)*
Contract	*Fixed Price*
Engineering Phase Duration (execution)	*52 weeks*
Construction Phase Duration (execution)	*69 weeks*
Commissioning Phase Duration (execution)	*13 weeks*

17.4.3.1. Summary description of scope

The scope of work ("SOW") includes engineering, procurement, construction, and management associated with the mechanical equipment within the battery limits.

The mechanical equipment contains major plant sections:

- Crushing, stockpiling, reclaiming, agglomeration, heap leaching (by others), IX concentration, reagents.
- Ion exchange, solvent extraction, neutralisation, ADU precipitation and packaging.
- Site infrastructure, facilities and services.

17.4.3.2. Battery limits

17.4.3.2.1. Incoming

- Bin grizzly at Central Plant for receiving ROM by dump truck
- Battery limit of electrical and instrumentation ("E&I") is the outgoing terminals of the 11 kV MV Switchgear supplied by others
- Potable water plant distribution pump flange
- Solution from dams/ponds interface with pumps (penstock discharge flanges, submersible pump stilling wells)
- Reagent supply in liquids, slurry and fine powder format from transport vehicle offloading hose connecting flanges
- Reagent supply in solids - Bulk bags hoist hooking points and drums handler hoist hooking points
- Fuel storage fill points.

17.4.3.2.2. Outgoing

- Agglomerated ore discharge conveyor discharge to heap leach
- Gypsum discharge
- Drummed product loaded with hoist onto pallets for storage and transport
- Sewage water plant intake sump
- Water distribution points
- Possible residual dust emissions after dust suppression
- Stack emissions to atmosphere.

17.4.3.3. Design and estimate basis

The plant design and equipment selections are in accordance with the process flow sheets, the mass and energy balances and the process design criteria.

The Mechanical Design Criteria established the minimum engineering specifications, standards and practices for the design, manufacture, and supply including inspection and testing, installation, and commissioning requirements of mechanical equipment. The Mechanical Design Criteria states or cross-references the basic specifications of plant performance and construction, and the engineering methodology, to which the project was executed including all mandatory requirements which impact the design, i.e., those specified by the client, and all applicable statutes and regulations or requirements needed to satisfy statutory bodies.

The level of design and mechanical estimation was according to the project estimation plan. The detailed guidelines for feasibility studies were followed, and it is believed that the Mechanical Equipment estimate meets the required accuracy levels with inconsequential issues due to changes at late stages during the study.

These guidelines and procedures for preparing the project cost estimate were in accordance with SGS Bateman Standards for a Class 2 FS. This closely conforms to both the SGS Bateman Estimator's Best Practice Guide PCNG-0920-002 Rev 1 definition for a Class 2 estimate classification and the Association for the Advancement of Cost Engineering International (AACEI) Class 2 estimate classification guidelines.

All tenders from the market were commercially and technically adjudicated and recommendations were prepared in Technical Bid Evaluations ("TBE") and Commercial Bid Adjudications ("CBA"). These recommendations formed the basis of pricing.

Equipment supply enquiries were issued through the SGS Procurement and Contracts department with SGS Standard Procurement Terms as the basis for the Muntanga Uranium's terms and conditions. The structural, Mechanical, Platework and Piping ("SMPP") construction contract was prepared based on the FIDIC red book, while the site fabricated tanks contract included detailing design, and was based on the FIDIC yellow book terms and conditions.

17.4.3.4. Mechanical package summary

Pricing of mechanical equipment was based on the requisitions for fixed and firm pricing for the items identified from the mechanical equipment list developed from the initial process flow diagrams ("PFDs"), subsequent piping and instrumentation diagrams ("P&IDs"), plant layouts and general plant arrangement drawings. Some package suppliers, where extensive design work was required to submit a tender, responded with FS -level budgetary quotations and excluded the without comprehensive requested detailed design information for this phase.

Enquiries were issued to multiple vendors however, in some cases with specific technology supply packages, only a single supplier responded suitably.

mechanical equipment equipment packages, shown in the table that follows, were issued to the market for the FS as "Formal" comprehensive technical enquiries with all specifications and schedules. "Short" form packages included only a scope specification and associated data sheets and PFD drawings where applicable. Small value packages or off-the-shelf items were based on similar recent database prices "Estimation".

Table 17-22: List of mechanical procurement packages

Number	Description	Tags	Type
M001	Agitators	14	Formal
M002	Agglomerators	4	Formal
M003	Belt Magnet & Metal Detector	8	Short
M004	Column Adsorption & Elution	6	Formal
M005	Compressed Air	30	Formal
M006	Conveyor	30	Formal
M007	Crane and Hoist	18	Formal
M008	Crusher	2	Formal
M009	Dryers Acid Tank	4	Short
M010	Dust Extraction	4	Short
M011	Dust Suppression	11	Short
M012	Equipment Lab	0	Short*
M013	Equipment Workshop	0	Short*
M014	Feeder Apron	8	Formal
M015	Feeder Vibrating	6	Formal
M016	Filter Press	2	Formal
M017	Filter Seal Water	6	Short
M018	Filter Water & Solution	16	Short
M019	Fire Protection	12	Formal
M020	Grasshopper (SRK)	2	Not in scope
M021	Heat Exchanger	2	Short
M022	Irrigation (SRK)	12	Not in scope
M023	Mineral Sizer	4	Formal
M024	Mixer Inline	6	Short
M025	Plant Calcining	26	Formal
M026	Plant Demin & Water Treatment	3	Short
M027	Plant Flocculant	22	Short
M028	Plant Lime	4	Short
M029	Plant Nanofiltration	18	Short
M030	Plant Peroxide	7	Short
M031	Plant Sodium Hydroxide	7	Short
M032	Pump Chemical	88	Short
M033	Pump Dosing	0	Short#
M034	Pump Peristaltic	7	Short
M035	Pump Spillage	52	Short
M036	Pump Submersible	8	Short
M037	Pump Centrifugal	26	Short
M038	Rock Breaker	2	Formal

Number	Description	Tags	Type
M039	Safety Shower	29	Estimation
M040	Sampler Hammer	2	Short
M041	Sampler Solution	7	Short
M042	Screen Vibrating	5	Formal
M043	Stacker & Reclaimer (SRK)	7	Short
M044	Tank Shop Fabricated	10	Formal
M045	Thickener and Clarifier	18	Formal
M046	Vessel Transfer	4	Formal
M047	Weightometer	22	Short
M044B	Tank Shop Fabricated	8	Formal
M025B	Packaging	13	Formal
X-M001	SMPP	147	Formal
X-M002	Tanks - Site fab	29	Formal

17.4.3.5. Mechanical equipment packages

The general approach for mechanical packages was to:

- Prepare detailed packages with tender SOW, technical specifications, battery limits, data sheets, drawings, and pricing schedules as identified on the Mechanical Equipment List (“MEL”) based on the PFD
- After an internal review cycle by relevant discipline, updated and approved packages were issued to Procurement to add standard procurement commercial conditions, and approach short-listed suppliers by issuing enquiry package
- Following the tender period suppliers submitted tenders on the requested closing dates driven by the DFS phase planning schedule. On request, procurement granted a closing date extension
- TBE was done to assess compliance to issued specifications on submissions and issued for internal review, while clarifications were sent to suppliers. CBA were done on technically acceptable offer recommendations. A final selection of the most suitable supplier was agreed to, following approval of the CBA. Tender information was incorporated in the MEL and distributed to other disciplines and Estimator
- Clarifications were mainly requested on critical issues and focused on the leading offer that would have significant cost impacts, concurrent with the internal review and procurement adjudications
- Data Base Estimates were submitted for only minor items not included in the procurement packages issued, or supplier omissions, based on estimated and escalated budget costs of previous projects.

The MEL was aligned with the latest PFD and P&ID and formed the basis for procurement packages, electrical load lists, plate work schedules, and construction BOQs.

Standard specifications issued with formal packages include mainly GS-5 general mechanical equipment specifications, GS-109 packaging specifications, and GS-11-1 protective coatings specifications. Detailed paint coating systems for dry and wet areas were incorporated in the Mechanical Design Criteria and issued mainly to formal vendor plant supply packages. Short-form procurement packages requested only suitable supplier standard coating systems.

Comprehensive supplier document and drawing requirements (“SDDR”) were generated for formal packages, while short-form packages request suppliers to allow for documentation typically supplied for execution projects.

17.4.3.6. Basis of design

Drawings were generated for DFS-level requirements, focusing on the holistic layouts for optimal process material flow, operations, access safety, construction, and major interfaces with other disciplines' requirements. The activities detailed below were carried out during the design phase of the FS:

- Development of design criteria
- Production of equipment specifications.
- Input to P&IDs for mechanical items
- Briefing of draughtsmen and engineering input into drawing office
- Calculation and checking of pump heads, conveyors etc.
- Development of design concepts
- Design allowances of special chutes, plate-work details and tanks including calculation of fabrication masses based on preliminary design estimates
- Selection of equipment to meet duty and other requirements

- Co-ordination of equipment into cohesive, effective systems
- Line sizing checking for equipment package interfaces
- Maintenance of equipment lists
- Technical input to enquiry and purchase requisitions
- Technical evaluation of tenders
- Technical clarification with suppliers for significant discrepancies
- Review of supplier information for compliance with design requirements
- Co-ordination of other disciplines in package unit designs
- Drawing review and approval internally and with Client
- Review and approval of Mechanical Layouts for other disciplines to proceed with design and detailing
- Input to flowsheet reviews and hazard and operability ("HAZOP") studies.

17.4.3.7. Estimate exclusions

- No detailed design was performed during the study
- No mechanical provision was made for any item not appearing on the MEL and POP packages or quantity submitted along with the estimate
- Provision for a fire consultant.

17.4.4. Piping engineering basis of design and estimation

The following documents developed during the FS form part of the piping design cost estimate development.

Table 17-23: Piping engineering basis of design and estimation

Item	Activity description	Developed Y/N	Deliverable Y/N
2.1	Project set up – Codes, standards, procedures, filing system etc.	Y	N
2.2	Project set up – PUMA	Y	N
2.3	Discipline strategy document	N	N
2.4	Piping design criteria	Y	Y
2.5	Fluid code list	Y	Y
2.6	Preliminary piping material line classes	Y	N
2.7	Input into P&ID's	Y	Y
2.8	General piping supply specifications	Y	Y
2.9	Overland Piping Layout Input	N	N
2.10	In-plant piping layout input	Y	N
2.11	Piping general arrangements input and review	N	N
2.12	Isometric drawing review and input	N	N
2.13	Piping tie-in / Battery limit schedule	Y	Y
2.14	Piping line list	Y	Y
2.15	Piping valve list/Bill of quantity ("BOQ")	Y	Y
2.16	Estimated Special piping item list/BOQ	Y	N
2.17	Pipe support schedule	N	N
2.18	Bolt and gasket schedule	N	N
2.19	Preliminary valve data sheets	Y	N
2.20	Preliminary SPI data sheets	Y	N
2.21	Preliminary piping bulk BOQ for large bore piping	Y	Y
2.22	SMPP technical adjudication	Y	Y
2.23	Manual valve technical adjudication	Y	Y
2.24	SPI technical adjudication	Y	N

17.4.4.1. In-plant piping bill of quantity development

The in-plant piping design is based on the project P&IDs Rev D including the updated Rev F P&IDs (updated after the enquiry phase). The quantities were developed from a preliminary 3D piping model routed as per the preliminary plant layout. Only lines 80NB and above were routed with basic fittings and flanges. All BOQs were generated per line and materials based on the preliminary piping line class specifications selected as per the FS fluid List. Small bore utility piping was not modelled and was estimated as 15 % of the piping cost.

All manual and automated valves shown on the P&IDs were captured in the valve list and formed part of the valve enquiry. The design of the valves was based on the preliminary valve data sheets selected from standard SGS Bateman valve data sheets and needs tomust be finalised during the next phase of the project.

Major special piping items were allowed such as bellows and hoses as per the preliminary SPI schedule. Standard materials were used based on the commodity conditions listed in the fluid list. Other SPI items were allowed for as part of SPI, site run and infrastructure allowance. The materials of all special piping items must be confirmed during the next phase of the project.

The primary pipe supports were quantified, by using the pipe length multiple by allowable pipe spans per size, as per the standards SGS Bateman norms. The quantity of required supports was then converted into kilograms of steel and formed part of the SMPP tender.

No piping insulation was quantified and was allowed as 1 % of the overall piping cost.

17.4.4.2. Overland piping bill of quantity development

PLS, piping from the heap leach to the plant and barren return piping are included under process piping. No other overland piping was allowed during this phase.

17.4.4.3. Infrastructure piping bill of quantity development

Infrastructure piping for potable water, fire water and air forms part of the additional allowance noted as "SPI, Site-Run and Infrastructure allowance".

17.4.4.4. Battery limits (incoming and outgoing)

Reference must be made to the battery limit schedule and be read in conjunction with the project P&IDs that indicate the battery limits between the client and SGS Bateman design. All physical limits are within the plant perimeters and no overland piping was allowed for.

17.4.4.5. Piping procurement packages

17.4.4.5.1. Manual and automated valve supply package

The Valve enquiry went out to four-valve vendors. TVC was selected to be technically the preferred supplier for the DFS. Due to competitive pricing, the package will be split into two between TVC and BRAY.

17.4.4.5.2. SPI supply package

The SPI enquiry went out to two vendors with the selected supplier being Truco Rubber. The supplier is well known in the industry and supplied SGS Bateman successfully on previous projects. The enquiry consisted of bellows, couplings, spray shields and hoses.

17.4.4.5.3. Piping supply and installation

The in-plant piping supply and installation costs formed part of the project SMPP package. The technical evaluation for the piping portion indicated the acceptance of two Vendors with technical clarifications, recommendations, and adjustments as noted therein. The quantities of the selected SMPP supplier were adjusted to suit the latest process changes.

17.4.5. Civil engineering basis of design and estimation

The basis of estimate for civil, bulk earthworks and infrastructure explains the SOWs and how the quantities were estimated and derived. Note that this section also covers the civil engineering basis of design for the ore preparation stage detailed in Section 17.2 as it was part of the same scope of work. The preliminary civil, bulk earthworks design, infrastructure design, MTOs and BOQs were prepared for each area based on the below:

- DFS plant layout
- Developed project design criteria
- Civil and earthworks specifications and standards
- DFS civil and earthworks preliminary designs
- Structural steel specifications and standards
- DFS structural steel preliminary designs.

The SOW for civils, earthworks and infrastructure includes all the plant areas, as per the plant layout developed. Preliminary designs were developed for each plant area, and the MTOs and BOQs were developed, and issued for enquiries in separate packages: civil works, bulk earthworks, infrastructure plant buildings, plant roads and containerised buildings package. From the tenders that were received, a preliminary cost estimate was developed. The cost estimate developed includes all labour, materials, transport to the site, storage on site, and site construction.

The SOW for structural steelwork includes all the plant areas, as per the plant layout developed. Preliminary designs were developed for each plant area, where MTOs and BOQs were developed, and issued for enquiries in the SMPP package. From the tenders that were received, a preliminary cost estimate was developed. The cost estimate developed includes all labour, material, fabrication, corrosion protection, transport to site, storage on site, and site installation.

The structural steelwork material selection was based on the suitability of materials, durability, maintenance, constructability, costs, industry practices and the nature of the interface environment.

17.4.5.1. Quantity and cost development

The quantities developed were based on the FS mechanical model layout and drawings produced for each plant area. MTOs were taken, and BOQs developed from the preliminary design. BOQs that were developed were issued for the SMPP package, civil works package, bulk earthworks package and buildings packages. All cost estimates were developed from the tender documents received.

17.4.5.2. Procurement packages

The civil, bulk earthworks and infrastructure were executed as individual packages. The structural steelwork was executed as part of the SMPP package. Enquiry documents were issued to the market, tenders were received, and technically and commercially adjudicated to select the most suitable contractor's price for the costing of the FS.

17.4.5.3. Engineering analysis

17.4.5.3.1. Material selection

The material selection for the FS was based on the suitability of materials, durability, maintenance, constructability, costs, and industry practice.

17.4.5.3.2. Design approach

The following design approach was used:

- The preliminary design was based on the mechanical model layout and drawings provided
- Where feasible, the following general strategic principles have been adopted
  - Use of standard designs from the SGS Bateman database
  - Use of previous similar designs
  - Use of structural steelwork standard designs from the SGS Bateman database
  - Use of structural steelwork for previous similar designs.

17.4.5.3.3. Material take-offs build-up

All civil, bulk earthworks and infrastructure MTOs were prepared from the preliminary design prepared from the mechanical model and drawings provided.

All bulk earthworks were measured from the balancing of the cut and fill, and the recommendation of the Geotech report.

All concrete works such as foundations, were based on preliminary foundation sizes required. All roads, stormwater, dams, and sewers were based on the plant layout developed. All structural steelwork materials take-offs were taken from the preliminary design prepared, and the mechanical model and drawings provided.

Open grid flooring was measured in square metres from the mechanical model and drawings provided.

Standardised hand-railing was measured in linear metres from the mechanical model and drawings provided.

The quantities estimate excludes any contingency for items/events of an unforeseen nature and design growth allowance.

17.4.5.3.4. Assumptions

The preliminary designs prepared, estimated sizes of structural elements, and consequent material take-offs are a fair representation of quantities required at this phase of the project.

The execution of the project shall be based on South African National Standards ("SANS").

The employer will provide an adequate laydown area to suit the contractor's requirements.

The Geotech report provided is a fair representation of the soil conditions on-site.

Quality soil construction materials, as well as quality aggregate for concrete, will be available.

17.4.5.3.5. Risks

The quantities may change during the execution phase when the certified drawings and loads are available.

17.4.5.3.6. Engineering documents

Where reference is made to a document, specification, code or standard, the reference will be taken to mean the latest edition of the document, specification, code or standard, including the latest addenda, supplements, and revisions thereto, as at the base date for the Project.

The works shall conform to the documents prepared during the FS such as the design criteria, structural steel specifications, concrete specifications, bulk earthworks specifications, SANS code of practice, and the structural steel, the civil, bulk earthworks and infrastructure bill of quantities prepared.

17.4.6. Electrical engineering basis of design and estimation

This section serves to describe the methodology upon which the electrical engineering discipline used to develop the engineering, design and the associated cost estimate for electrical infrastructure and systems associated with Project FS. Note that this section also covers the electrical engineering basis of design for the ore preparation stage detailed in Section 17.2 as it was part of the same scope of work.

The cost estimate philosophy is based on the following:

- Preliminary and historical information
- SGS Bateman electrical specifications and standards
- Electrical Design Criteria, M7610-E670-001
- Mechanical Equipment List, M7610-M810-001 from which the Electrical Load List, M7610- E831-001 was developed.

The design criteria were based on SGS Bateman templates with some SANS replaced with international standards ("IEC") where possible. Standard SGS Bateman specifications and standards were utilised to provide a cost-effective, safe and sustainable design basis for this estimate.

The basis that follows was used to derive the project's electrical engineering costs estimate. The same basis was used to derive the Execution Manhour Estimate – Electrical Discipline.

The electrical engineering SOW was to quantify the electrical requirements for the design of electrical distribution system architecture inclusive of equipment, installation and commissioning.

The electrical engineering design, equipment, installation, and commissioning scope were developed in accordance with the Electrical Design Criteria in line with the Preliminary Single Line Diagram.

The electrical engineering scope was inclusive of MV cabling, preliminary plant MV switchgear, plant LV containerised substations, miniature substations, LV cabling, motor field stop-start stations, lighting and small power, earthing and lightning protection, electrical installation and erection contract.

The contents of both MV and LV container substations included switchgear(s), battery tripping units, motor control centre(s) ("MCCs"), uninterrupted power supply ("UPS"), fire protection, heating, ventilation and air conditioning ("HVAC"), lighting and small power and free space provision for free issued control and instrumentation ("C&I") equipment. The C&I control room was included in the LV container substation package.

Bulk quantities such as cable racking were based on cable routing following pipe racks wherever possible.

Area and street lighting and small power quantities were estimated using the plant layout and assuming coverage requirements. Substation placement detail design and quantities will require updating in the detailed design phase.

The following electrical distribution assumptions were made:

- Bulk power supply infrastructure and MV switchgear shall be supplied by others
- The power supply (as per above) shall have sufficient electrical capacity and physical space for supplying the project's electrical loads
- The main electrical point of supply for the plant power distribution shall be at the plant main substation MV feeder downstream terminals
- The fault levels were assumed to be 20 kA at 11 kV for cable calculations
- Average low voltage cable lengths were assumed to be 25 m from transformer to MCC Incomers and in trefoil configuration
- Average low-voltage cable lengths were assumed to be 100 m from MCCs to plant loads. The volt drop considerations were according to the design criteria specification
- The plant infrastructure buildings i.e., admin, medical facilities, crib rooms, change houses, stores, workshop, etc. are to be fed from the mini substations
- The plant infrastructure buildings i.e., admin, medical facilities, crib rooms, change houses, stores, workshop, etc., internal power and lighting distribution is on the civil and structural contract package scope hence no costs.

17.4.6.1. Estimate bases

Electrical procurement and contract packages were grouped into similar types to minimise the number of packages. The below procurement and contract packages were issued to the market for multiple bidders to price.

17.4.6.1.1. Group 1: Equipment supply

- M7610-E001 - MV Switchgear, BTU and RMU (excluded from estimate, Utilink to finalise the package)
- M7610-E002 - LV Container Substations and control room
- M7610-E003 - Distribution Transformers & Mini–Substations.

17.4.6.1.2. Group 2: Supply and install contracts

- M7610-X-E001 - Electrical Protection Consultant
- M7610-X-E002 - Earthing and Lightning Protection
- M7610-X-E003 - Electrical and Instrumentation.

The costs for the above packages were obtained through the below process:

- The LV container substations and control room budget costs were obtained through budget quotations with technical and commercial evaluation of the received bids
- The distribution transformers, and miniature substations budget costs were obtained through budget quotations with technical and commercial evaluation of the received bids
- The protection specialist services contract budget cost was obtained through a single budget quote with a technical and commercial evaluation of the received bid
- The earthing and lightning protection contract budget costs were obtained through quotations with a technical and commercial evaluation of the received bids.
- Electrical and instrumentation contracts, including equipment and bulk supply, installation and commissioning budget costs were obtained through quotations with technical and commercial evaluation of the received bids.

17.4.6.2. Battery limits

The following electrical battery limits apply:

- Feeder breakers outgoing terminals of the plant main MV substation (11 kV MV Switchgear supplied by others)
- The top of the civil plinths
- The field connections as stated in the cable schedules
- The structural steel to which supports, and electrical equipment shall be attached
- Incomer breaker of the plant buildings, maintenance facilities such as admin, medical facilities, lab, firehouse, crib rooms, change houses, stores, workshop, etc.
- Incomer breaker of the vendor package such as the RADOS package.

17.4.6.3. Qualifications

The following qualifications apply:

- Only terminations to the incomer breaker allowed for infrastructure buildings, maintenance facilities internal lighting and small power – admin, medical facilities, crib rooms, change houses, stores, workshop, etc. as there were prices on civil and structural contract packages

- Electrical power and control cables, cable racking, earthing and lightning protection system, motor field stop-start stations, welding socket outlet, junction boxes, lighting and small power are costed on the installation contract costing
- A preliminary overall layout plan of the equipment and main rack routing was used to prepare a detailed figure for cable runs.

17.4.6.4. Consultants and sub-contractors

No electrical engineering consultant design inputs were considered during this phase of the project.

Pricing for protection system, earthing and lightning protection services is included for the execution phase of the project.

17.4.7. Control and instrumentation engineering basis of design and estimation

The purpose of this section is to describe the C&I strategy, design and the basis of costing thereof used in the estimate development. Note that this section also covers the C&I engineering basis of design for the ore preparation stage detailed in Section 17.2 as it was part of the same scope of work.

The costing of the C&I works, as a minimum, is based on the following fundamental documents:

- Plant layout drawings
- P&IDs
- Project specifications and SOW documents
- C&I design criteria
- C&I drawings and schedules.

17.4.7.1. Instrumentation

The post HAZOP P&IDs, and corresponding layouts, were used for all indexes and reports.
An Instrument index was compiled from the P&IDs.

All other schedules, drawings and BOQs have been based on post-HAZOP P&IDs.

17.4.7.2. Process control system

The instrument index and electrical drive index were used to create the input/ output ("I/O") and electrical interface counts. Possible vendor package interface I/O has not been allowed. Vendor packages have only been allowed if indicated on P&IDs. These instruments and I/O are only indicative and will be finalised during execution. These quantities, together with the plant layout were used to develop a basic (preliminary) process control system ("PCS") and Network Topology drawing to quantify the PCS effort. A spare I/O of 25 % of the design quantity was allowed.

17.4.7.3. Site installation and construction

Equipment installation quantities were derived from the instrument index, BOQ, layout drawings and PCS topology drawing.

A cable schedule was compiled, thus cable and termination MTOs were verified per instrument/ installation point for the BOQ. Cable lengths however were estimated. Cable racking MTOs were estimated per plant area. Detailed racking drawings were not done. Valve air manifolds, tubing and fittings were also estimated.

17.4.7.4. Equipment cost

Procurement packages were created and priced via formal enquiries (three vendor tenders where possible) for the following C&I items.

17.4.7.4.1. Group 1: Equipment supply

M7610-I-001 - Field Instruments (Analyser, density, flow, level, pressure and temperature).

17.4.7.4.2. Group 2: Supply and install contracts

- M7610-X-I001 - Security, Access Control, IT & Communications (Telephones)
- M7610-X-I002 - Process Control System.

- M7610-X-E003 - Electrical and Instrumentation Construction/Installation package submitted by Electrical.

Quotes were received for the following C&I items:

- Field Instruments: Two tenderers priced on field instruments; both were found to be technically compliant
- PCS: Full tenders were sent to four suppliers of which only one returned a quote
- Electrical and Instrumentation Construction/Installation package submitted by Electrical. Full tenders were sent to six suppliers of which only two returned quotes
- Security, Access Control, Information technology (“IT”_ & and Communications communications (Telephonestelephones). Full tenders were sent to five suppliers of which only one returned a quote.

Database pricing was not utilised since budget prices were obtained via enquiry tenders for pricing.

17.4.7.5. Battery limits

The following C&I battery limits apply:

- Flanges on tanks
- Top of concrete plinths
- Foundations
- Supporting structures, etc.

17.4.8. Drawing office engineering

- An overall 3D model was generated, and it was built up from individual 3D models from each process area of the plant
- 3D models were generated for all process areas and infrastructure areas of the plant
- 2D mechanical layouts were generated from the 3D models for MTO purposes
- 3D piping models were generated after the completion of the mechanical models for MTO purposes.

18. Project infrastructure

18.1. Primary access roads

18.1.1. Summary

The Project's primary access roads ("PARs") connect plant and mine sites to the nearest national road. They will be used during construction and operation and will be used by local traffic. This SOW includes access roads to the reconciliation action plan ("RAP") sites and a detour road at Gwabi. There are three PARs:

1. Muntanga
2. Dibbwi and
3. Gwabi.

Njame is immediately adjacent to the D500 and is accessed directly from this road.

Early feedback from the environmental and social survey works confirmed that "un-gazetted" local roads could be used by the Project if these are adequately upgraded and subject to suitable traffic controls and design to allow both mine and local traffic to use the road safely. Therefore, the strategy to follow the alignment of existing roads where possible, formed the basis of the assessment.

In summary:

- Significant trade-off work was carried out to define the best route for the Muntanga access road (Figure 18-1) to minimise the impact on the village of Hachibozu and successfully cross the Machinga River and also to re-route the existing local road around the mining areas of Muntanga and Dibbwi East.
- The Dibbwi road (Figure 18-1) follows the alignment of a track which was developed by GoviEx to access the site for drilling from Muntanga and connects the Dibbwi mine site to the Muntanga access road

The study was conducted in two phases: Phase 1 where options were reviewed and compared at scoping or PFS level, and Phase 2, which comprised the FS design and cost estimation. A detailed assessment of the Machinga River bridge was carried out with support from ASD Engineers based in Lusaka.

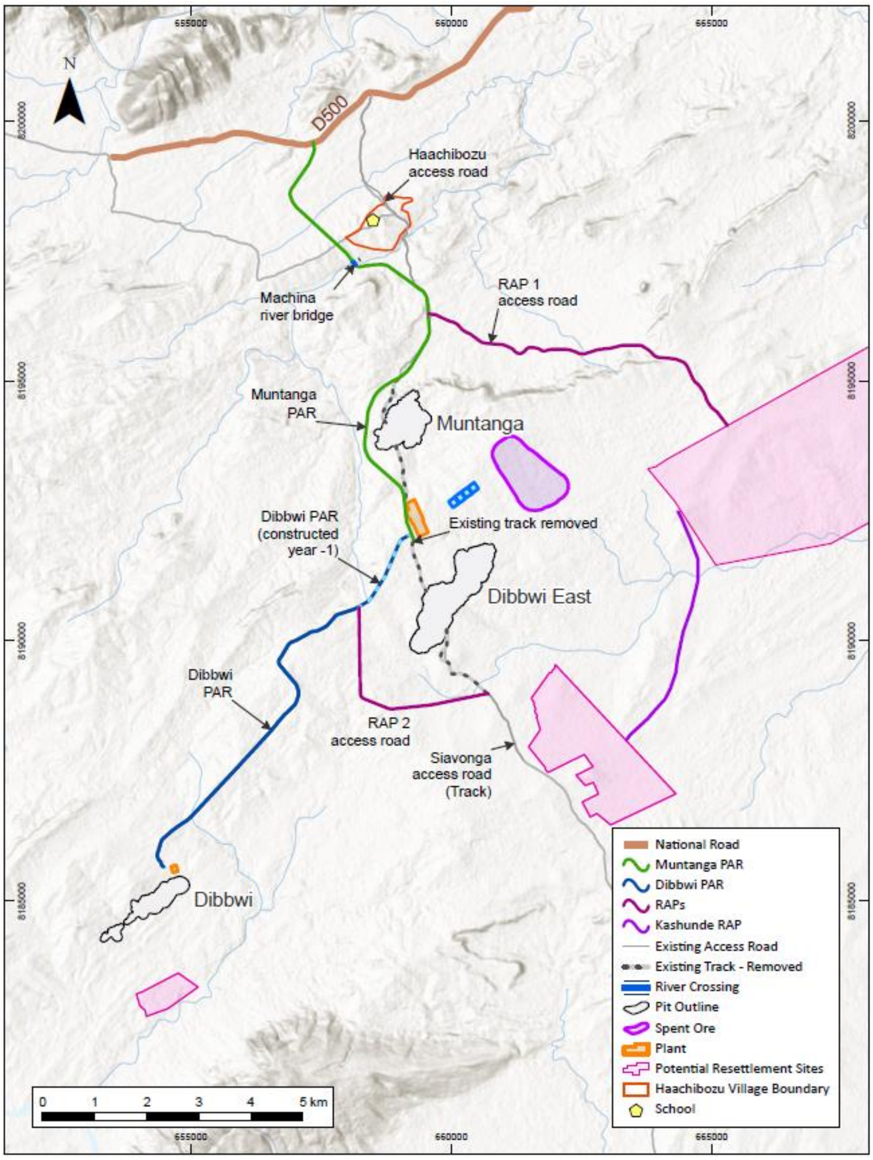

Figure 18-1: Access roads for Muntanga, Dibbwi East and Dibbwi

18.1.2.National road infrastructure

The Road Transport and Safety Agency is a road and motor vehicle government regulation body in Zambia. The Road Development Agency is a statutory body established through the Public Roads Act No. 12 of 2002 that provides for the care, maintenance, and construction of public roads in Zambia. This includes all gazetted roads and the Core Road Network ("CRN"). The CRN is split into Inter-Territorial Roads ("T-Roads"), Territorial Roads ("M-Roads"), District Roads ("D-Roads"), and Urban ("U") and Primary feeder ("PF") roads. Figure 18-2 shows the T2 road to Chirundu, which is proximal to Gwabi, the M15 road and D500 (referred to as locally as the "Bottom Road"), which are proximal to Njame, Muntanga, Dibbwi and Dibbwi East.

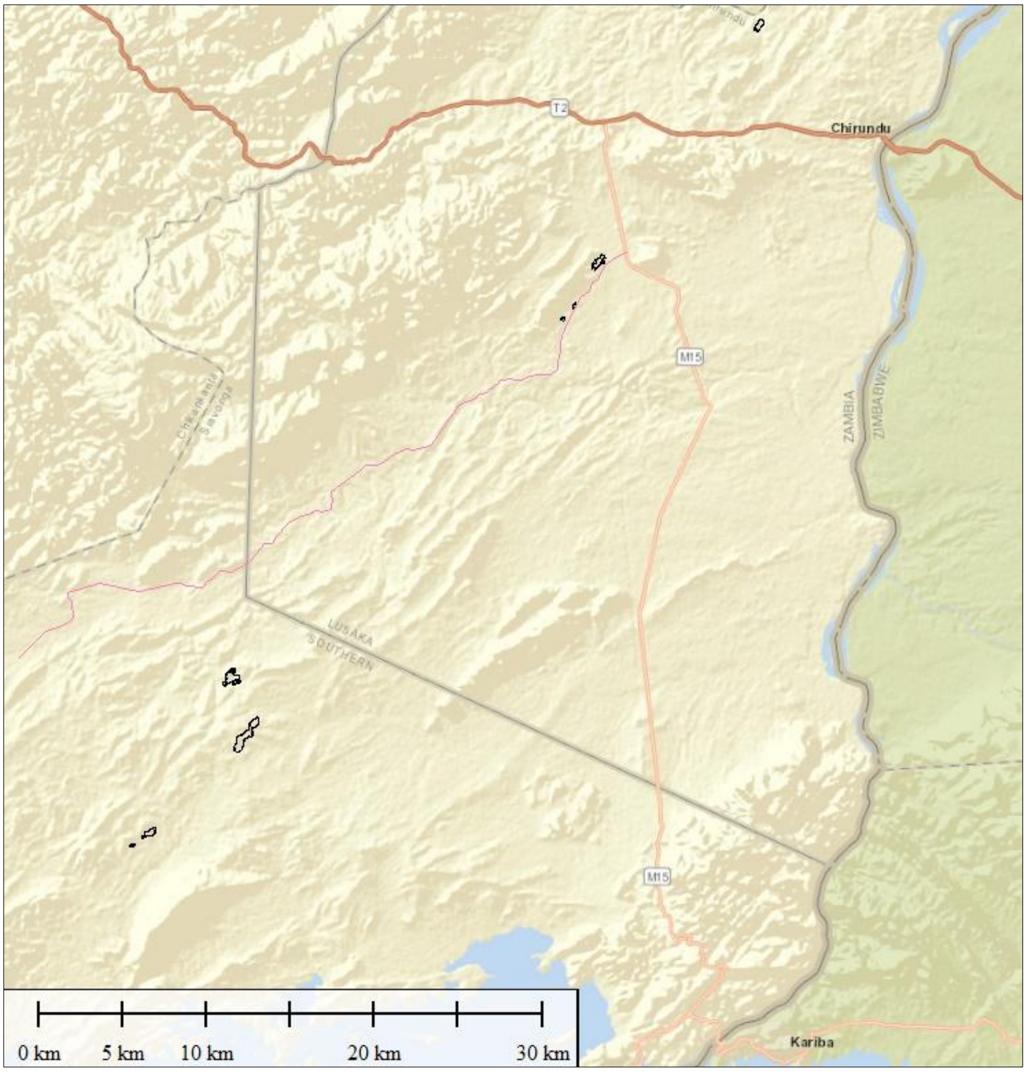

Figure 18-2: National roads in the Project area

Zambia has a total “gazetted” road network of around 67 671 km of which 40 454 km is the Core Road Network. In addition, there are an estimated 30 000 km of “non-gazetted” community roads comprising tracks, trails, and footpaths. The interested communities maintain non-gazetted roads.


Figure 18-3: T2 at the turning to the existing Gwabi access road (on the right) (July 2023).

Figure 18-4: D500 in the adjacent (south) of Njame (Njame are immediately to the right) (July 2023)

Figure 18-5: D500 directly north of Muntanga Area (Muntanga is to the left) (July 2023)

18.1.3.Existing access tracks

SRK conducted a site visit in July 2023 during which all the routes were driven, the Muntanga road alignment trade-off was started and a section of the Machinga River was walked.

18.1.3.1. Muntanga

From the D500, Muntanga is accessed via a dirt track (a reasonably representative view of the first 4km is shown in Figure 18-6) which bisects the village of Hachibozu, crosses the Machinga River (Figure 18-7) before rising to the Muntanga plateau area. The road is 2.5 m wide, unmade, although it has been sheeted in places leading up to the village. To the north of the village, the existing road climbs the escarpment with a number of sharp turns and steep gradients. The existing route would come very close to the edge of the Muntanga open pit.

It was concluded that the existing route to Muntanga could not be simply upgraded, and a route selection study would be required to determine the optimal route and mitigate impacts (see Section 18.1.4).

Figure 18-6: view of existing access track from D500 to Hachibozu village (showing recent repairs undertaken by GoviEx) looking south (July 2023)

Figure 18-7: view of existing access track (looking north) and Machinga river crossing located just to the south of the village of Hachibozu (July 2023)

18.1.3.2. Dibbwi

From Muntanga, Dibbwi is accessed via a dirt track (a representative view is shown in Figure 18-8). The road is 2.0 m to 2.5 m wide, unmade, has sharp changes in vertical and horizontal alignment and is only accessible using a four-wheel drive vehicle. There are four noticeable drainage crossings. The route is used by local traffic but has no noticeable constraints against being upgraded. Therefore, the route was proposed as the FS design base case.

Figure 18-8: View of the existing access track to Dibbwi (July 2023)

18.1.3.3. Gwabi

Gwabi is accessed via a dirt track (a representative view is shown in Figure 18-9). The road is 2.5m to 4.0m wide, is around 5.6 km to the proposed open pit area, has been sheeted to facilitate access to local businesses close to the Kafue River (e.g. a fish farm), and has some sharp changes in vertical and horizontal alignment. There are two noticeable drainage crossings at 2.1 km and around 5.3 km. There are two schools (at 0.4 km and 5.0 km) which need to be considered. Otherwise, the route was deemed adequate for upgrading for use as the Gwabi satellite pit access road.

Figure 18-9: view of the existing access track to Gwabi

18.1.4.Selection of Muntanga road alignment and Machinga River bridge studies

18.1.4.1. Overview

Potential access road route options for connecting the Muntanga site to the D500 were developed and assessed because the existing route to Muntanga could not be simply upgraded, and a route selection study is required to determine the optimal route and mitigate impacts. Significant factors were identified early on such as:

- How to avoid or minimise the impact on the village of Hachibozu through which the existing access track runs, noting the location of the school on the north side of the village
- The best option for crossing the Machinga River and how this might influence the road alignment and
- The most appropriate route for climbing up the escarpment to the plateau area where the mine and plant will be situated but cognisant of the open pit area.

To assess options, a multicriteria analysis ("MCA"), which incorporates a ranking of options based on a relative assessment of appropriate criteria, was used to assess the route and define a preferred route for FS design and costing.

Two iterations of the MCA were made Revision 1 ("MCA Rev1") following the site visit and MCA Revision 2 ("MCA Rev2") which ultimately recommended the access road route to be designed and costed at an FS level. MCA Rev 2 was updated once results of the PFS level comparative analysis for Machinga River Crossings were made. The finally selected route, which is "alignment option C6", was added to the MCA Rev 2 although very similar options were included in the MCA Rev1.

18.1.4.2. Sequence

The following sequence of study was undertaken:

- A site visit was undertaken in July 2023, with a qualitative appraisal of river crossing options
- Scoping level route development and MCA Rev 1 (Q4 2023) and conceptual costing
- Collection of detailed topography for selected river crossing locations and comparative analysis of the river crossing structure options for preferred river crossing locations
- MCA Analysis Revision 2 ("MCA Rev 2") and concept costing as reported in January 2024 with recommendation for FS access road route.

18.1.4.3. Segments and route options

In determining the route options a series of road segments were developed as follows:

- Following existing access tracks and local roads where possible
- Selecting preferred Machinga River crossing points
- Determining the D500 connection point and minimising distance and
- Minimising distance and avoiding obvious areas of disadvantage or impact (e.g. Hachibozu village, steep gradients etc).

The segments and routes assessed in MCA Rev 1 are presented in Figure 18-11 and Figure 18-12.

18.1.4.4. Machinga River crossing locations

The selection of an optimal location to cross the Machinga River was seen as pivotal to the delineation of a suitable and cost-effective access road route.

The Machinga River is located immediately south of the Hachibozu Village. The existing crossing point has no permanent structure, it is directly south of the village, has several steep gradients and alignment changes and is not considered suitable for the development of a bridge.

Figure 18-10: Existing crossing point on the Machinga River, south of the Hachibozu River

During the site visit, approximately 5 km of the drainage alignment was walked to locate potentially suitable crossing points. Based on a qualitative assessment of key characteristics, eleven potential options were proposed and compared - these are shown in Figure 18-13 and final comments are presented in Table 18-1. From these options, six crossing locations were considered plausible (although MRVE-A / B are adjacent) and informed the route segments in the road options assessment were linked to these crossing options (see Table 18-2, Figure 18-11, and Figure 18-12).

The crossing locations are:

- MRVE-A/B, two adjacent options from east of the village
- MRVW-A, an option close to the village on the west
- MRVW-D, E, and F, which are further along the river west of the village.

The other options were discounted. SRK then ranked the five preferred crossing point options relative (Figure 18-13) and the results of this analysis were used in the MCA revision 1.

Figure 18-11: Muntanga Access Road - Segments

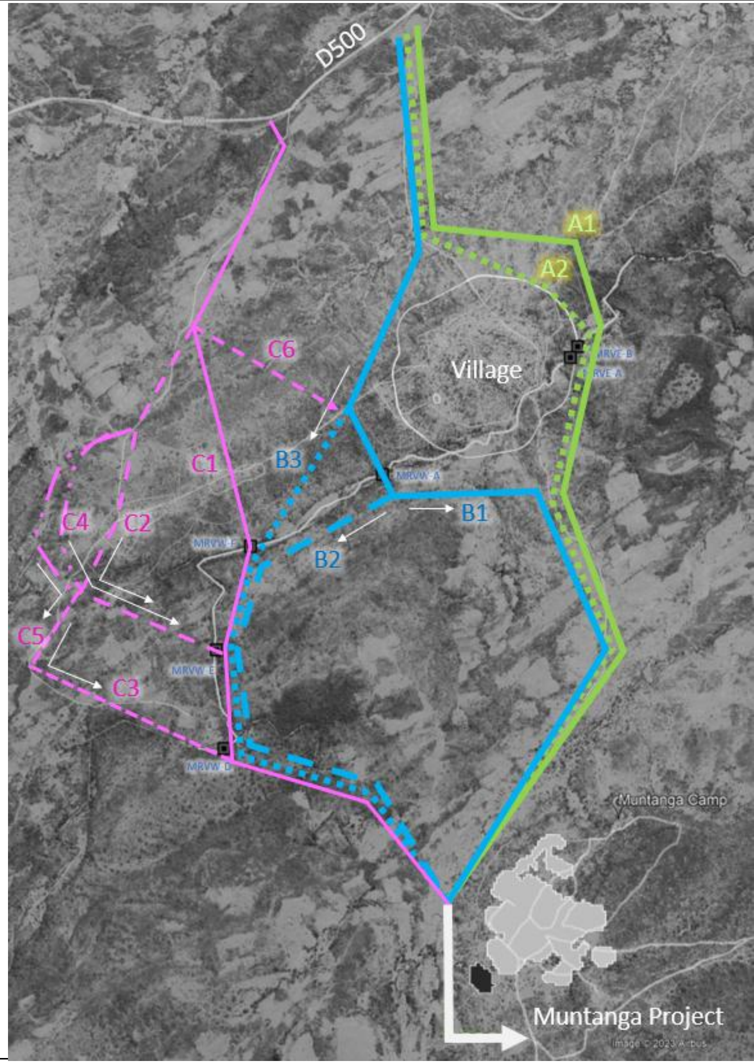

Figure 18-12: Muntanga Access Road – Routes (MCA Rev 2)

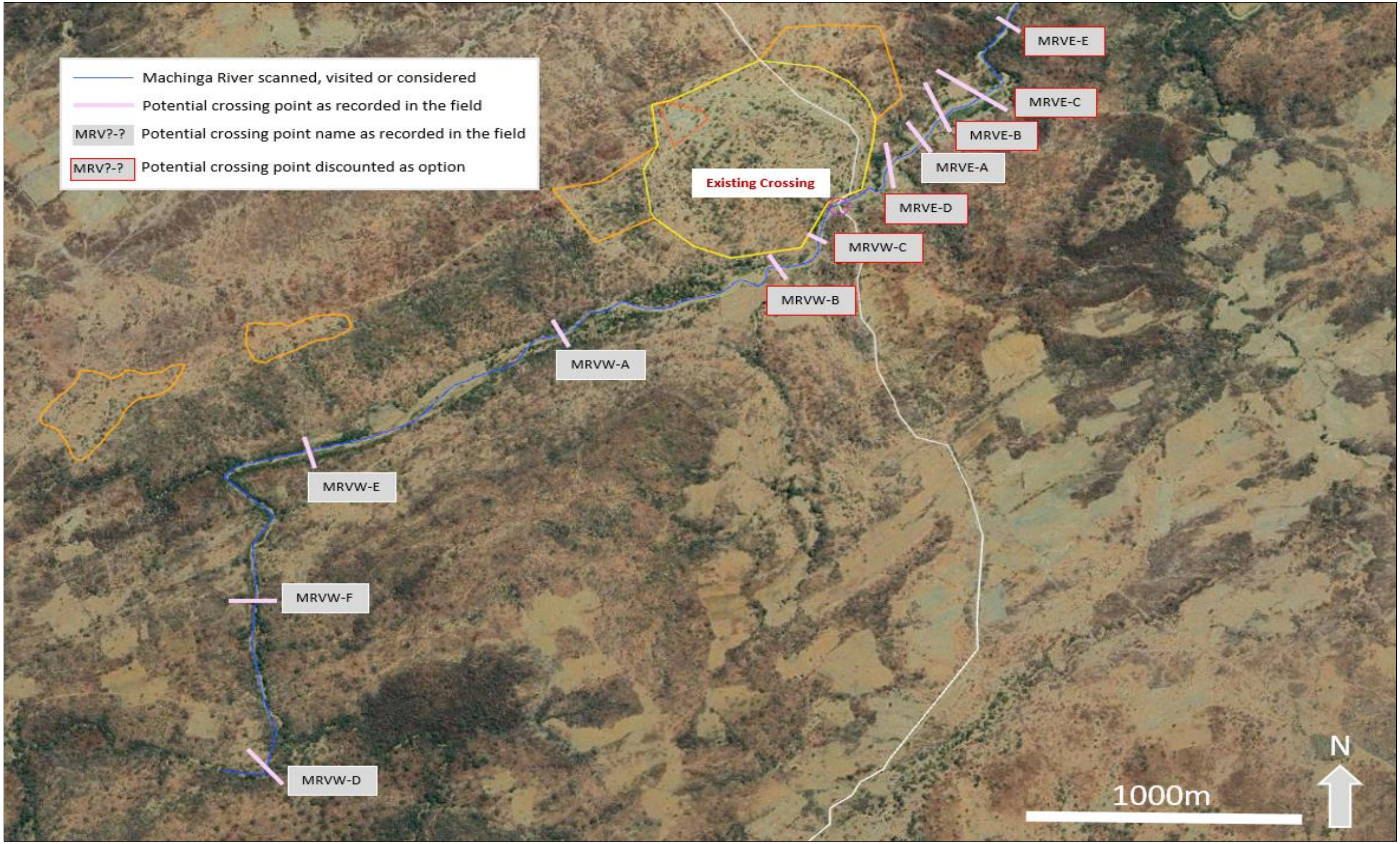

Figure 18-13: Location of Machinga River crossing options delineated on the site visit for assessment.

Table 18-1: River crossings assessment (see Figure 18-13)

Crossings	Pros	Cons
East of Village		
MRVE-A	Narrower channel No agriculture at crossing point Rock outcrop Reasonable approaches	Upwind of the village Proximity (<200 m) to the village Disrupt agriculture area and access to the area Likely to result in some relocation activity
MRVE-B	Reasonable approaches Narrower channel Beyond 200 m buffer	Proximity @ 200 m upwind from the village Agriculture (20 m) Wider flood plain (80 m) Soils evident
MRVE-C	Reasonable approaches Narrower channel Beyond 200 m buffer	*Extensive* agriculture (50 m) *Widest* flood plain (190 m) Soft soils evident
MRVE-D	Upwind but further from village Narrow channel Rock outcrop Straight channel alignment	Deep and steeply incised river channel Adverse terrain on approaches Result in increased road distance
MRVE-E	Narrower channel Rock outcrop Reasonable approach Should reduce overall road length (allows for a more direct crossing point for a specific road option)	Upwind of the village Very proximal (<100 m) to the village footprint Disrupt agriculture area and access to the area Likely to result in some relocation activity Likely to be very high-energy river dynamics
West of Village		
MRVW-B	Narrower channel than nearby sections No agriculture at crossing point Rock outcrop Reasonable approaches Should reduce overall road length (allows for a more direct crossing point for a specific road option)	Very proximal (<100 m) to the village footprint Disrupt agriculture area and access to the area May result in sharp alignment changes Likely to result in some relocation activity
MRVW-C	Narrower channel than nearby sections Rock outcrop Reasonable approaches	Very proximal (<100 m) to the village footprint This will result in sharp alignment changes Very rock difficult ground Likely to be very high-energy river dynamics Likely to result in some relocation activity
MRVW-A, MRVW-E, MRVW-F	Narrower defined and consistent Nearby rock outcrop Reasonable approaches Far from village No horticulture in the river	Likely to result in some relocation activity
MRVW-D	Narrower defined and consistent Nearby rock outcrop Reasonable approaches Far from village No horticulture in the river	Likely to result in some relocation activity More dynamic environment with tributaries and intersections.

Based on the visits made to the sites, SRK ranked the crossing points relative to one another as per Table 18-2.

Table 18-2: Relative ranking of selected river crossings assessment

"Position"	Crossings	
1	MRVW-E	These crossing points are on straight sections of river, relatively narrow and consistent, symmetrical form. Bedrock is noted near to surface. The approaches are reasonable and consistent.
2	MRVW-F	As above. The northern approach is slightly steeper.
3	MRVW-A	As above. The northern approach is steeper. Some rocky terrain at surface. On the south bank, there may be a need for a short causeway.
4	MRVW-D	Reasonable approaches. Upstream of two tributaries but on a section with some directional changes.
5	MRVE-A/B	MRVE-A/B is upwind of the village and very close to population and agriculture. MRVE-A is within the 200m buffer but has a narrower stream section and MRVE-B has a wider floodplain.

18.1.4.5. Bridge option study

ASD Engineering (Lusaka/ Johannesburg) was commissioned to review crossings MRVW-A, MRVW-D, and MRVW-E, and subsequently, to analyse the potential for a bridge structure and low water crossing structure at MRVW-A and MRVW-E (see Figure 18-13) to determine a preferred option from A or E, which would then be incorporated into a Revision 2 of the Muntanga Access Road MCA.

The assessment of the locations is summarised as:

- MRVW-A: provides both a wide low-lying area towards the east and a portion with a narrower channel created by the river. There is evidence of water ingress into one of the embankments. This should be considered while designing the crossing options, as signs of strong current in peak flow and/or poor topsoil can easily be eroded
- MRVW-D: is characterised by exceptionally low elevation, with flood lines encroaching upon the entire area. Additionally, the presence of a steep incline poses considerable challenges for grading efforts related to crossings. Another notable feature is the convergence of river channels at this site, resulting in complex hydraulic dynamics and structural obstacles. Considering the dynamic nature of the area, including the potential for bank erosion and meandering water courses, it is evident that this site is less suitable compared to others
- MRVW-E: is another site that was considered for both the low-water crossing as well as a potential elevated crossing, due to the topography not having an extremely wide river channel, in addition to an existing path that is in usage, at least occasionally
- Hydraulic/hydrology analysis findings for each site have confirmed all locations to be low-lying, while some sites present advantages over others. Furthermore, based on guidelines for hydraulic design of bridges the flood line mark has been estimated at 536 m above sea level (“ASL”) on the cross section. This proposed height has been given to be suitable for a 20-year flood
- While further and more detailed survey is required, it is assumed that at this stage, designs should be made to allow for overtopping of the structure that will be selected.

The low water crossing concept has a concrete pavement supported on dump rocks that have been deposited onto a prepared base. The dump rocks are covered in a geotextile membrane to allow some infiltration without occasioning erosion of the base. The pavement is provided with down-stand beams to anchor the structure and further prevent undercutting. See Figure 18-14 for an example.

A low water crossing for MRVW-A is estimated to be 60m long although during detailed design consideration may need to be given to the northern approach as approach gradients need to be considered and minimised (in the order of 6 % to 8 %). MRVW-A has quite a steep approach from the north which would require a significant earthwork “cutting” to accommodate the low water crossing. The detailed topography collected for this assessment allowed for modification of the alignment to take advantage of a topographic low to minimise cut and fill. A low water crossing for MRVW-E is estimated to be 80 m long and would equally be at risk of resulting in a large cutting to achieve the approach gradient.

The bridge structure option consists of cast-in-place reinforced concrete abutments, piers, and a bridge deck – see Figure 18-15. The bridge at MRVW-A is estimated at 60m long but would be longer for MRVW-E. The detailed topography collected for this assessment allowed for modification of the alignment to take advantage of a topographic high to minimise cut and fill at the north end. Some earthworks are required on the south approach, which are not considered in this assessment.

The following general observations and conclusions were made:

- It was concluded that MRVW-D is located in a more complicated hydraulic/ hydrological setting, which is sub-optimal compared to MRVW-A or E
- The comparative analysis indicates a crossing structure (bridge or low water) at MRVW-A is likely to be more cost-effective than MRVW-E, targeting MRVW-A should result in the lowest-cost crossing. The northern approach road to MRVW-E would also require the relocation of houses adding more expense
- Although the low water crossing is likely to be less costly than a bridge crossing, given the requirement to achieve the design gradient on the approach, the potential for disruption to traffic in the wet season and the risks this introduces, a bridge structure is recommended
- Although not rigorously monitored, various observations were made by the GoviEx team of the Machinga River in flood during precipitation events in the rainy season (January time)
- MCA input is to be weighted according to each location.

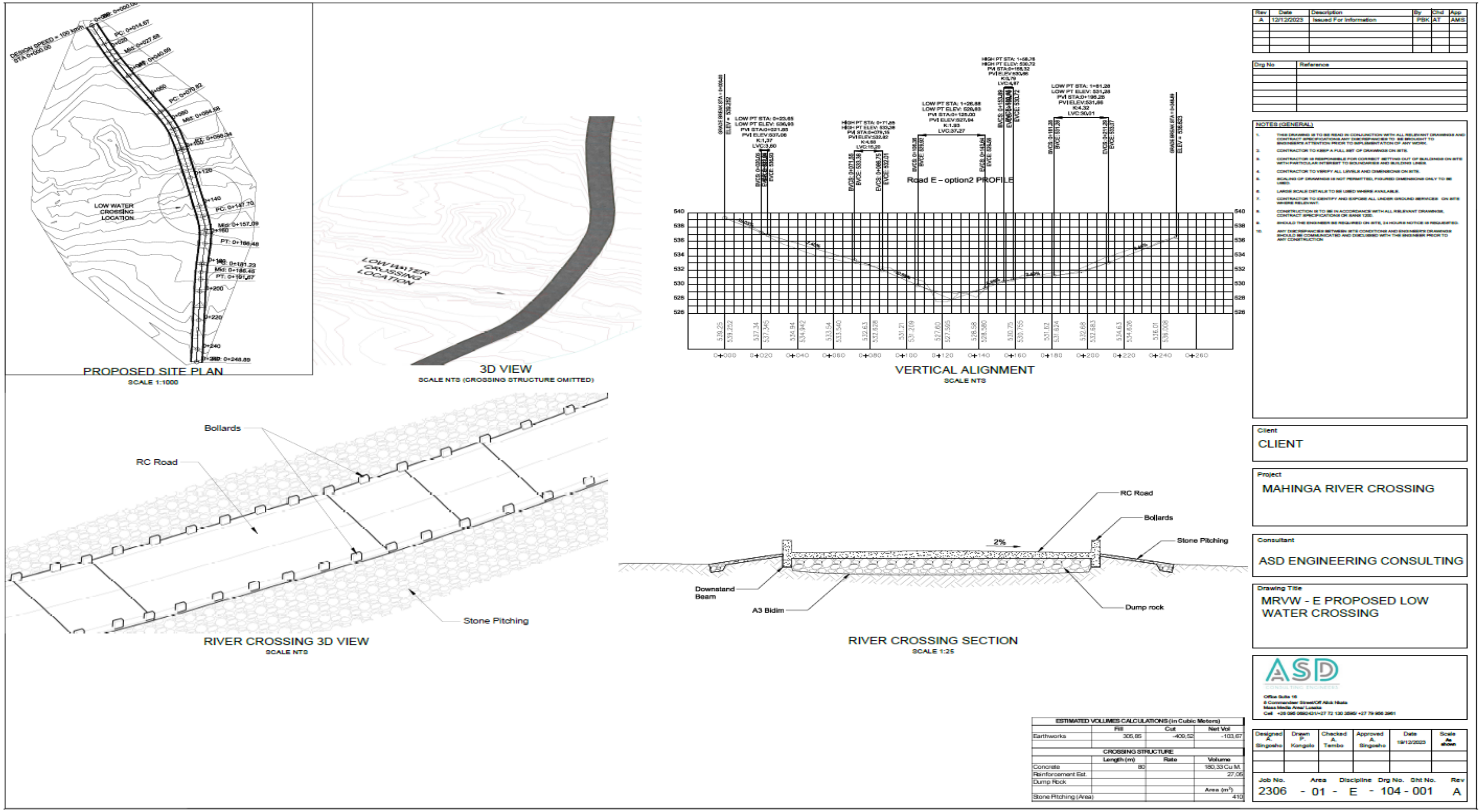

Figure 18-14: ASD drawing example for a low water crossing structure (please refer to Appendix A for all drawings)

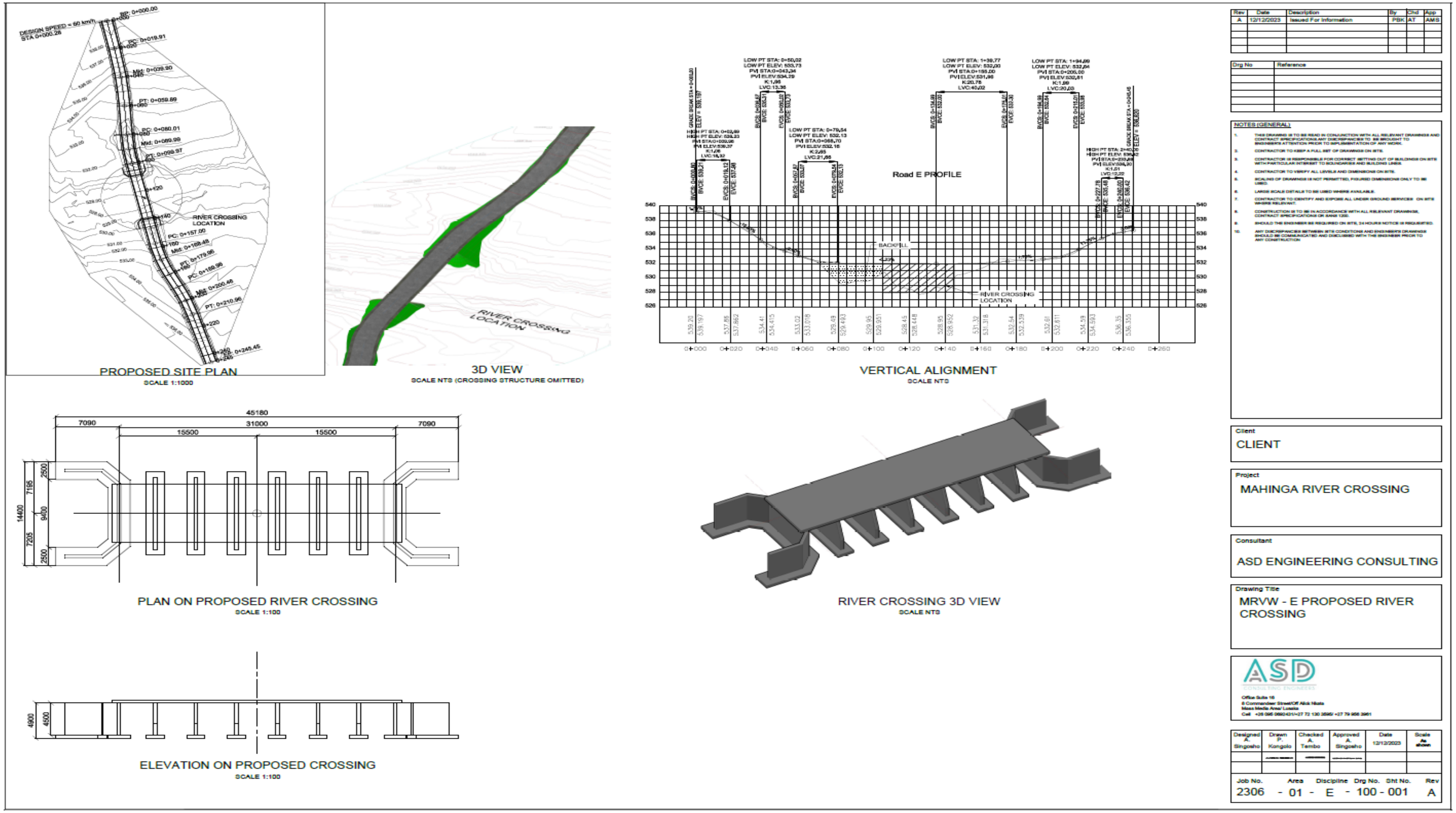

Figure 18-15: ASD drawing example for a bridge structure (please refer to Appendix A for all drawings)

A bridge in this location would be 80 m long. As with MRVW-A, the detailed topography collected for this assessment allowed for modification of the alignment to take advantage of a topographic high to minimise cut and fill at the north end. Similar to MRVW-A, earthworks are required on the south approach, which are not considered in this assessment.

Preliminary engineer's costs excluding the following indicated a crossing at MRVWA to be more cost-effective than MRVWE.

- Bulk earthworks
- Slope protection
- River channel works
- Signage
- Engineering, procurement, and construction management ("EPCM")
- Environmental management and control.

18.1.4.6. Comments and recommendations

SRK makes the following general observations:

- The engineer concluded that MRVW-D is a more complicated hydraulic/hydrological setting, which is sub-optimal compared to MRVW-A or E
- Given that the comparative analysis indicates a crossing structure (bridge or low water) at MRVW-A is likely to cost less than MRVW-E, targeting MRVW-A should result in the lowest-cost crossing. The northern approach road to MRVW-E would require the relocation of houses adding more expense
- The low water crossing is likely to be less costly than a bridge crossing; however, the volume and cost of bulk earthworks and the potential for interaction and safety issues in the rainy season must be accounted for.

SRK recommends the following:

- Modify the MCA weightings on the bridge crossings and rerun the MCA (as Rev 2).

18.1.4.7. Rainy season observations

Although not rigorously monitored, various observations were made by the GoviEx team of the Machinga River in flood during precipitation events in the rainy season (January time).

18.1.4.8. Final comments

Based on the visits made to the sites, the following recommendations were made to be incorporated into a second revision of the MCA:

- Modify the MCA weightings on the bridge crossings
- Given that the comparative analysis indicates that on -balance, a crossing structure at MRVW-A is likely to cost less than MRVW-E, targeting MRVW-A should result in the lowest-cost crossing
- Although the low water crossing is less costly than a bridge crossing, given all the uncertainties.

18.1.4.9. Multicriteria analysis – Methodology

The methodology selected for ranking the alternative site locations is a structured procedure. The various sites can be compared and evaluated to select the preferred option for the access road.

Determine positive and negative indicators:

1. Civil construction factors – perceived to affect cost – e.g. length, ground conditions, culverts, gradients, upgrade or new
2. ESG – Construction/ Capex: bisection of agriculture, buffer zones around the village
3. ESG – Operational impacts; wind direction relative to population and requirement for local populations to cross the road to reach schools and areas of horticulture
4. River Crossing - Drainage bed width. Outcrop at or nearby location. Floodplain. Flow characteristic. Overall assessment factor (relative).

Weighting factors were then applied to each element. Assignment of the weighting factors is an opportunity to apply values to the evaluation process so that the final scores for each site reflect the perceived issues involved at this stage. There is a higher weighting towards:

- Distance (infrastructure capital cost and LOM operating costs)
- Requirement for relocation and
- Proximity to population.

Table 18-3: Weightings

Item	Name	Overall weighting
1	Civil construction	30
2	ESG – Capital items	30
3	ESG – Operational impacts	30
4	River crossing	10

Following the assignment of the weighting factors, each selection element is evaluated according to established criteria and ranked on the simple five-point scale from 1 (excellent: “most desirable” or “cost-effective”) to 5 (unacceptable or “least desirable/ least cost-effective”). The detailed ranking criteria are to be used, which differentiate between optimal (lowest score) and sub-optimal conditions (highest score).

The criteria ranking scores are multiplied by the criteria weightings to produce a comparative ranking of sites.

The access road will carry construction and operational traffic The road needs to connect the site with the D500. The existing road passes through Hachibozu Village and through a section which will be affected by the mining footprint. Both will need to be realigned. The constraints discussed with GoviEx’s environmental consultant are listed in Table 18-4.

Table 18-4: Constraints

Constraint	Value
Hachibozu Village	Optimally, should be bypassed
Other receptors	Village School
Advisable distance from road to village or house	Minimum 200 m
Cost for relocation of family	USD15-20 k
River crossing points	As determined during the site visit.

18.1.4.10. Multicriteria analysis – Results

Various road segment options were defined, measured, and numbered (see Table 18-5) and routes were defined comprising of selected segments. Initially, the analysis was run for 11 routes (“MCA Rev 1”) but following receipt of the Machinga Bridge study results and the addition of Route C6 a second analysis was undertaken (“MCA for Rev 2”).

Route C6 is an amalgamation of C1 and B1, achieves the optimal crossing point as determined from the Machinga Bridge Study (MRVW-A) but requires a segment of new road and removes the following impacts:

- Brings the access road route away from the existing Hachibozu village road and the western side of the village
- Removes the requirement for relocation associated with Option C6.

The routes are summarised below in Table 18-5 and in Figure 18-12.

Table 18-5: Routes summary (MCA Rev 2)

Item	Name
A1	East of Hachibozu Village (MRVE-A crossing) following the existing access road where possible. Some relocation of families will be required.
A2	As per A1 but at MRVE-B crossing. No relocation is deemed to be required.
B1	West of Hachiro Village (MRVW-A crossing) following the existing access road where possible.
B2	West of Hachibozu Village (MRVW-A crossing) with routing west around hill
B3	West of Hachibozu Village (MRVW-F crossing) with routing west around hill
C1	West D500 route through the outer village via MRVW-F. Relocation required
C2	West D500 route through the outer village via MRVW-E. Relocation required.
C3	West D500 route through the outer village via MRVW-D. Relocation required.
C4	As per C2 but less relocation
C5	As per C3 but less relocation.
C6	D500 access point as per C-routes, new road segment, to meet B routes thereby no relocation and MRVWA

The indicators, scores, weightings and selection criteria were analysed. The results were presented and showed the following overall conclusions:

- The MCA indicated the best options to be C1, B1, B2 and C5, with the exact ranking depending on the weighting of the main indicators
- In general, these are routes that pass west (downwind) of the village and school and utilise MRVW-A crossing or MRVW-F crossing
- Option C5 passes west of the village and an outer part of the village but utilises MRVW-D / E crossing and results in a longer road distance and longer sections of new road
- Option C1 would require some additional relocation of a small outer settlement associated with the village
- Options B1 and B2 avoid the village and pass to the west, although would still have an impact on the western edge of the main village (noise, dust, lights, traffic) despite being beyond the 200 m buffer as advised by the environmental and social specialists
- In summary, in Rev 1, B1, B2 and C1 were deemed to be most desirable but the results were seen to be very similar and all had disadvantages that needed to be considered
- Once C6 was added, this option consistently showed the best scores regardless of the weighing of the indicators and thus was selected as the base-case FS road route.

18.1.5.Surveys

18.1.5.1. Topographic

LiDAR surveys to cover the mine sites were initially available and further LiDAR surveys were commissioned to cover the road alignments to inform FS design. These surveys should also be adequate for detailed design. Freely available SRTM data was used for the RAP access roads where LiDAR data was not available.

18.1.5.2. Geotechnical

The selected ground investigation was undertaken along the route of the Muntanga access road along with a desk-based assessment of bedrock types and likely impact on subgrade conditions. The first 5km are underlain by Madumabisa Mudstone, corresponding to weathered, cohesive soils at the surface, and Braided/ Meandering Facies resulting in weathered rock and granular soils at the surface.

18.1.5.3. Hydrological

Hydrological analysis was used in the assessment of the Machinga River bridge and is described in Section 18.3.

18.1.6.Design criteria

18.1.6.1. Road types

There are two principal types of access roads:

1. Primary (project) access routes; and
2. RAP access / local roads.

18.1.6.2. Traffic Analysis

At project inception, the preliminary assessment of traffic for operations indicated traffic to be equivalent to 0.6 million equivalent standard axles. On this basis, the principle of “Low Volume Roads” design was adopted. A number of “Design Manuals for Low Volume Roads” exist and the Zambian Roads Authority manual (2019) and Ethiopian Roads Authority (ERB, 2011) were referred to.

18.1.6.3. Basis of design – Primary access roads

Design criteria for the PARs are presented in Table 18-6 and correspond to Recommended Parameters General for Unpaved Roads (e.g. DC3 or DC4 Class LVR) Flat / Rolling (as per the LVR Manual, ERB, 2011).

Table 18-6: Design criteria for PARs

Item	Units	Value
Road pavement	#	Gravel / Unbound
Design speed	km/h	60 (30, where speed restriction shall apply)
Road width	m	7.0
Min-stopping sight distance	m	105.0
Min horizontal radius	m	175 (locally 100)
Max desirable gradient	%	5
Max gradient	%	6
Max. super-elevation	%	6
Min crest vertical curve	K	19
Min sag vertical curve	m	3.5
Normal cross-fall	%	6.0
Minimum cross-fall	%	4.0
Shoulder width (each side)	m	0.75

Based on the desktop review of the underlying Subgrade California Bearing Ratio (“CBR”) value to be used in pavement design, the subgrade strength class was established as a minimum of “S3” associated with the section of road developed on low-lying ground underlain by the Madumabisa Mudstone.

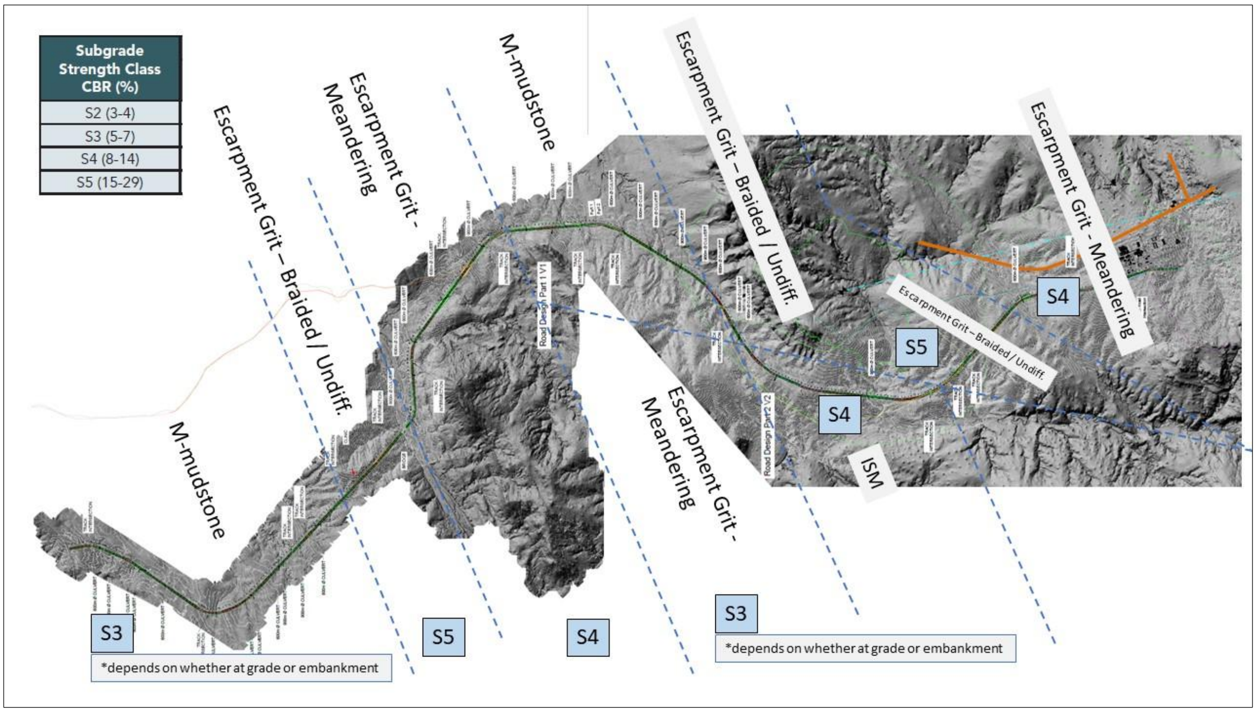

Figure 18-16: Predicted subgrade conditions

The width of 7 m and selected crest/ sag curves are considered sufficient for the import of mining mobile equipment on low-bed transporters.

18.1.6.4. Basis of design–Reconciliation action plan access roads/ Local road detours

Design criteria for the RAP access and local roads are presented in Table 18-7 and correspond to Recommended Parameters General for Unpaved Roads (e.g. DC1 Class) Flat/ rolling (as per the LVR Manual, ERB, 2011). Cutting slopes were assumed to be 1-in-1 and embankment slopes 1-in-2.

Table 18-7: DC1 lower volume roads: Recommended parameters general for unpaved roads flat/ rolling

Unpaved gravel roads, DC3 / DC4 for flat terrain		
Road pavement	#	200 mm base course.
Design speed	km/h	30
Road width	m	3.3
Max gradient	%	7 (max for gravel)
Max. super-elevation	%	7 (max for gravel)
Min crest vertical curve	K	6 (construction equipment ok as can drive it)
Min sag vertical curve	m	2
Normal cross-fall	%	4.0
Minimum cross-fall	%	4.0
Shoulder width	m	0.5

18.1.7.Muntanga primary access road

18.1.7.1. Design development

Road alignment option “C6” was taken forward into FS design. The road was designed in AutoCAD Civil 3D software in two sections; “P1” and “P2”, which were then merged for analysis of cut and fill. Multiple iterations were undertaken to broadly balance cut and fill with sufficient “net site won material” for crushing for capping and/or the subbase course. Section P1 covers chainage 0 + 000 (D500) to 5+200. Section P2 extends from chainage 5 + 200 to 10 + 150 (access point to mine site area for haul trucks from Dibbwi). The Machinga River bridge is located in P1 at chainage 2 + 930 to 2 + 990.

18.1.7.2. Alignment

The main access road has a total length of 10.1 km long and links the site to the D500 national. The route alignment is shown in Figure 18-17.

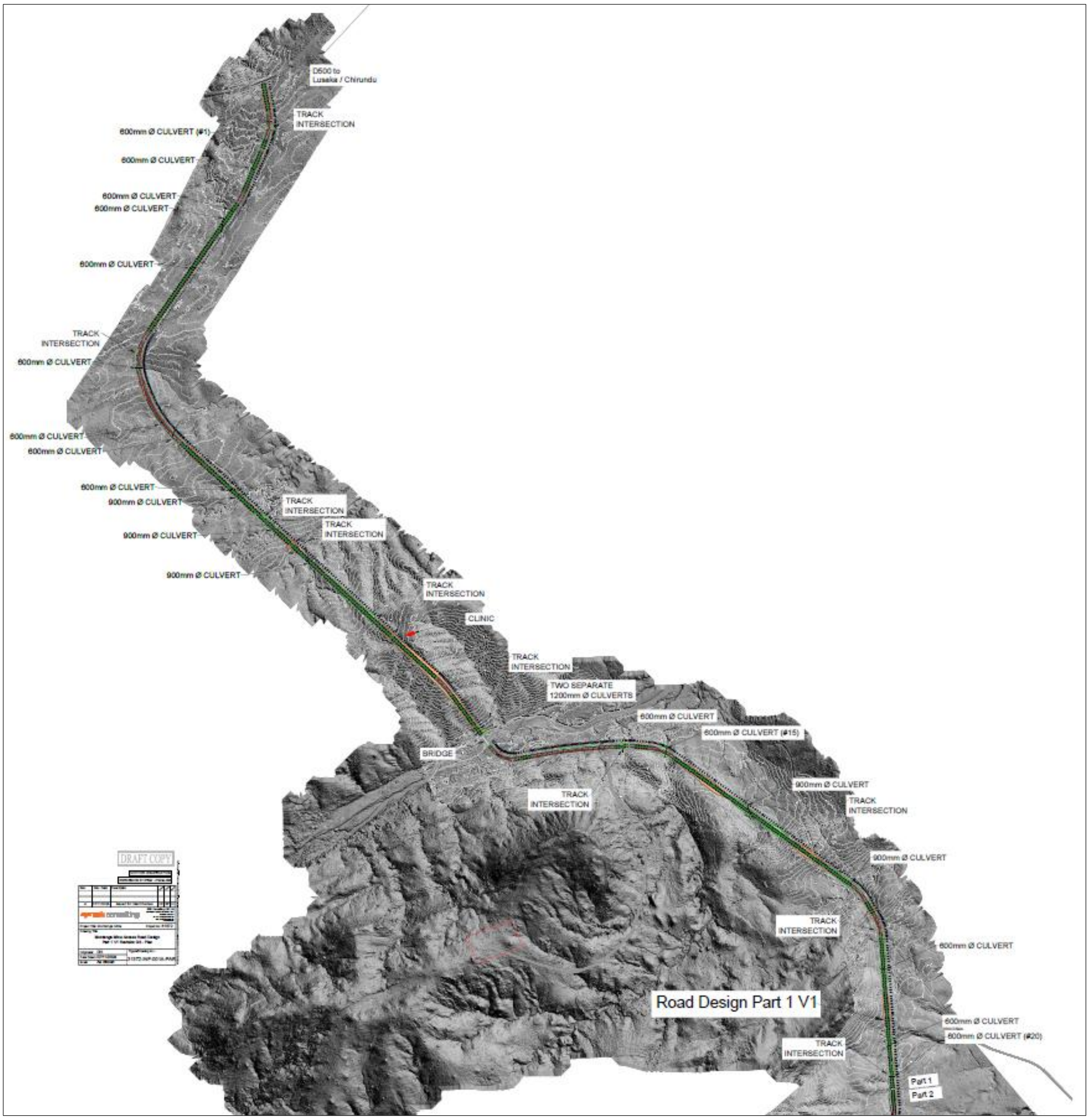

Figure 18-17: An image of the Muntanga Road P1 Chainage 0+000 (D500 intersection) to 5+200 (31372-INF-001A_PAR_MuntangaPlan _P1_V1_RevG3_20241107.pdf)

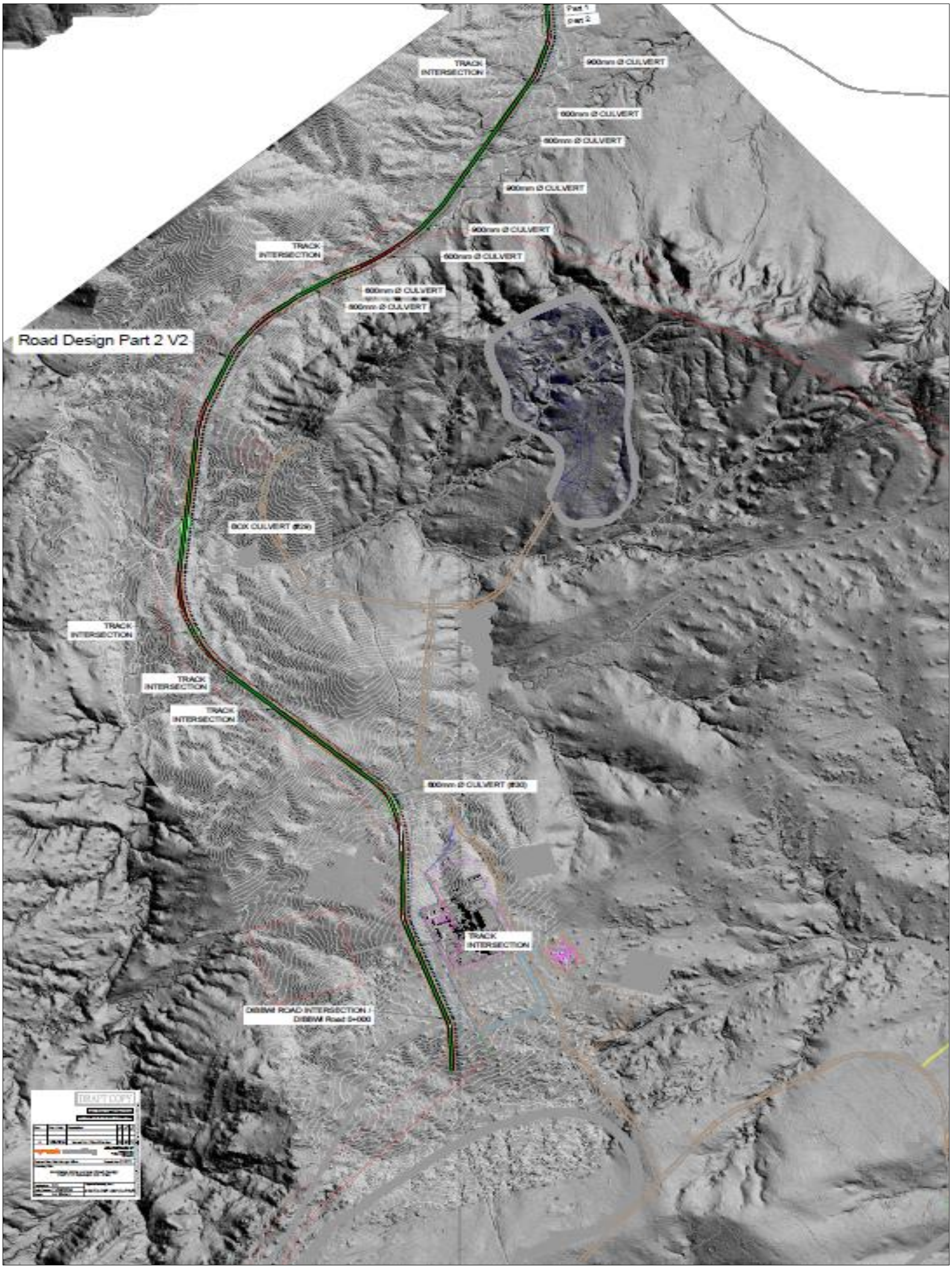

Figure 18-18: An image of the Muntanga Road P1 Chainage 5+200 to 10+150 (31372-INF-001C_PAR_MuntangaPlan_P2_V2_RevG3_20240911.pdf).

Road design parameters are presented in Table 18-6. The total change in natural ground level over the length of the road is approximately 85 m with a maximum gradient of 7 %. The alignment can be split into segments, which are detailed in Table 18-8. The overall approach has been to minimise earthworks within the constraints of the design criteria but also to broadly balance cut and fill for bulk earthworks and initial layer works.

Table 18-8: Muntanga access road segments

#	Description	Chainage start	Chainage finish	Min elevation	Max elevation	Length [m]	Minimum subgrade conditions
1	D500 to Hachibozu Hill	0+000	1+900	530	535	2 000	S2 / S3
2	Hachibozu Hill (New Clinic at 2+500)	1+900	2+900	530	550	1 000	S5
3	Machinga River Bridge	2+900	3+100	530	526	200 (structure is 60 m)	S4
4	Flat, low-lying ground	3+100	5+200	526	550	2 100	S2 / S3
5	Flat, low-lying ground	5+200	6+300	526	550	1 100	S3
6	“Muntanga Rise”	6+300	9+100	550	610	2 800	S4
7	Muntanga Mine (Dibbwi Road intersection at 9+950)	9+100	10+150	608	614	1 050	S5
	Total		**10+150**			**10 150**	

18.1.7.3. Geotechnical

The available topographic and geological survey data were used in the determination of the route alignment. Geotechnical investigations were carried out at selected sites and indicated CBR of 5 % to 30 % depending on location (see Table 18-8).

18.1.7.3.1. Route constraints

The main route constraints were considered in the delineation of the route in Phase 1. An additional constraint was discovered during the LiDAR survey where construction of a local clinic had commenced on the access road alignment; the alignment and bridge location were subsequently modified and the road width widened between D500 connection point and the clinic to provide additional width of the local traffic (chainage 0+000 to 2+900).

18.1.7.3.2. Design vehicles and traffic

Primary usage (>90 %) will be for 4-axle rigid highway construction trucks with a 20 t to 30 t payload or five/six-axle articulated delivery lorries. Other vehicles will be delivery trucks (similar loading) and light vehicles. Maximum truck width will be around 2.6 m.

18.1.7.3.3. Road geometry

See Figure 18-19 below for the road geometry as developed in the Design Criteria for 0+000 to 2+500 (Clinic) and 2+500 to 10+150m chainages.

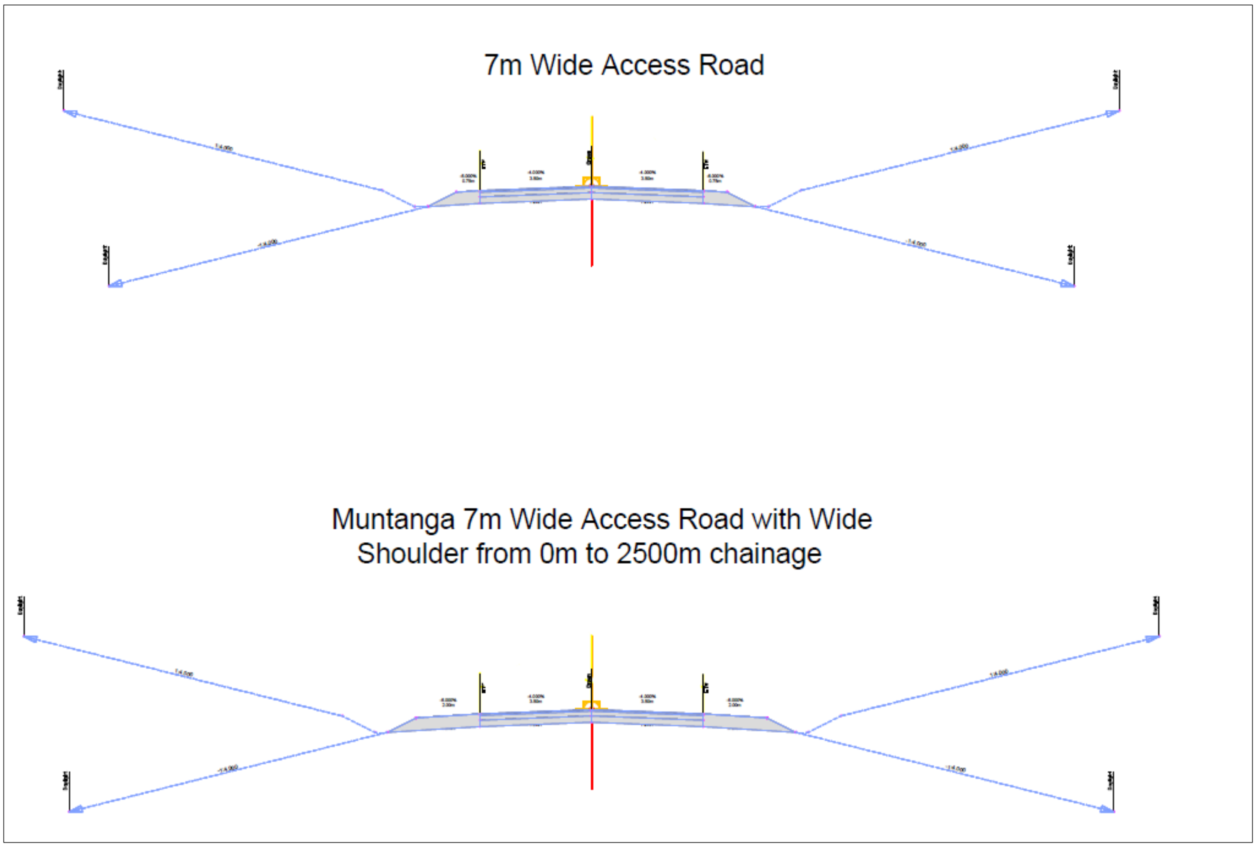

Figure 18-19: Road geometry – Muntanga access road

18.1.7.4. Pavement design

Pavement design parameters are presented in Table 18-9. These are subject to field and laboratory trials of waste rock for use as a pavement aggregate.

Table 18-9: Pavement design

Layer	Details
Wearing course of compacted crushed and graded waste rock to 98 % MOD.AASHTO equivalent to a G2	150mm thick
Base course of compacted crushed and graded waste rock to 95 % MOD.AASHTO equivalent to a G5	150mm thick
Subbase course of compacted crushed and graded waste rock to 93 % MOD.AASHTO equivalent to a G7	150mm thick
In-situ subgrade ripped and compacted 90 % MOD.AASHTO	150mm thick

If ROM ore from Gwabi is to be trucked to the site, additional base course thickness must be added.

18.1.7.5. Earthworks

Vegetation will be grubbed out and topsoil across the footprint of the earthworks will be removed and stockpiled for future closure works. Topsoil is assumed as 150 mm thick.

Bulk earthworks will be conducted to achieve subgrade level. At the subgrade level, the material will be ripped, watered and compacted to form the sub-grade vertical profile to receive the engineered layers of the pavement. Materials deemed “unsuitable” for construction will be stockpiled (currently foreseen to be around 10 % of site-won material).

The FS design earthworks balance results in a surplus of reusable material of 32 000 m^3, which is sufficient for subbase and base course construction to facilitate construction activities. The majority (circa 23 000m^3) came from excavations between chainage 1+900 to 3+100, to achieve the northern approach to the Machinga River Bridge.

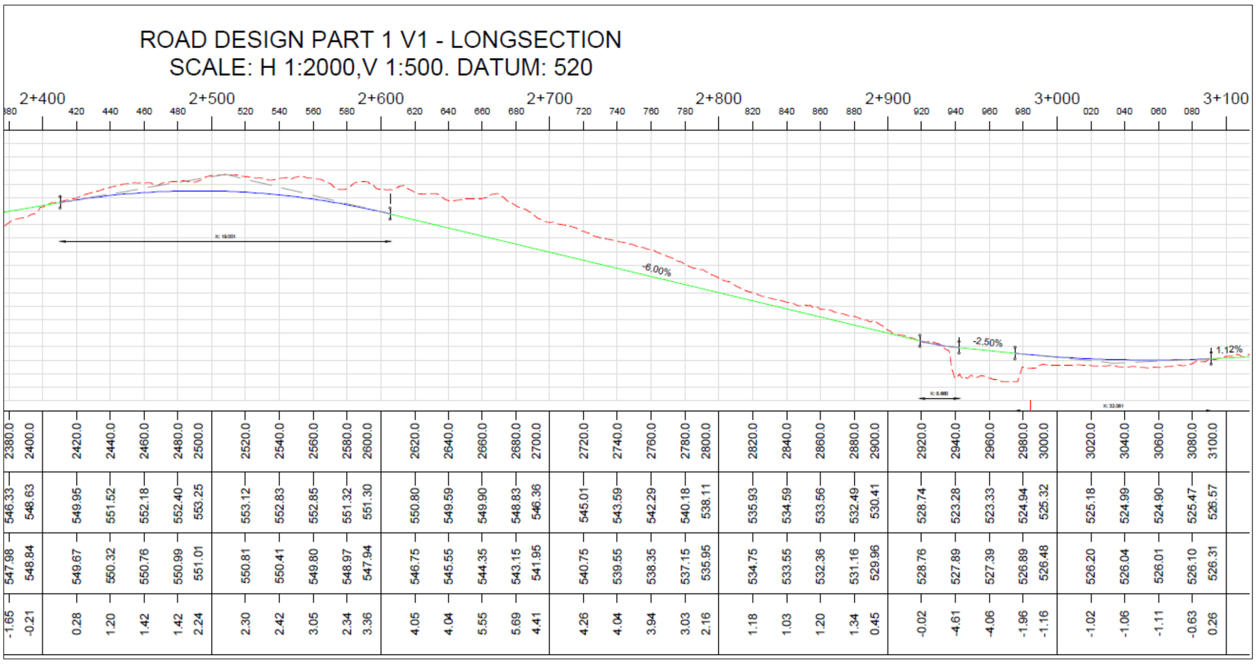

Figure 18-20: Extract from the long section through the Muntanga access road

Final base course and wearing course layer materials are to be placed during ramp-up phases and will be drawn from stockpiles of waste rock from pre-stripping activities or commercial sources.

It is intended that inert waste rock stockpiles are used for material for re-sheeting and replacement of gravel wearing course over the LOM. Samples of these stockpiles will be tested to determine the quality of material that can be produced once crushed, screened and blended.

18.1.7.6. Culverts/ Drainage

A grader-formed V drain will be constructed on the upslope side of the road to direct overland stormwater flows to pipe culverts that are positioned along the route. The series of culverts are installed to manage surface water flows and the detailed topography has been used to locate the culverts. Three principal pipe culvert sizes have been adopted with box culverts ("major culverts) used at specific locations:

- Type 1 Culverts are single, double or triple 600 mm ID concrete pipe culverts with concrete head and wing walls and inlet and outlets
- Type 2 Culverts are single, double or triple 900 mm ID concrete pipe culverts with concrete head and wing walls and inlet and outlets
- Type 2 Culverts are single, double or triple 1 200 mm ID concrete pipe culverts with concrete head and wing walls and inlet and outlets
- Major culverts have been considered in more detail and incorporate one or more 1 200 m width box culverts (pre-cast or cast in situ) with a reinforced concrete deck, with concrete head and wing walls and inlet and outlets.

18.1.7.7. Machinga River bridge

The Machinga River Bridge is located on the Muntanga PAR at approximate chainage 2+900 and chainage 3+100. Following the bridge location trade-off study, which fed into a wider access road trade-off study, the location MRVW-A was selected as the optimal crossing point of the Machinga River.

An assessment of bridge structure versus low water crossing established a bridge as the preferred option largely due to the constraint from the topography immediately north of the bridge. Subsequent to these works the approach alignment was revisited due to the newly established local clinic located to the north, which concluded the location now presented.

The overall layout and alignment and typical details are presented in Figure 18-21 and Figure 18-22. The design of the bridge and approaches are optimal for the following based on the survey data provided by GoviEx:

- Earthworks excavation on the approach, while maintaining the design criteria
- Bridge deck height and assumed flood levels
- Creating a southern approach reaching an elevation to facilitate overtopping, if necessary.

At the start of detailed engineering, a comprehensive construction methodology trade-off and a full geotechnical survey will be conducted. For the FS design, a composite construction method was chosen: a cast in-situ deck supported by a permanent shutter spanning between steel I-Beams. This method was selected over alternatives such as a cast in-situ solid deck or a composite deck with precast concrete beams due to site location and unknowns regarding concrete supply and Contractor capability and approach. During detailed design, bridge construction methodologies will be discussed with preferred Contractors to receive constructability inputs.

Another key feature of the FS bridge design is the southern approach embankment whose vertical alignment is at a lower level to the bridge, and with reinforced embankment slopes is intended to act as a relief point during any periods of flooding.

The main sources of reference/ guidelines used for the structural analyses and design were:

- CSRA, 1994, “Code of Practice for the Design of Highway Bridges and Culverts in South Africa”, Technical Manual for Highways (TMH) 7 – Part 1 and 2
- E. Kruger, 2002, “*Code of Procedure for the Planning and Design of Highway and Road Structures in South Africa*”, SANRAL, Pretoria, pp. 10.1 – 10.5
- South African National Standards, (SANS) 10100 – 1: Structural Use of Concrete.

The following loads were applied to the bridge to determine the maximum internal stresses and forces for designing the foundations, beams, and deck and the determination of stresses, shear and bending reinforcement steel. Traffic loads allowed for are as follows:

- Type NA loading – 36 kN per linear metre of the notational lane
- Type NB loading (Type NB36).

Dead load (self-weight) included deck slab, parapet, steel beam, piers, and abutments.

18.1.7.8. Fencing and signage

Speed limit and traffic signage will be deployed as well as appropriate signage to ensure the safe approach, entry, and operation on the road. Around 44 signs of various designs (according to the Zambian standards) are required according to the preliminary signage plan developed for the project.

18.1.7.9. Road polymer

The application of a polymer binder for road stabilisation and dust suppression was considered during the FS. An allowance for dust suppression treatment of the access road in the vicinity of the new clinic (“DUST/BLOKR” product from Cypher Environmental) has been made.

18.1.7.10. Other

Between 6+000 and 8+000, the road alignment is within the 500 m boundary around the final pit outline. GoviEx will locate a guard post and the entry point to this section and the road will be closed during blasting activities.

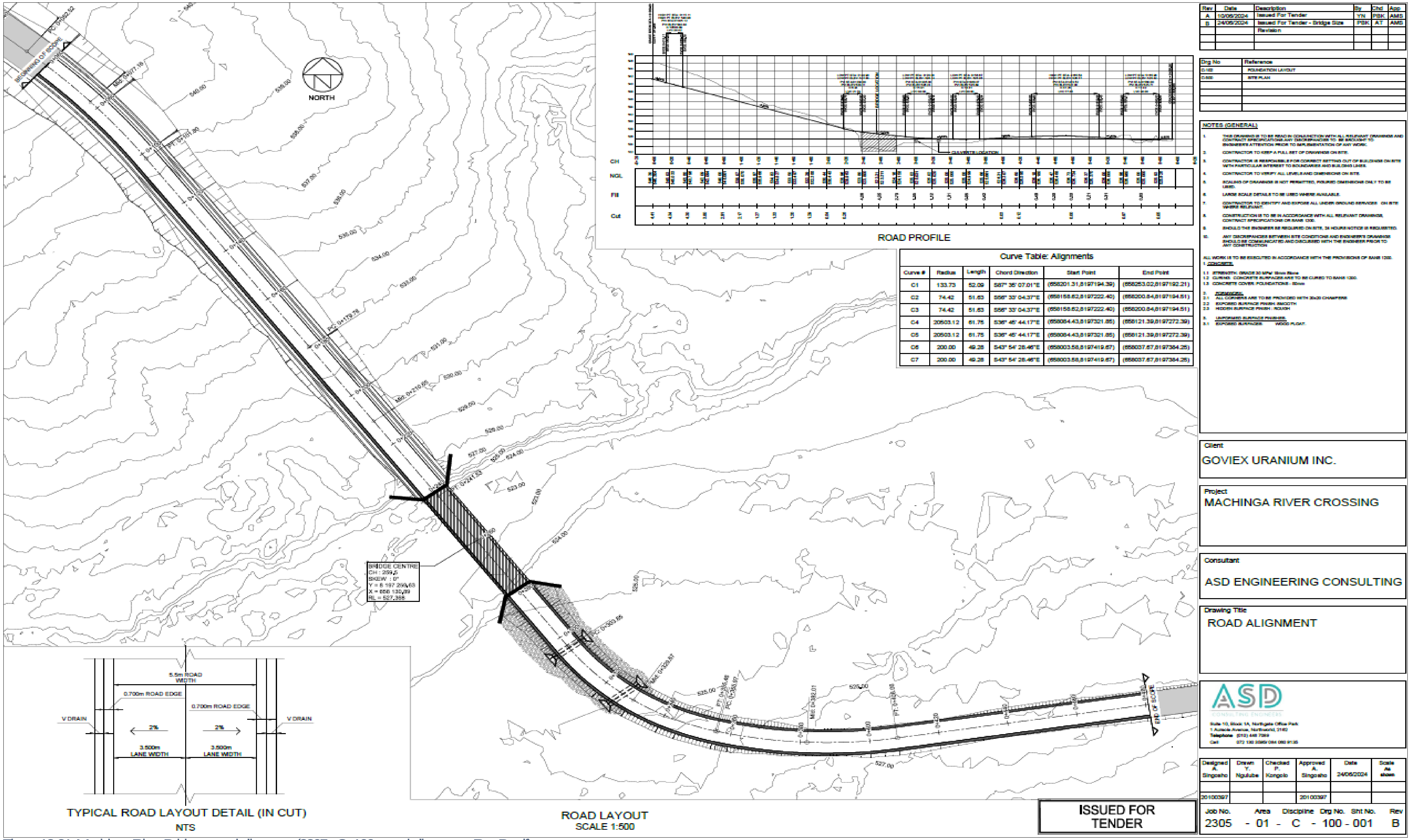

Figure 18-21: Machinga River Bridge – road alignment (2305 - C - 100 - road alignment - Rev B.pdf)

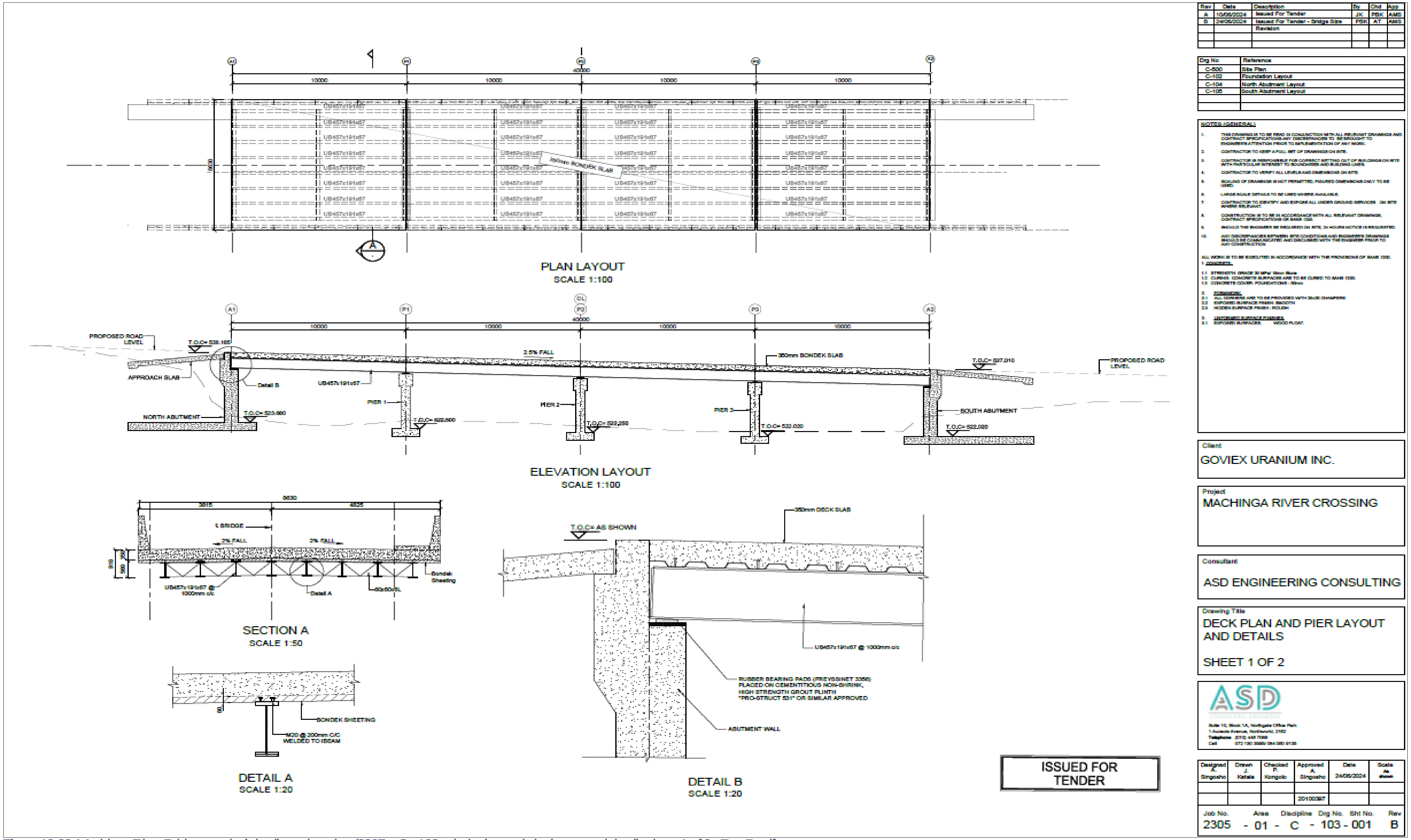

Figure 18-22: Machinga River Bridge – typical details and section (2305 -C - 103 - deck plan and pier layout and details sheet 1 of 2 - Rev B.pdf)

18.1.8.Dibbwi primary access road design

18.1.8.1. Design development

The Dibbwi access road will support a haulage operation. The existing access road alignment was taken forward into FS design. The road was designed in AutoCAD Civil 3D software in a single section. Multiple iterations were undertaken; the objective in the final iteration was to minimise cutting earthworks as GoviEx will seek to construct the road using waste rock as bulk fill. Therefore, earthwork embankment construction was preferred resulting in a significant “net site won material” balance. The only exception is the initial 2.35 km, which will be constructed as part of pre-production Capex, and this is to ensure a clear rerouting of the existing local road which connects the D500 to Siavonga around the mine on its western side.

18.1.8.2. Alignment

The access road has a total length of 8 920 m long as shown in Figure 18-23 and Figure 18-24.

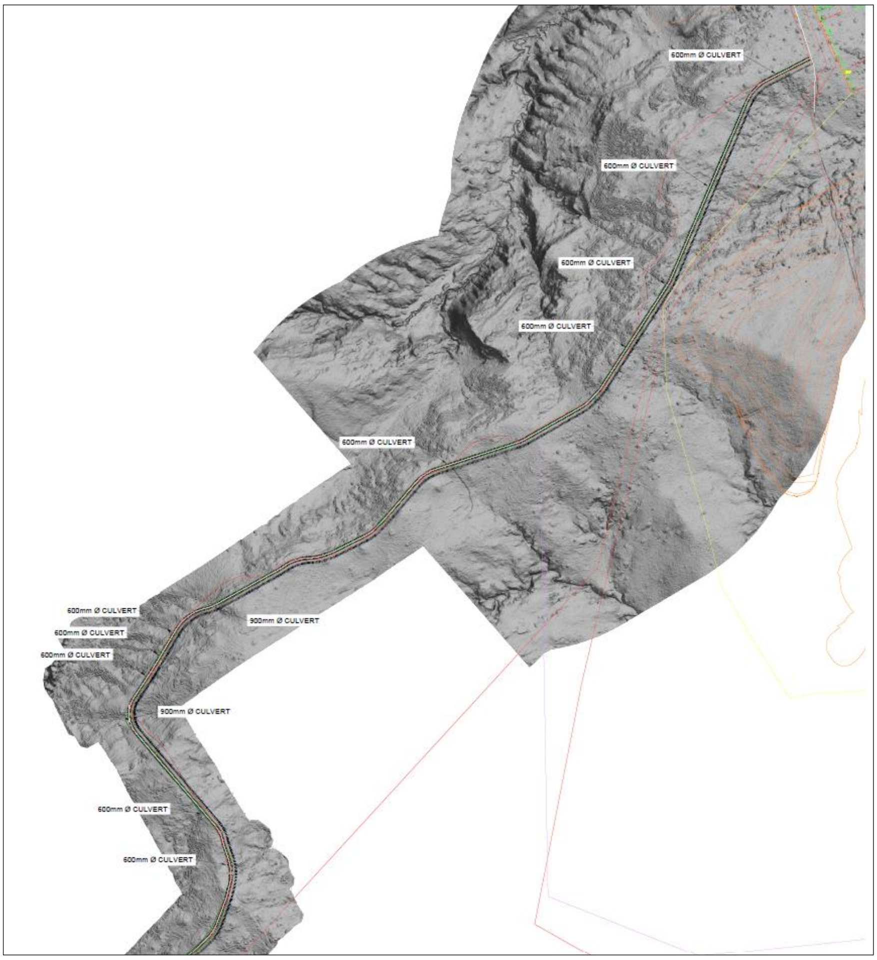

Figure 18-23: An image of the Dibbwi Road chainage 0+000 (D500 intersection) to 4+600 (31372-INF-003C_DibbwiHaulRoadDesign_V1_7m_Plan_20241202.pdf)

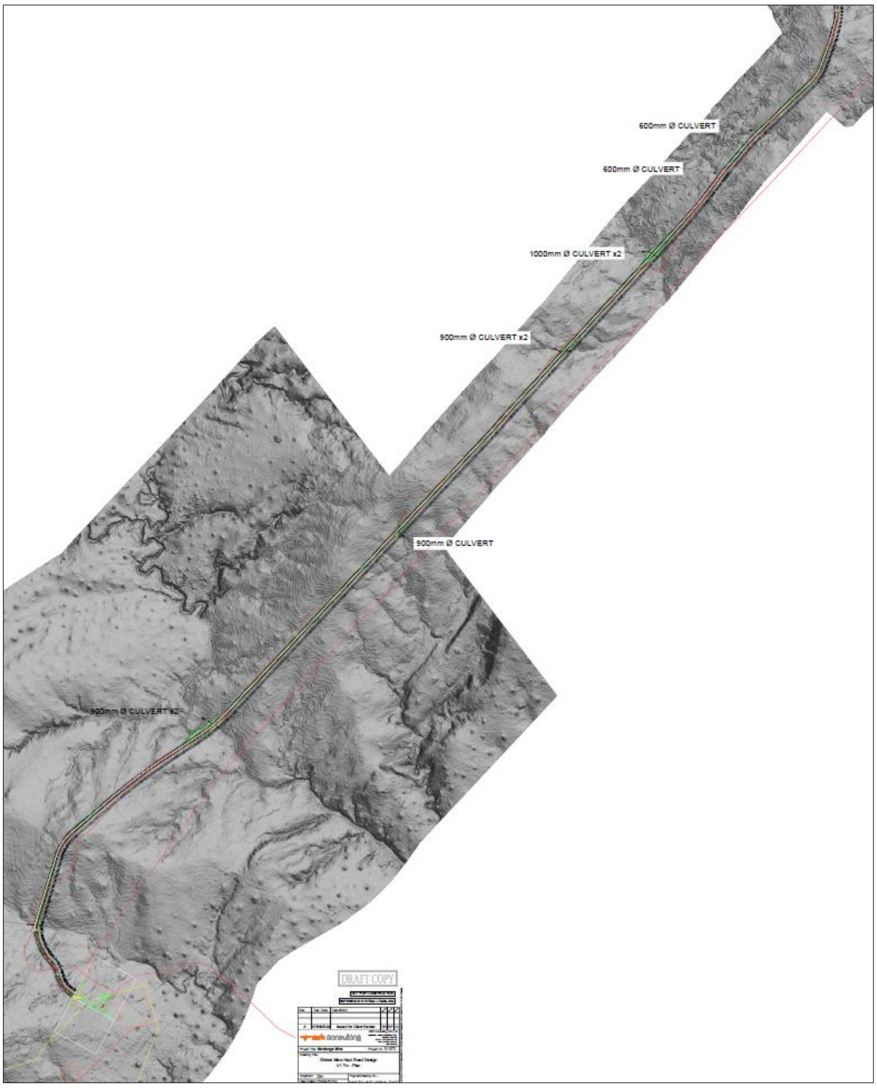

Figure 18-24: An image of the Dibbwi Road P1 Chainage 4+600 to 8+920 (31372-INF-003C_DibbwiHaulRoadDesign_V1_7m_Plan_20241202.pdf)

Road design parameters are presented in Table 18-6. The total change in natural ground level over the length of the road is approximately -32 m with a maximum gradient of 7 %. At the connection with the Muntanga access road, the elevation is 614 m ASL. After 600 m, the ground level rises to around 640 m ASL before gradually descending to 582 m ASL in the Dibbwi area.

18.1.8.3. Design vehicles and traffic

Primary usage (>90 %) will be for 4-axle rigid highway trucks with a 30 t to 40 t payload delivering crushed and sorted ore material from Dibbwi to Muntanga while Dibbwi is in operation. Other vehicles will be delivery trucks (similar loading) and light vehicles. Maximum truck width will be around 2.6 m.

18.1.8.4. Road geometry

See Figure 18-25 for the road geometry as developed in the design criteria.

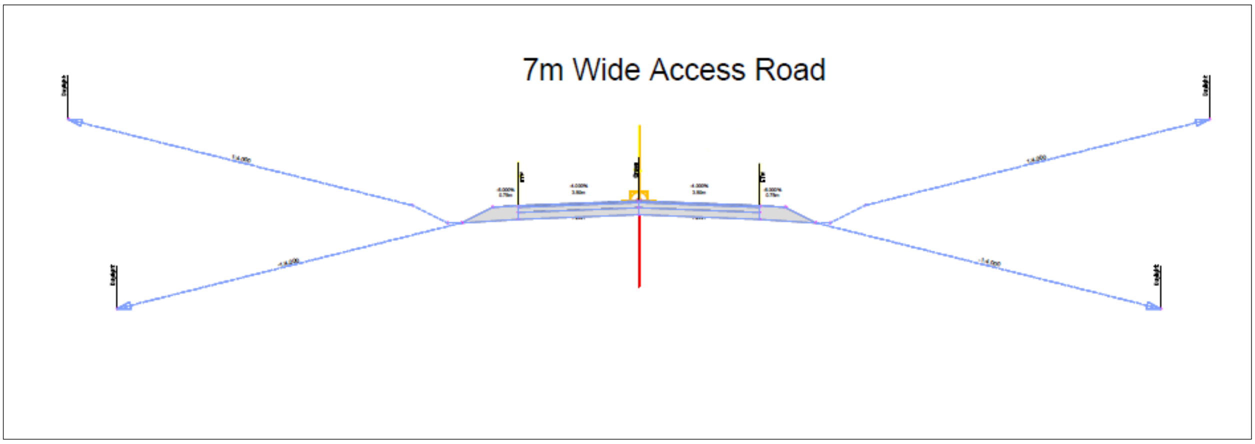

Figure 18-25: Road geometry – Dibbwi access road

18.1.8.5. Pavement design

Pavement design parameters are presented in Table 18-9. These are subject to field and laboratory trials of waste rock for use as a pavement aggregate and updated traffic analysis.

Table 18-10: Pavement design

Layer	Details
Base / wearing course of compacted crushed and graded waste rock to 95 % MOD.AASHTO equivalent to a G5	150+150 mm thick
Subbase course of compacted crushed and graded waste rock to 93 % MOD.AASHTO equivalent to a G7	150 mm thick
In-situ subgrade ripped and compacted 90 % MOD.AASHTO	150 mm depth

18.1.8.6. Earthworks

Vegetation will be grubbed out and topsoil across the footprint of the earthworks will be removed and stockpiled for future closure works. Topsoil is assumed as 150 mm thick.

Bulk earthworks will be conducted to achieve subgrade level. At the subgrade level, the material will be ripped, watered and compacted to form the sub-grade vertical profile to receive the engineered layers of the pavement. Materials deemed “unsuitable” for construction will be stockpiled.

The section 0+000 (Muntanga access road connection point) to 2+350 (RAP2 road connection point) will be constructed initially. FS design earthworks balance results in a broad balancing of suitable site-won material and fill requirements. There is predicted to be a surplus of around 1 000 m^3 assumed to be suitable for pavement layer construction.

The FS design earthworks balance for 2+350 to 8+920 results in a net deficit of 53 000 m^3 which needs to be balanced with waste material from mining operations. This waste material will be used as bulk fill (uncrushed run of mine) and when crushed and screened used for subbase and base course construction to facilitate construction activities.

It is intended that inert waste rock stockpiles are used for material for re-sheeting and replacement of gravel wearing course over the LOM. Samples of these stockpiles will be tested to determine the quality of material that can be produced once crushed, screened and blended.

18.1.8.7. Culverts/ Drainage

A grader-formed V drain will be constructed on the upslope side of the road to direct overland stormwater flows to pipe culverts that are positioned along the route.

The series of culverts are installed to manage surface water flows and the detailed topography has been used to locate the culverts. Three principal pipe culvert sizes have been adopted with box culverts (“major culverts) used at specific locations:

1. Type 1 Culverts are single, double or triple 600 mm ID concrete pipe culverts with concrete head and wing walls and inlet and outlets.
2. Type 2 Culverts are single, double or triple 900 mm ID concrete pipe culverts with concrete head and wing walls and inlet and outlets.
3. Type 2 Culverts are single, double or triple 1 200 mm ID concrete pipe culverts with concrete head and wing walls and inlet and outlets.

Major culverts have been considered in more detail and incorporate one or more 1 200 m width box culverts (pre-cast or cast in situ) with a reinforced concrete deck, with concrete head and wing walls and inlet and outlets.

18.1.8.8. Fencing and signage

Speed limit and traffic signage will be deployed as well as appropriate signage to ensure the safe approach, entry, and operation on the road.

18.1.8.9. Dust suppression

When the Dibbwi ore haulage operation commences, dust suppression will be via mobile water bowser.

18.1.9.Camp access road

The scope also included the 1.25 km access road to the GoviEx Mine Camp and the estimation of bulk earthworks required to form the development platform for mine camp construction. GoviEx has procured quotations for turnkey camp construction including all works upwards from the base of civils.

18.1.10. Resettlement action plan/ Detour road design

The following RAP roads were designed (Table 18-11) and in accordance with the design criteria set out in Table 18-6. The location of the roads is presented in Figure 18-1. These roads will be utilised to provide access to the villages for construction.

Table 18-11: Summary of RAP roads

Item	Units	Value
RAP1 Access road, Muntanga area		
Total length	**m**	**6 790**
Pavement surface area	m^2	20 370
Net elevation gain	m	+50
RAP2 Access road, Muntanga area		
Total length	**m**	**4 250**
Pavement surface area	m^2	12 750
Net elevation gain	m	-45
Kashunde link road, Muntanga area		
Total length	**m**	**4 950**
Pavement surface area	m^2	14 850
Net elevation gain	m	7
Njame RAP road, Njame area		
Total length	**m**	**4 750**
Pavement surface area	m^2	14 250
Net elevation gain	m	18
Gwabi detour road, Gwabi area		
Total Length	**m**	**2 100**
Pavement surface area	m^2	6 300
Net elevation gain	m	5

18.1.11. Project execution plan

18.1.11.1. Implementation

Construction will be undertaken under an EPCM. GoviEx will appoint an engineering company to undertake detailed design, manage the construction contractor tendering process, and monitor construction as the owner's engineer.

The objective of the implementation plan is to achieve the construction of the primary access road to the sub-base level, including a temporary low water crossing at the Machinga River, to facilitate the import of civil construction.

Overall construction durations are derived from proposed schedules from potential construction contractors included in pricing enquiries.

Package breakdown

The Project comprises four packages:

1. Construction of Muntanga PAR chainage 0+000 to 2+900
2. Construction of Muntanga PAR chainage 2+900 to 10+150, and the short proportion of the Dibbwi PAR as a local road
3. Construction of the RAP1 access road
4. Construction of the Machinga Bridge.

The Muntanga PAR is split into two sub-packages because GoviEx is understood to already be able to commence construction on chainage section 2+900 to 10+150 (and Dibbwi 0+000 to 2+350) under the current permits but are still investigating options to commence construction of chainage 0+000 to 2+900 at a similar time.

Responsibilities

GoviEx will provide the following:

- Selection and securing of land and right of way for the OHL
- Construction permits for construction
- Funding the works.

GoviEx's engineer will be responsible for engineering, procurement, review of construction management plans, and construction monitoring.

The construction contractor will be responsible for the construction of the works package including construction management plans, temporary works, traffic management, health and safety and environmental controls, and quality testing.

18.1.11.2. Construction schedule

A construction schedule has been developed for the FS. The schedule is reliant on all necessary approvals and permits being in place. RAP1 access road must be in place to facilitate the construction of the RAP village.

18.1.12. Risks

18.1.12.1. Geotechnical conditions and assumptions

Limited site-specific geotechnical and ground investigation works have been carried out along the road alignment, and more is needed to inform detailed design, especially for bridge foundations and any significant cutting earthworks and culverts. As such there are uncertainties and assumptions related to the following that need to be proved:

- Topsoil thickness and quantities
- Suitability of site won material for re-use in construction.

18.1.12.2. Machinga River bridge

The following aspects of the bridge need further consideration in the detailed design phase:

- Ground conditions
- Bridge foundations
- Temporary works and temporary river crossing
- Requirement for any up-stream or down-stream channel works
- Hydrological and hydraulic conditions during rainfall events.

Bridge deck height is informed by site observations related to likely flood levels and bridge and approach earthwork levels to the south are designed to provide a point of overtopping, if necessary, in flood conditions.

18.1.12.3. Transport of run of mine

During detailed design, the traffic analysis and pavement should be updated if ROM from Njame and Gwabi and transported on the Muntanga access road to specifically identify the required SOWs required in year 9 to ensure the Muntanga access road is upgraded to cope with this increase in traffic or additional allowances for increased maintenance are provided for.

18.1.12.4. Requirement to import layer works materials

From site observations, some of the areas of required cutting earthworks are anticipated to be in hard ground and suitable for re-use. The earthworks were designed to maximise the reuse of site-won materials as subbase and base-course, subject to geotechnical investigation work. However, there is a risk that additional imported materials may be required, and this could increase the cost of pavement construction.

18.1.12.5. Tree survey

In the design of the road, baobab trees were seen during the site visit and were avoided; however, a full survey is required to establish the location of any significant trees along the route and how the alignment can be best modified to mitigate the impact.

18.1.12.6. Typical construction risks

Typical civil and road construction risks must be assessed and mitigated through a risk assessment and preparation of management plans for all aspects of the works.

18.1.13. Opportunities

18.1.13.1. Culvert type

Precast concrete culverts are assumed in the design and costing. There is an opportunity to consider corrugated metal arch or pipe culverts or HDPE pipe culverts in various locations.

18.1.13.2. Polymer road binder

Once more geotechnical information is collected, the cost-effectiveness of incorporating a road stabiliser into the pavement layer works can be assessed. The application of a road stabiliser might be best applied on cessation of main phase construction works before the repair and resurfacing of the road.

18.1.13.3. Cutting slope angles

Once more geotechnical information from the surveys is undertaken to inform detailed design, there may be an opportunity to increase the slope angles for cutting slopes, which may reduce earthwork volumes; however, this would need to be considered in the detailed design with regards to earthworks balance and availably of material for pavement layering.

18.1.13.4. Borrow pit

An existing local borrow pit is located north of the D500 and should be investigated as a source of capping and subbase material.

18.1.14. Recommendations and next steps

The next steps are as follows:

- Commission a geotechnical investigation along the Muntanga access road (including but not limited to drilling at the location of the river bridge, the cutting north of the river bridge, and major culverts)
- Procurement of an engineer to undertake detailed design of the Muntanga access road and Machinga River Bridge and tender documents for the construction
- Detailed design should also include an investigation into the cost-effectiveness of a road stabiliser and the use of a chip-seal pavement methodology rather than unbound aggregate
- Commission a condition survey of the D500 road for future reference
- Confirm parameters for the use of waste rock in pavement layers
- Install flow gauges at the Machinga River Bridge and weather station at Muntanga.

18.2. Mining infrastructure

This section details the central mining complex, encompassing all mining-related infrastructure supporting operations at the central plant complex, and the Muntanga and Dibbwi East pits. The overall project infrastructure scope was developed collaboratively by several parties. This work is based on the recommendations and inputs provided by various specialists and consultants involved in the Project.

Potable, fire, and service water are supplied to the mining infrastructure area through connections with the processing plant's water distribution network. Sewage generated within the mining area is collected and pumped to the processing plant's sewage treatment system.

A number of the construction activities has been assigned to the mining operations team for execution. These activities, which include items such as site clearance, require meticulous planning and seamless integration into the operational readiness plan. This is crucial to prevent delays, cost overruns, and potential shortfalls in project delivery.

While substantial effort has been invested in developing facilities that meet the operational requirements, including the efficient sharing of facilities and resources, further opportunities exist for complete integration of activities and resources. By fully integrating these elements, significant synergies can be realised, potentially leading to substantial improvements in efficiency, cost reduction, and overall project success. Therefore, it is strongly recommended that all processes and activities be comprehensively reviewed as a holistic system to identify potential areas of optimisation and further enhance the project's value.

Reference is to be made to the project block plan, drawing number GUI534-M-A000-G-001. The block plan layout is designed to support mining operations while considering environmental and social factors. A new access road is to be developed from the D500 road, leading to the main access of the mine. Haul and access roads connect various areas and infrastructure within the site. Surface water management is addressed through a network of dams, including an excess water dam, an attenuation dam, and PCDs, strategically placed to manage surface water flow and potential runoff. Power is supplied through new lines routed from Kariba in the South, passing along the West of the Dibbwi East pit. Security measures include access control points and a fence line. Blasting control points are located near the mine, where roads enter into the blasting radius. The plan acknowledges existing community elements such as villages, agricultural fields, and individual graves, indicating an effort to minimise impact. Buffer zones further demonstrate this consideration. The layout prioritises functionality, with infrastructure strategically placed to support mining operations and water management, while aiming to minimise environmental and social impacts.

18.2.1. Perimeter fencing

The entire Project mining operation, including mining, waste dumps and processing complex, will be enclosed by a secure perimeter fence with controlled access points. This fence serves as a critical safety barrier to prevent unauthorised entry and ensure the security of the site.

The fence will be a robust 2.1 m high Bonnox Close Mesh 1984/6 model, topped with four strands of barbed wire for added security. Clear signage indicating an active mining area will be prominently displayed at 50 m intervals along the fence line (refer to Figure 18-26 for typical fence detail).

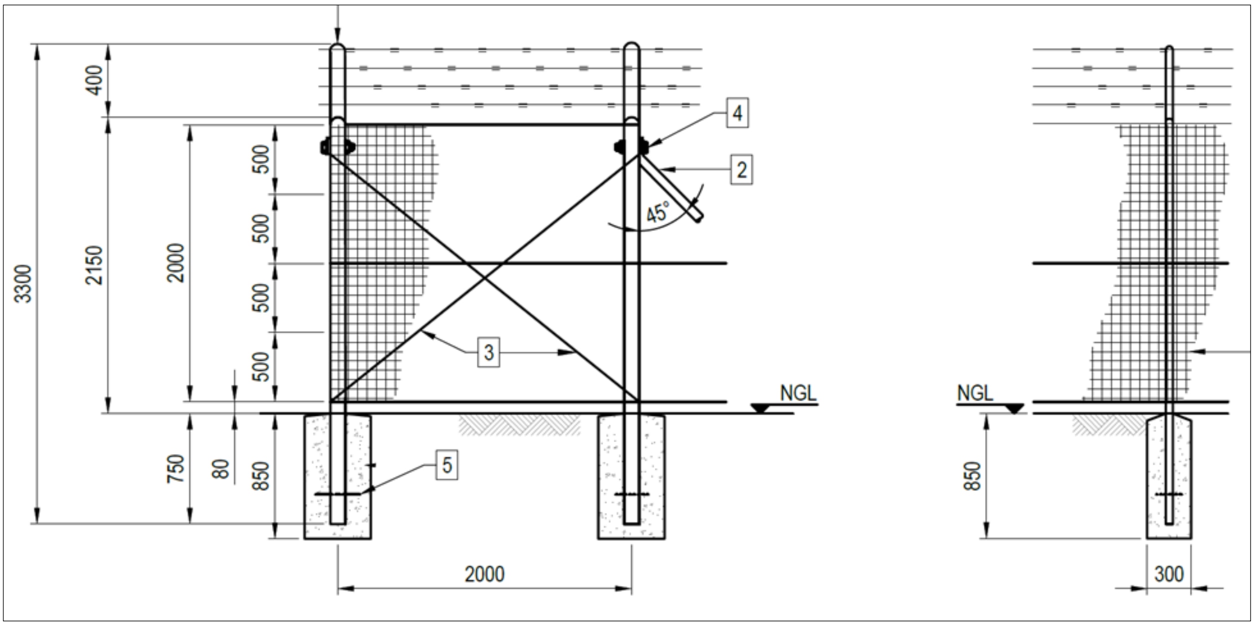

Figure 18-26: Perimeter fence typical detail

The fenced area encompasses all active mining zones and associated infrastructure, generally aligning with the 500 m blasting radius. While a few isolated operational areas fall outside this perimeter, these have been designated as "special areas" due to their temporary use. These special areas will be carefully patrolled and cleared prior to any blasting activities.

A section of the main access road falls within the 500 m blast radius of the Muntanga pit. To manage this, guard huts and traffic control points will be strategically positioned just outside the blast zone on both the north and south sides of the pit. These control points will be used to temporarily close the road during blasting operations. Following each blast, the road will be thoroughly inspected for debris and reopened only after it is deemed safe. Figure 18-27 illustrates the proposed fence line, blast radius, and the locations of these control points.

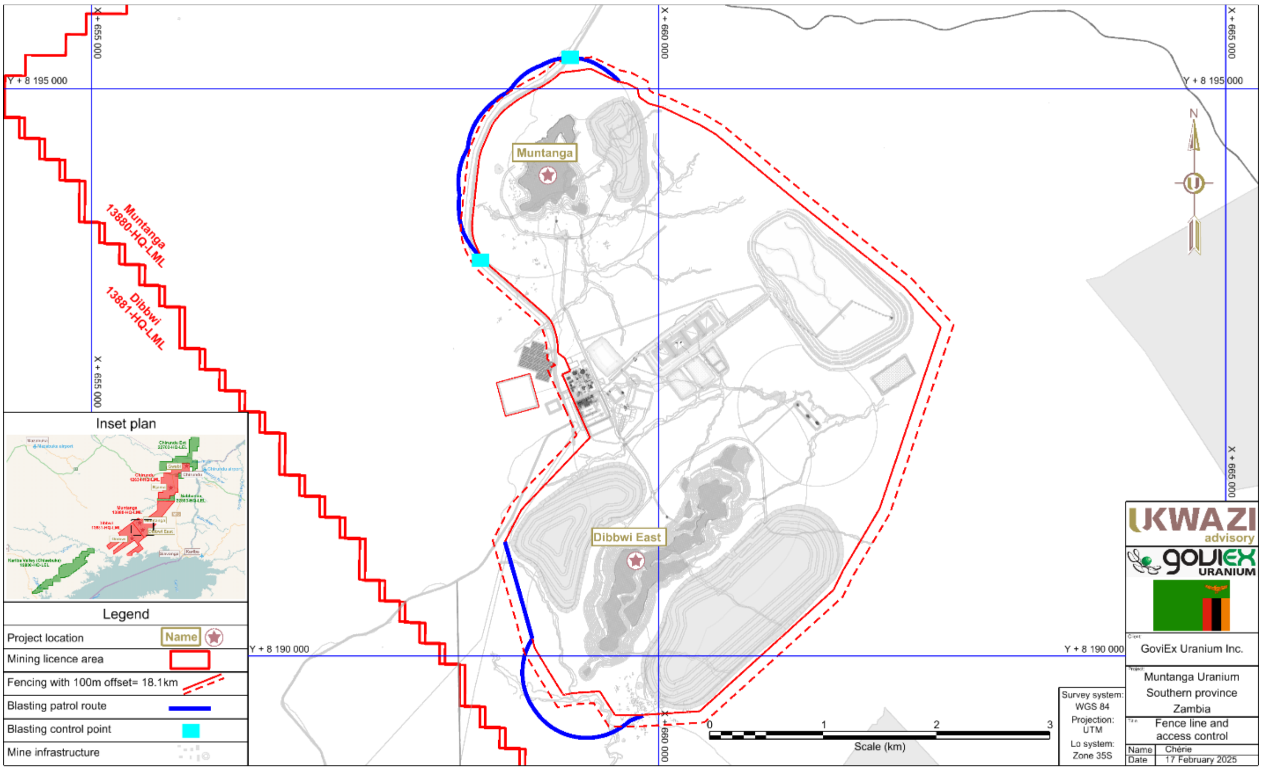

Figure 18-27: Fence line and access control

To facilitate maintenance and inspections, a 5m-wide cleared strip will be maintained along the inside of the fence line, accommodating a 3 m-wide access road. Figure 18-28 shows a typical section through the fence line.

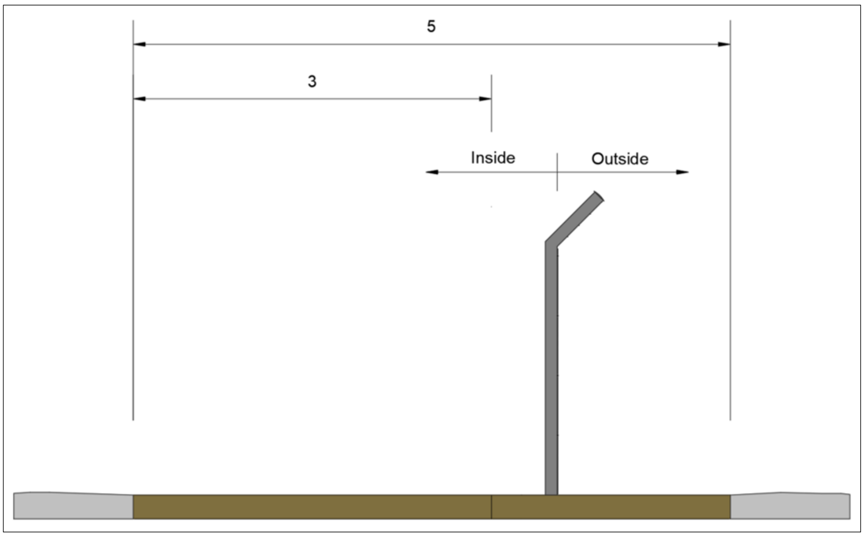

Figure 18-28: Section through fence line

The presence of local wildlife, including elephants, has been taken into consideration. The fence height is designed to deter common antelope and other medium-sized animals. Before the final closure of the fence, efforts will be made to encourage wildlife to leave the enclosed area. Any remaining animals within the perimeter will be managed in accordance with local wildlife management and conservation guidelines.

While no specific elephant deterrents are initially planned, the situation will be closely monitored. If necessary, electrified strands will be added to the outside of the fence to prevent elephant interactions.

18.2.2.Haul roads

The haul road network is designed to ensure safe and efficient transportation between various points of interest on site. This network is divided into two categories: primary and secondary roads. Primary roads, designed to a higher standard with engineered layer works, accommodate high traffic volumes and connect key areas like the pits, primary tip, and mining maintenance area (Figure 18-29). Secondary roads, with a less permanent construction method consisting of a cleared travel way with minimal cut and fill, support lower traffic volumes.

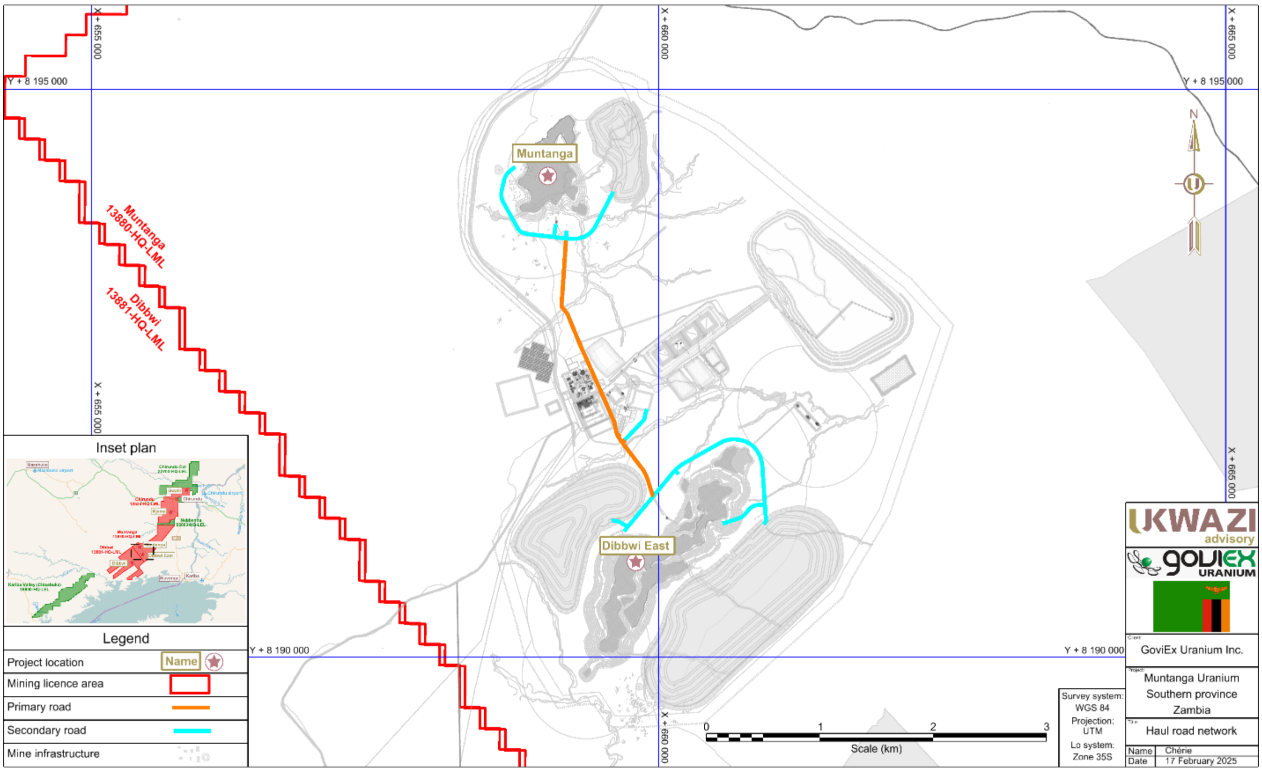

Figure 18-29: Haul road network

As described in the mining section, haul roads are designed with a width of 3.5 times the width of the largest haul truck for bi-directional traffic and 2.5 times for one-directional roads. The selected fleet consists of 40t ADTs with an overall width of approximately 4 m, requiring a 14 m wide travel way for bi-directional haul roads. Figure 18-30 provides a typical cross-section of the primary haul roads, showing the design and dimensions. Centre berms are incorporated at all intersections, ramps, and horizontal curves to enhance safety.

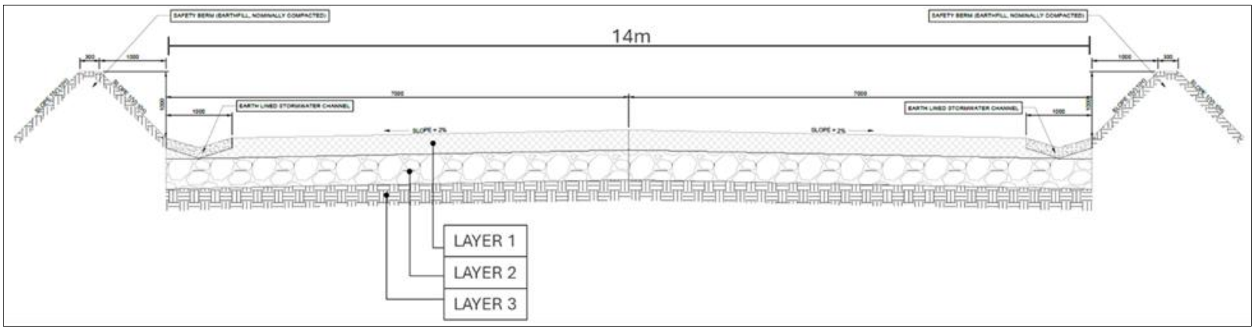

Figure 18-30: Typical primary haul road section

The recommended pavement design (Table 18-12) aims to minimise initial road maintenance and provide a durable, maintainable surface.

Table 18-12: Haul road pavement layerworks

Layer number	Specification
Layer 1	2 x 150 mm thick layers consisting of 1-part G7 material from borrow pit and 2 parts 50 mm crushed rock measured by volume. Compacted to 98 & mod AASHTO at OMC.
Layer 2	3 x 150 mm thick layers consisting of G7 material from borrow pit compacted to 95 % mod AASHTO
Layer 3	In-situ material compacted with 10 t roller using 8 passes

To reduce capital costs, the pavement layerworks for secondary haul roads have been minimised to clearing and shaping only and will be constructed by operations as part of the operational readiness activities. Primary haul roads will consist of a minimum of 450mm of G7 material, compacted in 150mm layers, followed by the application of a dust suppression emulsion.

The hydrological study (Section 18.3) identifies drainage crossings that would be impacted by the haul roads. A culvert crossing has been designed and included for each identified location. Refer to Ukwazi drawing GUI534-M-Z600-C-101 for the locations of the identified culverts. These structures typically consist of precast pipe culverts with reinforced concrete inlet and outlet structures. Energy dissipation measures have been incorporated to limit erosion. Figure 18-31 provides a typical pipe culvert haul road crossing design.

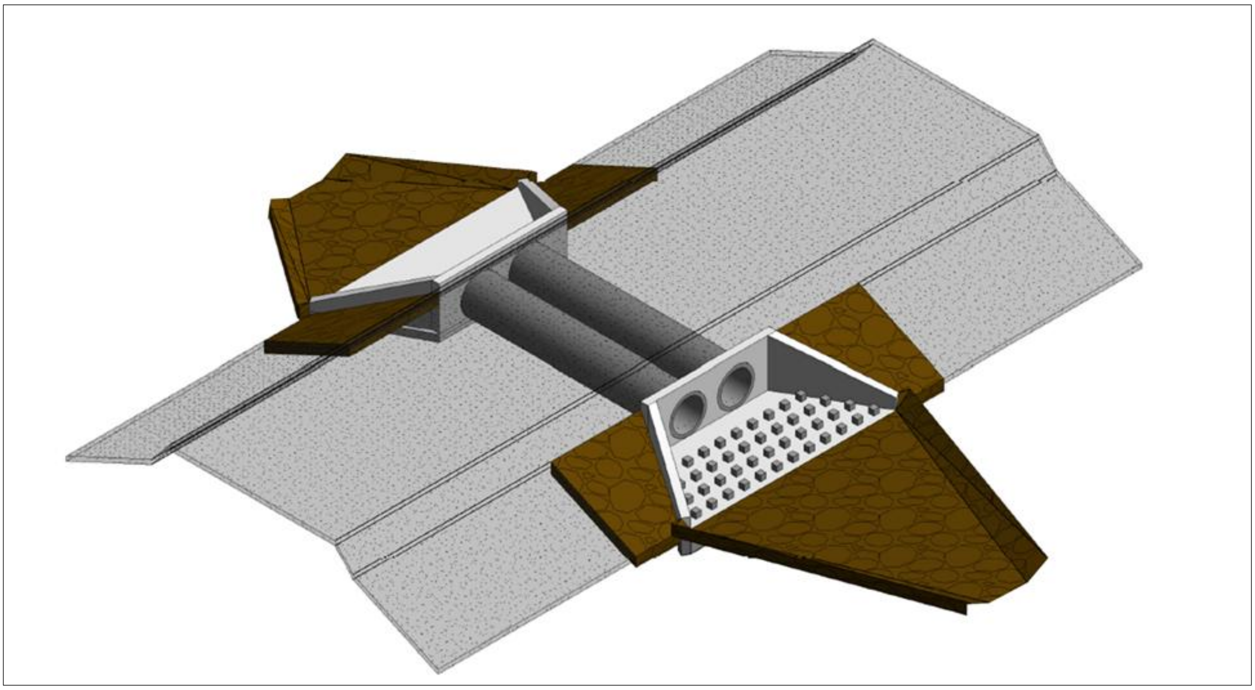

Figure 18-31: Typical pipe culvert crossing

While an initial application of dust suppression emulsion is included, no allowance has been made for ongoing application of dust suppression agents. Dust suppression will utilise water accumulated within the pits or collected in the PCD. The use of raw water for dust suppression should be avoided as far as reasonably possible. It is recommended that the use of emulsions or other suppression agents be investigated for implementation, as these can significantly reduce the volume of water required compared to conventional water suppression. Refer to Section 18.2.7 for further information on dust suppression systems.

No specific allowance has been made for brake test ramps in this study, as the proposed client requirements only call for static brake testing. It is recommended that the need for dynamic brake testing be investigated by means of a risk assessment. Space has been allocated for the construction of a brake test ramp at the infrastructure complex. Should the need arise, additional brake test ramps could be constructed adjacent to the primary haul road en route to the Muntanga and Dibbwi East pits.

18.2.3.Primary run of mine tip platform

The primary ROM tip platform is designed to support the primary ROM tipping bin and its associated primary crusher. Figure 18-32 shows the platform in plan view.

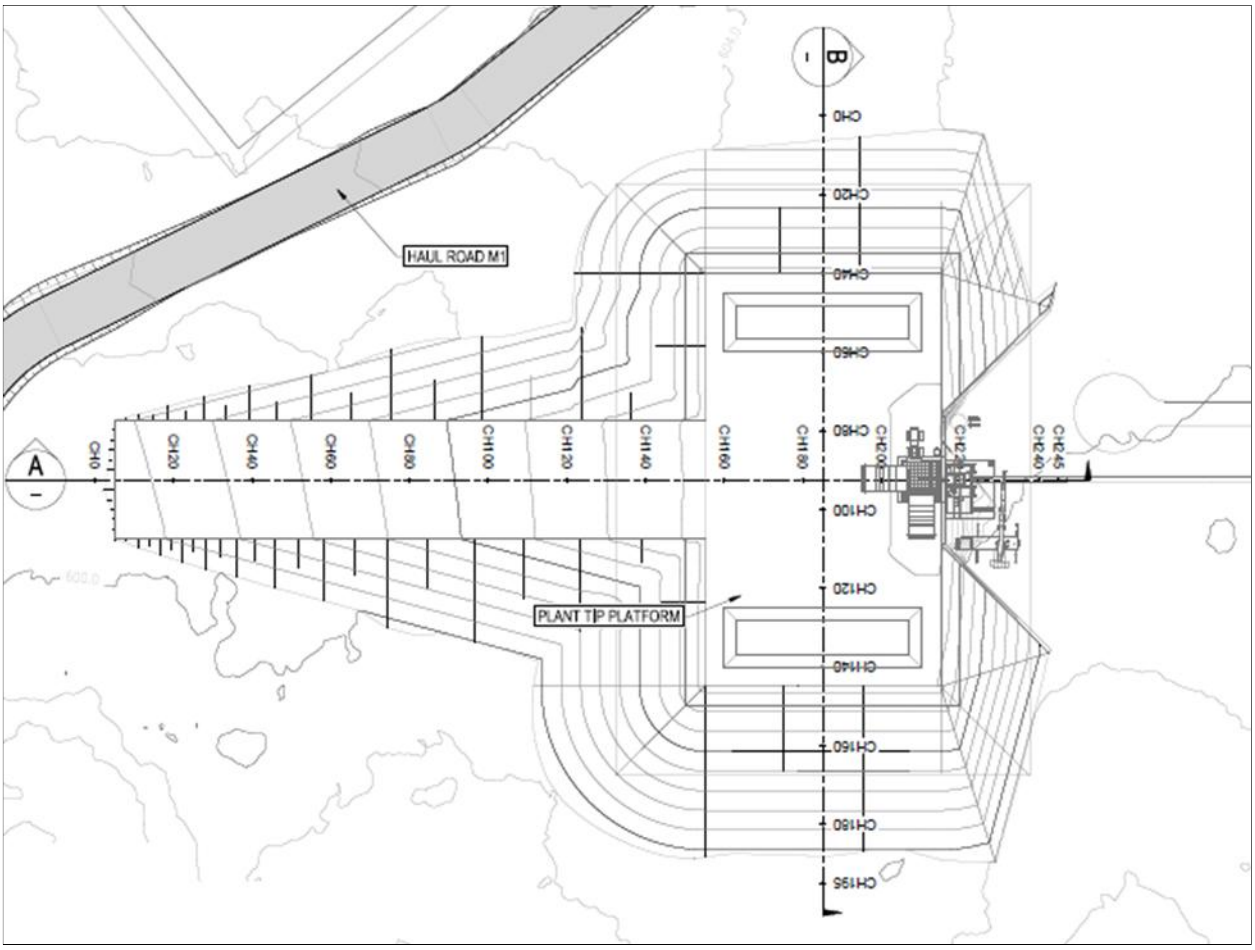

Figure 18-32: Plan view of ROM tipping platform

Both the tip platform and ramp will be constructed using selected mining waste material with a nominal particle size of less than 200 mm in diameter. Material will be placed in layers not exceeding 300 mm in thickness.

To optimise project execution, construction of the tip platform and ramp has been removed from the civil construction contractor's scope and assigned to the mining operational team. Therefore, no capital cost has been included for this work. Special attention shall be given to the type and fragmentation of the material sourced for the construction of the platform.

The three-sided bin, requires access from three sides. The platform is designed to accommodate two small, temporary ROM stockpiles and their associated recovery areas. These stockpiles provide operational flexibility by allowing trucks to continue tipping during upset conditions when the primary tip is unavailable. Safety berms will be constructed along all edges of the tip platform, and a centre berm will be installed on the ramp to enhance safety.

18.2.4. Infrastructure complex

The mining infrastructure complex houses all mining operational offices and mining-specific infrastructure. To manage personnel safety effectively, the complex is divided into two distinct zones. Figure 18-33 shows the layout and extents of the two zones. The office area, including the processing plant's general offices, is isolated from the active mining area with controlled access to ensure safety.

All mining maintenance infrastructure—workshops, wash bays, and the service station—are strategically positioned to facilitate the safe movement of both personnel and equipment. This layout allows for most services to be centralised within a single corridor, improving efficiency and accessibility.

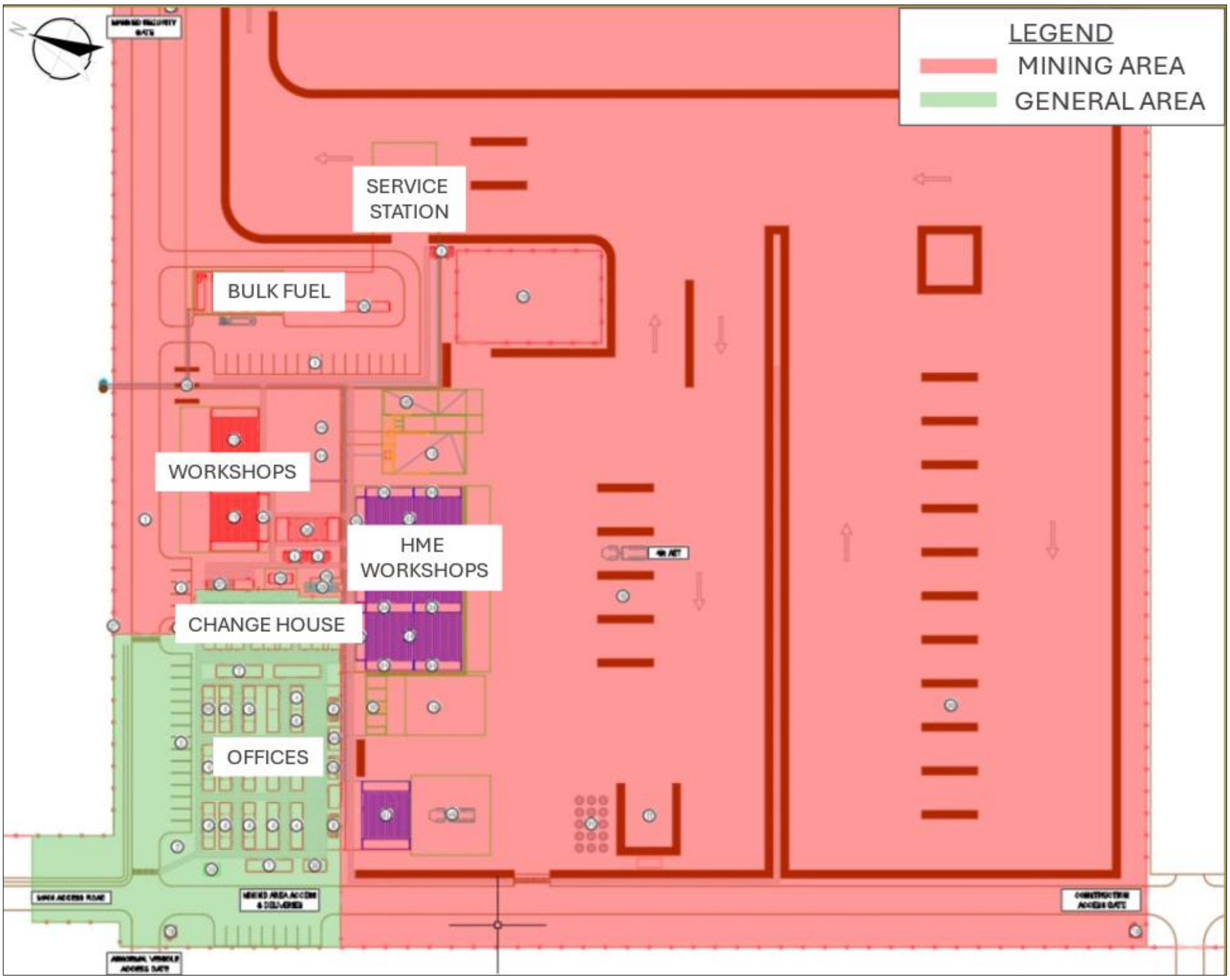

Figure 18-33: Mining infrastructure complex

The complex is primarily accessed through the main entrance gate located at the processing plant. An additional abnormal load gate allows for the delivery of large equipment without disrupting the main access point.

Internal roads within the complex are clearly marked travel ways on the infrastructure terrace. These roads are demarcated with small berms or windrows and appropriate signage. No additional layerworks are required beyond the terrace construction described below.

A single terrace serves as a platform for all infrastructure within the complex. As all structures and infrastructure are classified as light to medium load, a homogeneous layerworks design is employed across the entire area. This design also accommodates the movement of heavy mobile equipment while ensuring a trafficable and stable surface.

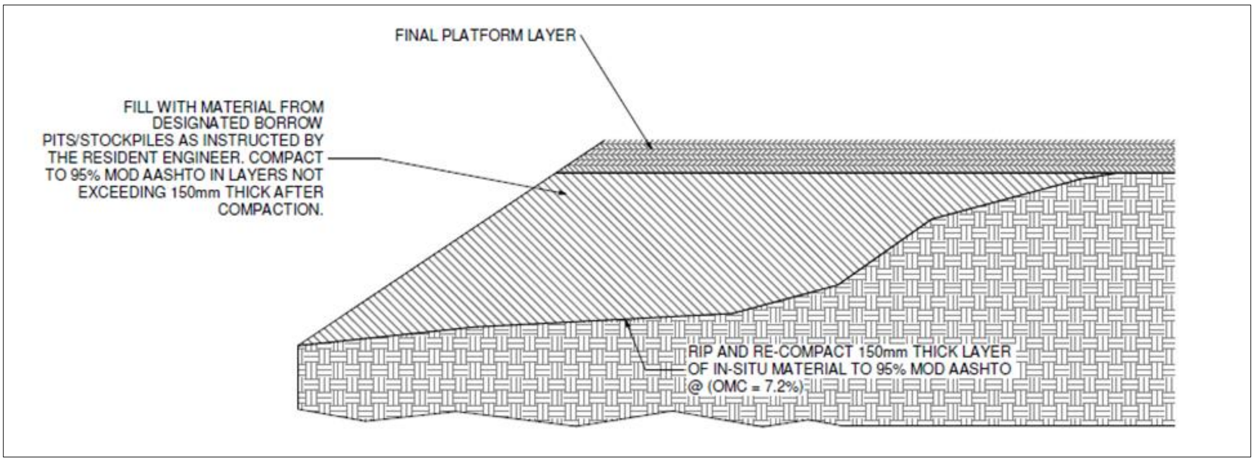

Figure 18-34: Typical infrastructure terrace layerworks

G7 bulk fill material was selected for the terrace layerworks to provide the necessary ground bearing pressures while minimising sourcing and production costs. The terrace is designed to integrate with the adjacent processing plant terrace. To minimise earthworks, cut and fill operations have been balanced as far as reasonably practicable. Topsoil will be stripped prior to earthworks and used to construct perimeter berms, which will be maintained for future rehabilitation activities.

18.2.5. Mining infrastructure complex stormwater and pollution control

The site-wide hydrology and surface water management is covered under Section 18.3. This section focuses on the stormwater and pollution control measures from the mining infrastructure complex area.

The heavy moving equipment ("HME") hardstand and maintenance area is considered a dirty water area due to the potential for oil spills and the presence of fine particulate matter. Runoff from this area is captured and managed within a dedicated dirty water system. The terrace slopes northeast towards the PCD, directing runoff towards the concrete-lined channels of the dirty water system. Figure 18-35 shows the terrace plan and stormwater control measures.

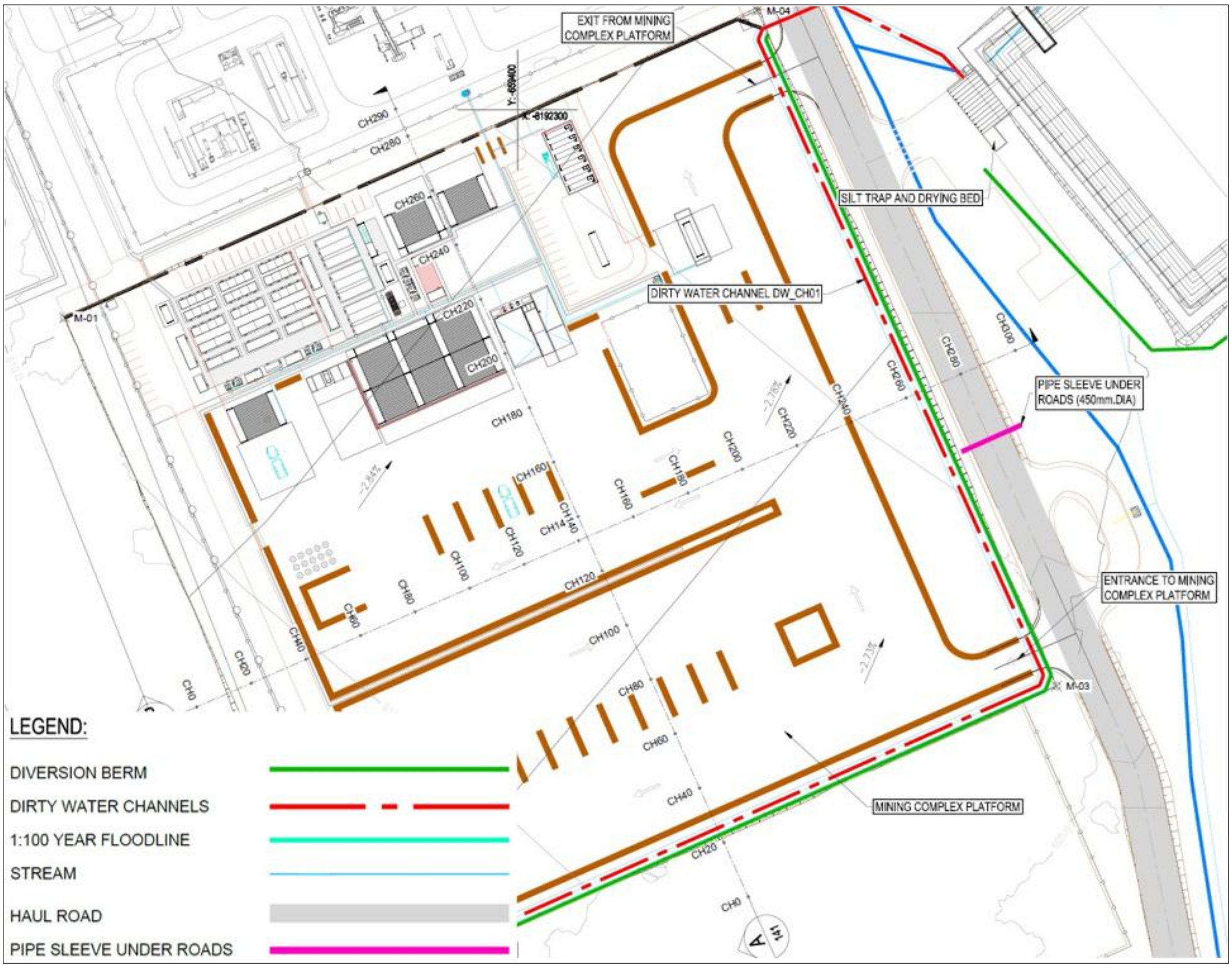

Figure 18-35: Infrastructure complex terrace and stormwater controls

All dirty water runoff from the infrastructure complex terrace will report, via a silt trap, to the PCD. The PCD is a fully lined facility, and the liner requirements comply with South African legislation. The liner system consists of a 2 mm high density polyethylene (“HDPE”) liner, atop a geosynthetic clay liner.

The PCD is sized to manage runoff exclusively from the mining infrastructure complex. Stormwater management for the processing plant area is addressed separately within the plant's infrastructure area. SRK determined the size of the PCD, with further details available in Section 18.3. Table 18-13 summarises the characteristics of the PCD.

Table 18-13: PCD characteristics

Item	Value
Type of Dam	Earth embankment
Lining System	2 mm HDPE geomembrane with geosynthetic clay liner
Gross Full Supply Level (“FSL”) Capacity (m³)	27 620
FSL Area (m²)	9 609
Embankment Height (m)	2.23
Embankment Slope (upstream) (V:H)	1:3
Embankment Slope (downstream) (V:H)	1:3

Item	Value
Length of Crest (m)	414
Spillway type	Broad crested
Spillway length (m)	2
Purpose	Stormwater management: pollution control

The silt trap, a reinforced concrete structure designed for easy cleaning (Figure 18-36), incorporates 100 mm x 16 mm steel plates cast into its base for enhanced durability and longevity.

Water enters the silt trap via either the dirty water channel system or a stilling chamber, which receives water from the pit dewatering system. The silt trap base is designed for a 1-in-10-year return period, while the overflow weir is designed for a 1-in-100-year return period. A length-to-width ratio of 1:5 was targeted for the silt trap where feasible. An adjacent silt drying bed allows wet silt to be deposited and dried before final disposal.

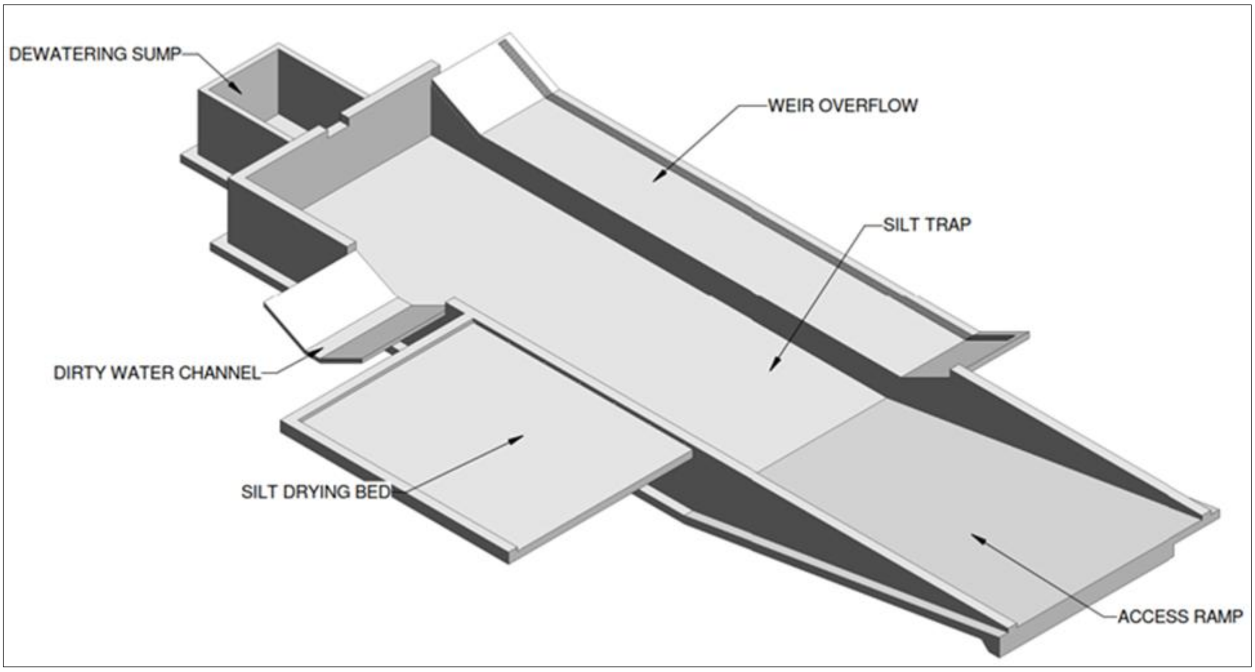

Figure 18-36: PCD silt trap

The PCD includes an emergency spillway designed to safely discharge water during storms exceeding the design return period (1:100 year flood event). A stilling basin at the spillway's base minimises hydraulic jump effects and erosion. As shown in Figure 18-37, the spillway consists of a Reno mattress with edge beams.

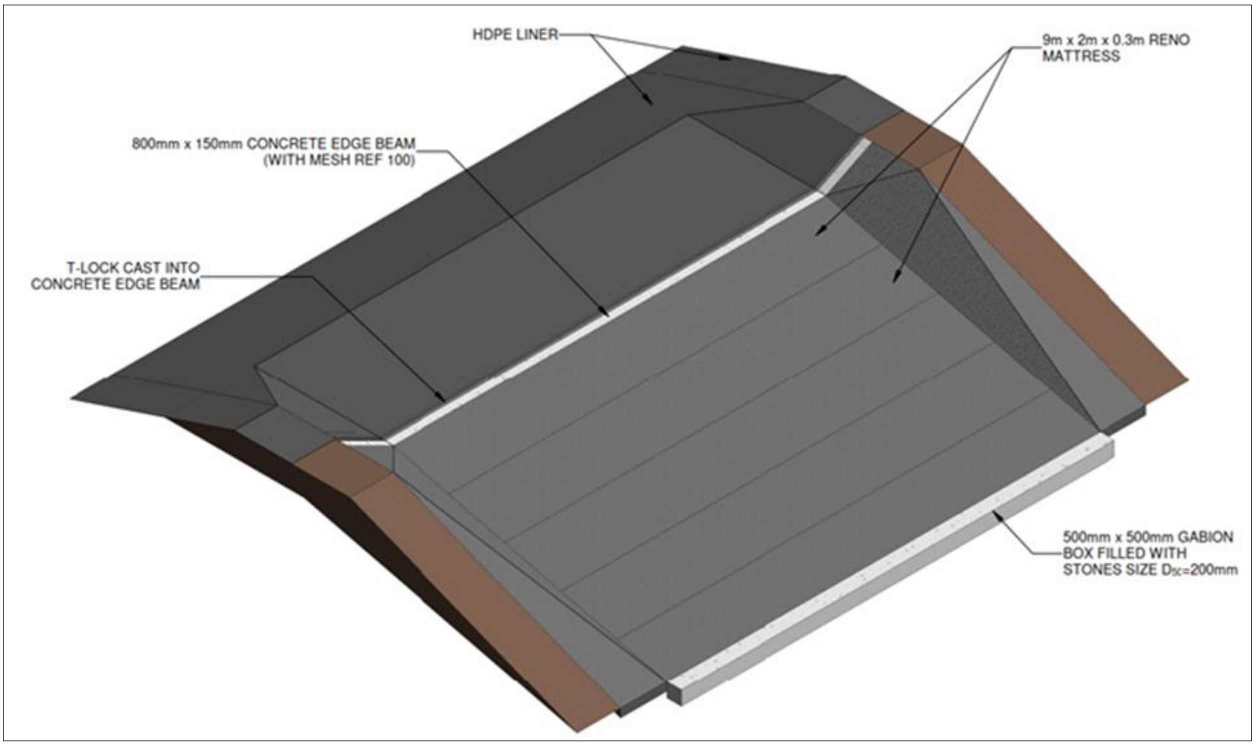

Figure 18-37: PCD emergency spillway

18.2.6. Pit dewatering

Pit dewatering is required due to the inflows from ground water and from surface runoff water. The ground water inflows are reduced with the use of dewatering boreholes around the pit. This section provides information regarding the pit inflows and dewatering boreholes.

The pit water inflows are accumulated in pit sumps that will progress with the mining face. The in-pit sumps are sized as storage buffers for groundwater inflows and surface run-off from storm and rainfall events reporting to the pit. Water from the in-pit sumps is pumped to surface using diesel powered water pumps. Figure 15 shows pump quantities included in the study (refer to PFD GUI534-M-G200-F-105 for details on the pit dewatering reticulation). The pumps are selected to pump water containing silt particles of less than 1 mm in size. The water is pumped to a transfer tank and booster pump station on surface which would remain in position for the life of the pit. The booster pump station is equipped with one duty and one standby electrically-powered pump. Power to the booster pump station is supplied using an overhead power line with a pole transformer and local control system. The transfer tank is sized to prevent unnecessary stopping and starting of the pumps when the in-pit pumps are feeding intermittently.

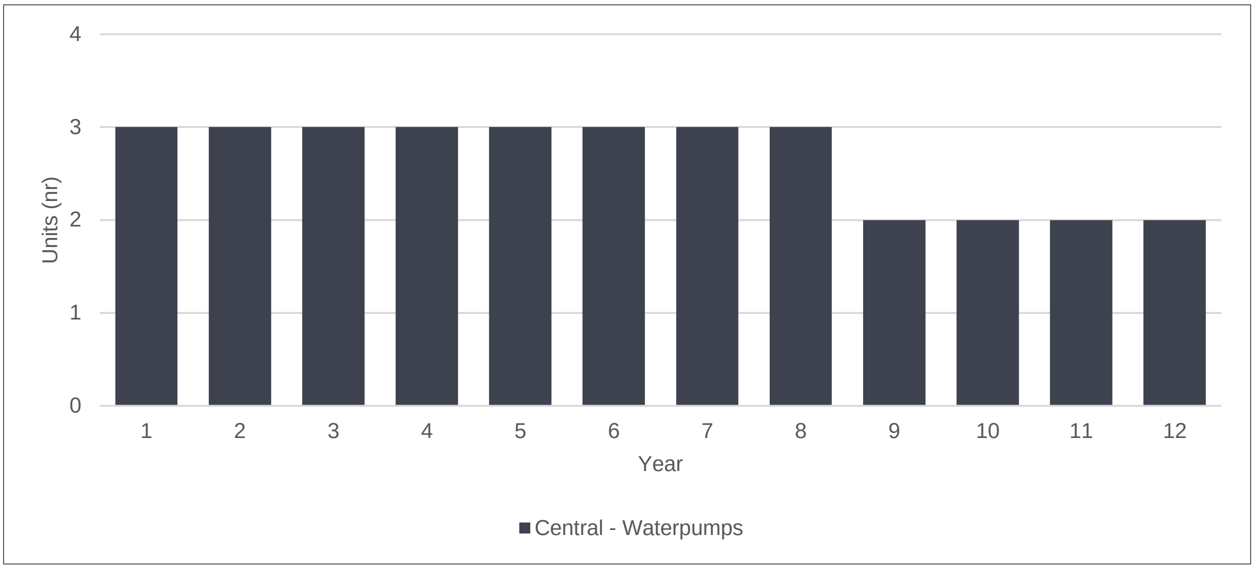

Figure 18-38: In-pit diesel pumps

The booster pump station pumps the water to the PCD at the central facility where the water is combined with water from other sources.

A dust suppression truck filling point is installed in the pit and at the PCD for efficient use of water for dust management. Refer to Section 18.2.7 for further information.

18.2.7.Dust suppression

This section only refers to dust suppression on haul roads and mining working area. Dust suppression on haul roads and in mining working areas will be done through the application of water onto the surfaces. Water will be applied by a water bowser, equipped with sprayers, on the surface of the road while traveling.

Allowance has been made for three water filling points, one in each pit and one at the centrally located PCD. At in-pit filling points, water will be abstracted from the in-pit sumps, where water accumulates after rain events or from ground water inflows. Water for the centrally located filling point at the PCD will be abstracted from the PCD. The PCD is equipped with an abstraction point, where water will be abstracted.

Figure 18-39: Typical dust suppression filling standpipe

Filling points will consist of a pump arrangement with a standpipe, as shown in Figure 18-39 above. The standpipe is designed to fill the water bowsers overhead, without the need to connect any piping. Filling points are designed to be mobile so that it can be relocated as necessary.

As far as reasonably possible, water shall be recycled for dust suppression. The use of raw or potable water for dust suppression should be avoided as far as practically possible.

18.2.8.Internal access roads

Access roads within the mining infrastructure complex area are clearly marked roadways, with the necessary signage. Demarcation will be done using small, constructed earth berms. Internal access roads will not consist of any layer works additional to that of the terrace. All access roads were designed to be 7 m wide for bi-directional vehicle movement and 4 m for one-way movement.

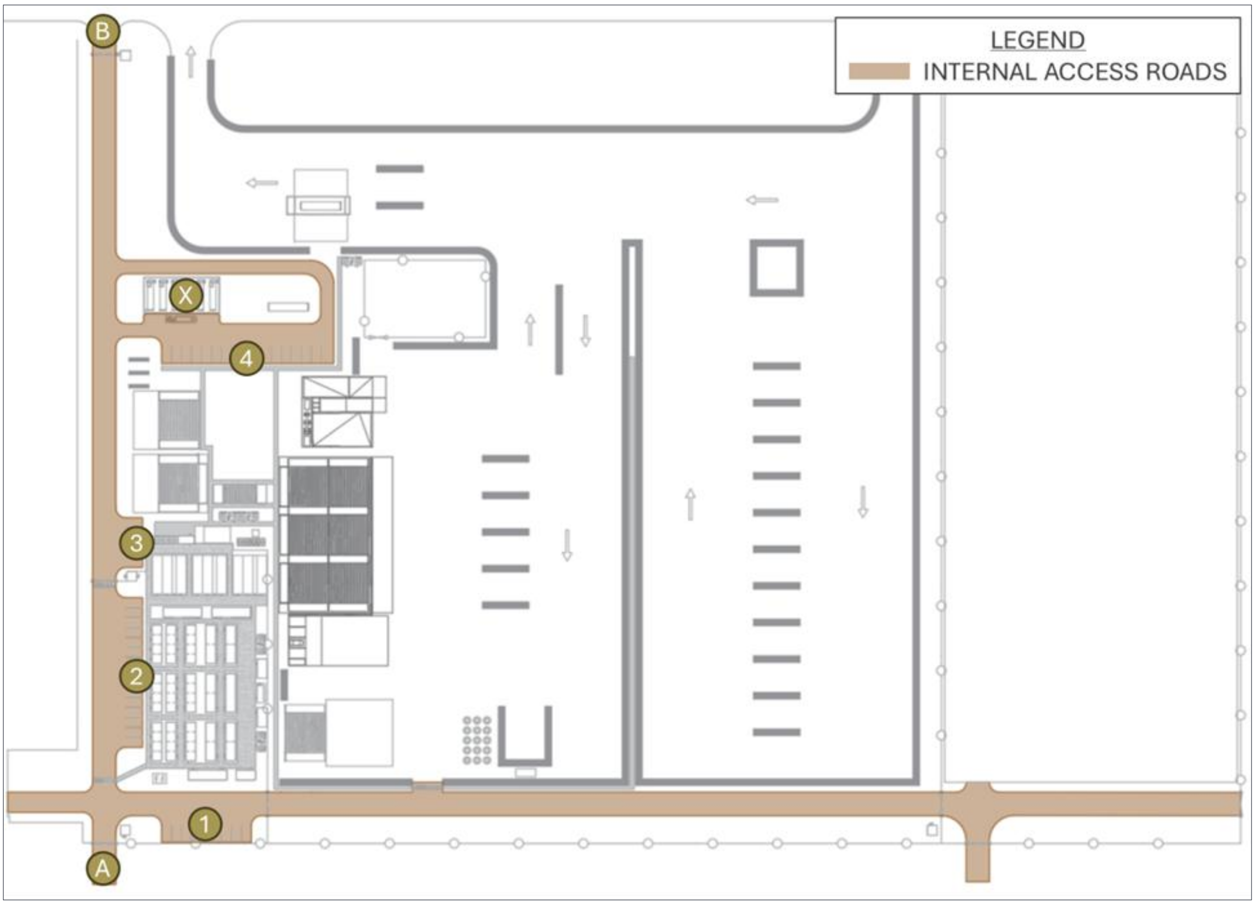

Figure 18-40: Mining infrastructure complex area - internal roads

The main access to the mining area is via the main gate situated at the processing plant. An additional access gate has been allowed for at point A, as shown in Figure 18-40, which provides direct access into the mining area at point B. This will allow for abnormal loads to be transported into the site without needing to use the main access gate.

The access road around the bulk fuel storage facility, located at point X, is a one-way road, controlling the movement of traffic in this high traffic zone. This area is considered a high traffic zone, especially during shift changes.

18.2.9.Parking

18.2.9.1. Light delivery vehicle parking

Allowance has been made for four formal parking areas at the mining infrastructure complex, two of which are located in the mining area and the others in the general area. Based on the transportation requirements it was calculated that 24 parking bays will be required for on-site mining LDVs. Based on the employee profile and numbers, it has been calculated that 19 parking bays are required in the general area. Table 18-14 provides the description of each of the areas:

Table 18-14: Parking area descriptions

Area number	Description	Number of bays
1	Visitor and overflow parking	9
2	Primary employee parking	15
3	Mining operator transport parking	5
4	Mining LDV parking area	17

All parking bays were designed to suit LDVs, with a width of 3 m and depth of 6 m. Parking bays will be demarcated, similar to the roads, using small, compacted earth berms on the rear and sides. A gap shall be left in the rear berm, and between the diving berm and the rear berm at each bay to allow for the movement of pedestrians.

All offsite staff transport vehicles will park at the main entrance gate when not in use.

18.2.9.2. Heavy mining equipment parking

The mining operation will utilise a "hot seat" shift change system, where operators exchange control of equipment at its current location. This approach eliminates the need for a large hard stand area. Consequently, the current design includes 12 parking bays within the hard stand. Should additional bays be required, the layout can be optimised, or the area expanded to the southeast.

18.2.10. Access control

The main gate, integrated with the processing plant infrastructure, serves as the primary access control point for the mine. All personnel will enter the mine through this point and proceed to their designated work areas. Mining personnel will report to the general mining infrastructure area, while office and processing plant personnel will report to their respective duties. Operational personnel will access the mining area after passing through the change house. The general area is separated from the mining area by a fence, with a secondary access control point regulating entry into the mining area. Due to the inherent risks associated with mining activities, appropriate training and authorisation are mandatory for access.

Boom gates are installed on either side of the haul road at its intersection with the access road. These gates control LDV traffic crossing or entering the haul road.

18.2.11. Offices and general buildings

All office buildings will comprise modular, insulated, double-walled, prefabricated units. Each unit will be elevated on small tripod footings and accessed via steps equipped with a handrail. No paving or concrete surface bed will be installed beneath the units. Units will be pre-fitted with the necessary electrical and network infrastructure.

Office facilities have been designed to accommodate 80 personnel. This number includes all staff operating from the infrastructure complex, excluding mining equipment operators. The majority of these employees work dayshift. In addition to the offices building allowed for here, allowance has been made to accommodate the maintenance personnel in the workshops.

Table 18-15: Office and general buildings

Unit type	Reference number	Number of units required
Single office	1	36
Double office (shared)	2	9
Open plan office	3	3
Meeting room (pax 8)	4	1
Large meeting room (pax 12)	5	1
Storeroom	6	2
Printing room	7	1
Basic kitchen	8	1
Mess room	9	1

Figure 18-41 provides the layout of the mining operational office area.

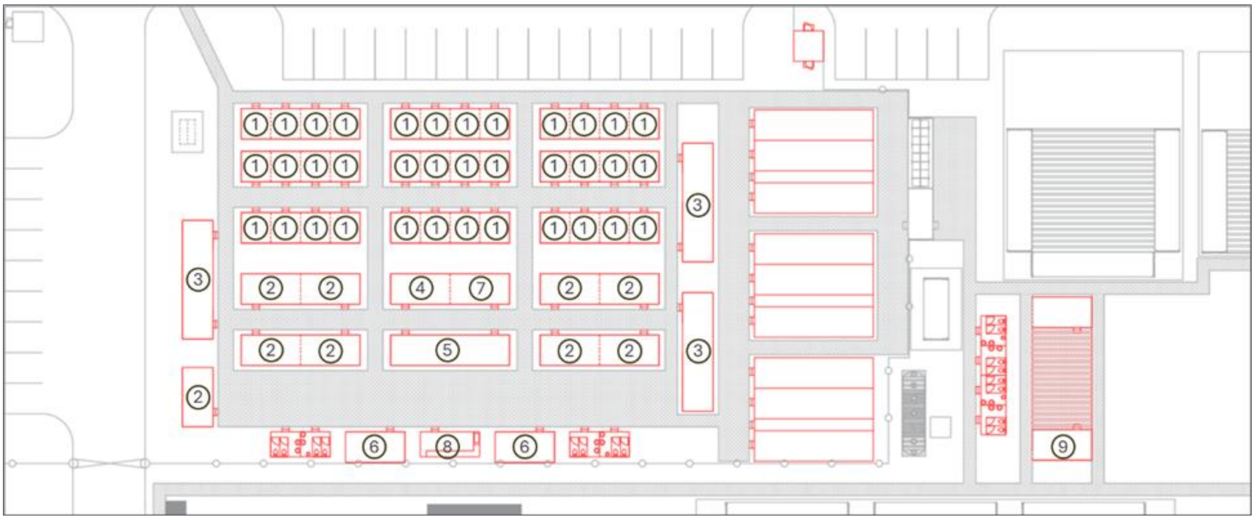

Figure 18-41: Mining operational offices and buildings

Allowance has been made to equip units with the following furniture and appliances:

Table 18-16: Office buildings and related furniture

Building type	Included furniture and appliance
Office	Desk, chair and bin
Open plan office	Desks, chairs, bins, filing cabinets and water dispenser
Boardroom/ meeting room	Boardroom table, chairs and water dispenser
Storeroom	Filing cabinets
Printing room	Table and multifunction printer
Kitchen	Fridge, microwave, hydro-boil, water dispenser, basic cutlery and crockery
Mess room	Water dispenser, benches and information display unit

While initially included in the design, paved walkways between buildings (using interlocking paving blocks) have been removed as a cost-saving measure. The site plans still depict these walkways.

The current design does not incorporate wheelchair access to the office buildings. It is recommended that accessibility be addressed during project implementation.

18.2.12. Ablutions

Similar to the office buildings, ablution facilities will be housed in prefabricated units. These units are strategically located in both the mining area and the general area, as shown in Figure 18-42. To ensure convenient access for personnel reporting to the service station, a dedicated ablution unit has been allocated to this location.

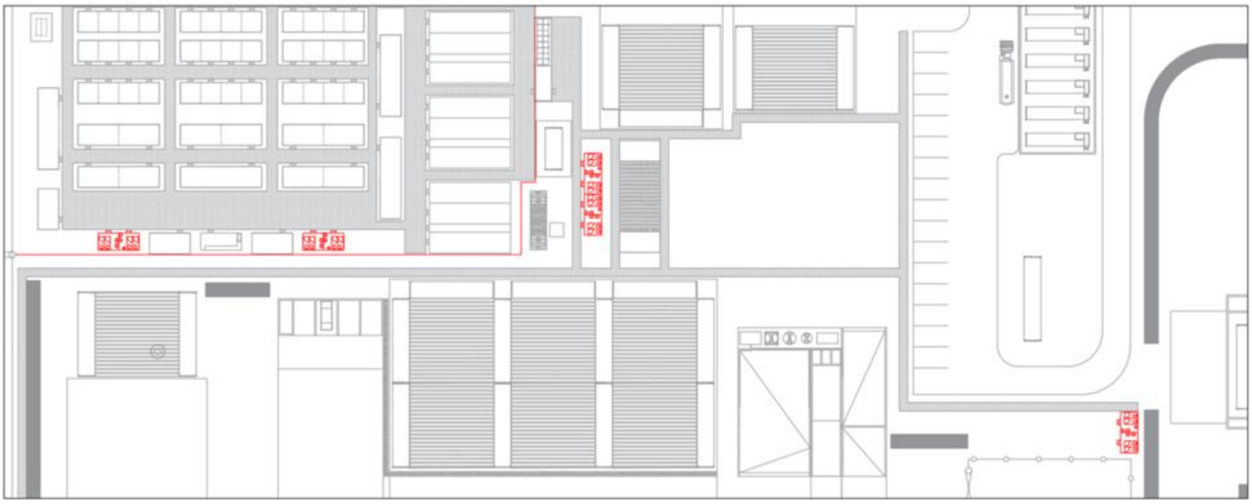
Figure 18-42: Fixed ablution facilities

The number of ablution units and sanitary fixtures has been determined based on the requirements stipulated in SANS 10400, with a projected initial workforce gender split of 20 % female and 80 % male, but will consider adjustment based on the success of the Company's corporate strategy for more balanced gender recruitment split. Table 18-17 provides a breakdown of the sanitary fixtures allocated per area to meet anticipated demand.

Table 18-17: Sanitary fixtures per area

Male			Female	
Water closet	Urinal	Hand wash basin	Water closet	Hand wash basin
General/office area				
3	6	5	3	2
Mining area				
3	6	5	3	2

Change houses are equipped with the necessary sanitary fixtures to accommodate operator needs during shift changes.

To support personnel working in remote locations, the provision of temporary sanitary facilities is recommended. Portable ablution units offer a practical solution, allowing for strategic placement in areas where continuous operations are required. This approach ensures worker well-being and minimises lost time associated with travel to distant ablution facilities.

The final layout of the ablution units will depend on the selected supplier and requirements set during this time.

18.2.13. Change houses

Change houses, like the office buildings and ablution blocks, will utilise modular, prefabricated units. These units are strategically located within the general area to ensure personnel pass through them before entering and after exiting the working area at the beginning and end of their shifts.

The design of the change houses incorporates the same gender split assumptions used for ablution facility calculations. The number of sanitary fixtures is based on an estimated 30-minute peak usage period, with an average shower time of five minutes. Each change house is equipped with water closets, hand wash basins, urinals (in the male section), showers, and lockers.

To facilitate maintenance activities without disrupting operations, the change house design includes a service passage located between the male and female sections. This passage aids ventilation by accommodating additional extractor fans within the dividing wall.

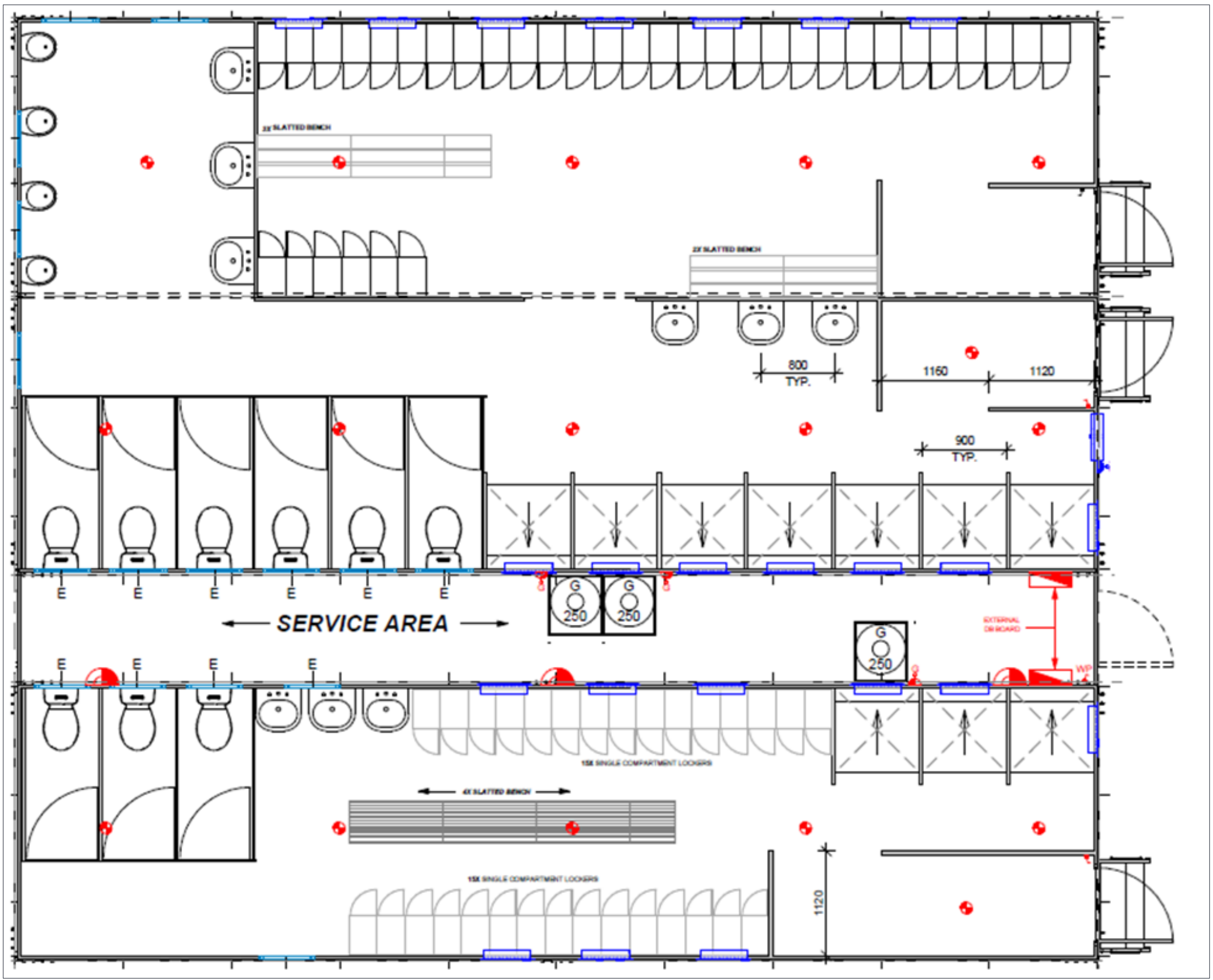

Figure 18-43: Proposed change house design

All fixtures within the change houses will be water-resistant and corrosion-protected. The floors will feature a fully rubberised surface to enable pressure cleaning and maintain hygiene.

High-pressure solar geysers will be installed in each change house unit to provide hot water. These geysers offer reduced energy consumption compared to traditional electric water heaters.

18.2.14. Engineering workshop

The engineering workshop is a critical single-bay facility designed to support a range of essential maintenance and repair activities. It comprises several key components working in concert:

- A reinforced concrete surface bed
- Three converted 40-foot shipping containers, and
- A container-mounted engineered fabric dome shelter.

Refer to Figure 18-44 for the layout of the workshop. This structure provides a versatile and protected workspace for engineering and maintenance operations, encompassing electrical, mechanical, boilermaking, and piping work.

The two side containers are designated as secure storerooms, providing organised storage for essential spare parts, tools, and maintenance supplies. The interior of these containers is fitted with heavy-duty shelving units, maximising storage capacity and ensuring easy access to frequently used items. This organised storage system contributes to efficient workflow and minimises downtime when repairs are needed. The rear container is converted into an office for the foreman. This office is large enough to accommodate small meetings, allowing for on-site coordination and planning.

The central bay, measuring approximately 12 m x 12 m, constitutes the primary work area. This open space provides ample room for technicians or artisans to manoeuvre and work on equipment. The reinforced concrete surface bed provides a durable and level working surface, capable of withstanding the weight and stresses associated with heavy equipment and repair activities.

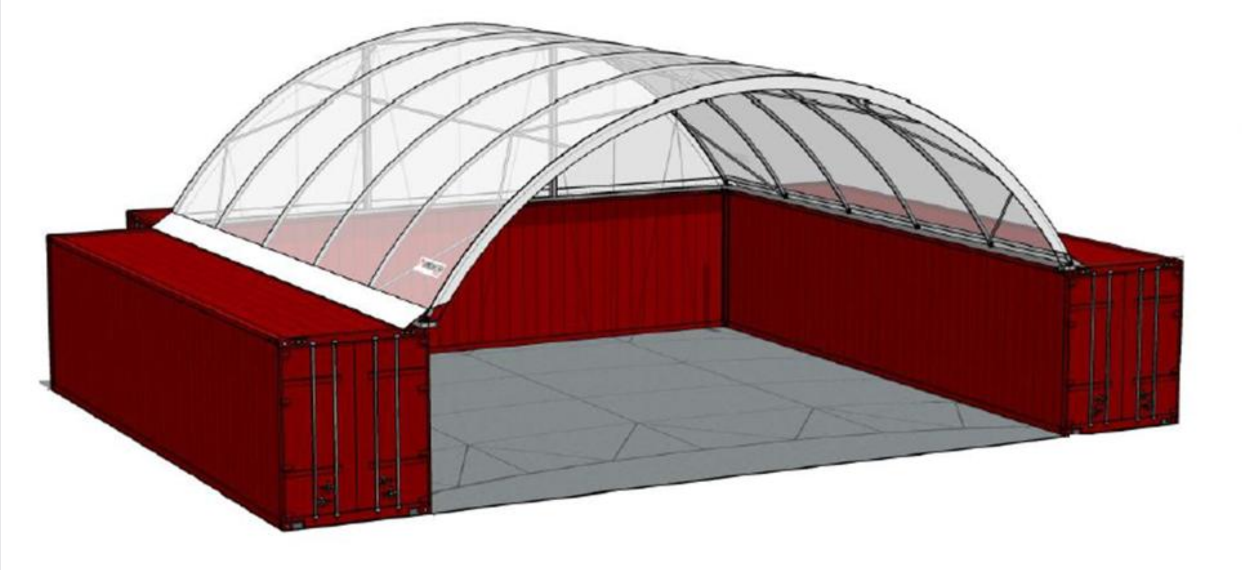

Figure 18-44: Engineering workshop

Access to the workshop is via the front, which will remain open to ensure adequate ventilation and prevent the buildup of fumes or dust. This open access facilitates the movement of equipment and materials into and out of the workshop.

The workshop is equipped with a range of essential tools and equipment, including a compressor for powering pneumatic tools, a welder for fabrication and repair work, and other general engineering tools and equipment necessary for a variety of maintenance tasks. Components required for maintenance will be stored at the central storage area and at the workshop.

Refuse bins are located outside the workshop for the disposal of general and recyclable waste. Other waste materials, including used oil and other hazardous waste, shall be collected and transported to designated waste skips located in the broader site waste management area, ensuring responsible disposal practices.

It is important to note that the current design of the workshop does not include a crane or fixed rigging apparatus. Lifting and material handling will need to be addressed through alternative methods, such as mobile cranes or forklifts, as required. This should be considered in future planning and resource allocation for the workshop.

18.2.15. Light delivery vehicle workshop

The LDV workshop is a dedicated facility designed for the servicing and repair of LDVs. Similar in construction to the engineering workshop, the LDV workshop comprises a reinforced concrete surface bed, converted 40-foot shipping containers, and a container-mounted engineered fabric dome shelter. A central bay, approximately 12 m x12 m, provides the primary work area, designed to accommodate either four smaller or two larger LDVs. The concrete surface bed extends outward to the front of the workshop, creating an additional area for vehicle checks or expanding the primary work zone as needed. The workshop's open design at both ends facilitates the easy passage of vehicles or allows for face-to-face vehicle arrangement within the bay.

One of the shipping containers has been fully converted into a parts store, ensuring readily available components for maintenance and repairs. The other container is divided into two distinct areas:

1. An office for the responsible foreman and
2. A dedicated tool store.

This configuration provides both administrative space and secure tool storage within the workshop itself. The LDV workshop is comprehensively equipped with the necessary tools, compressors, jacks, and chain hoists to enable safe and efficient maintenance activities.

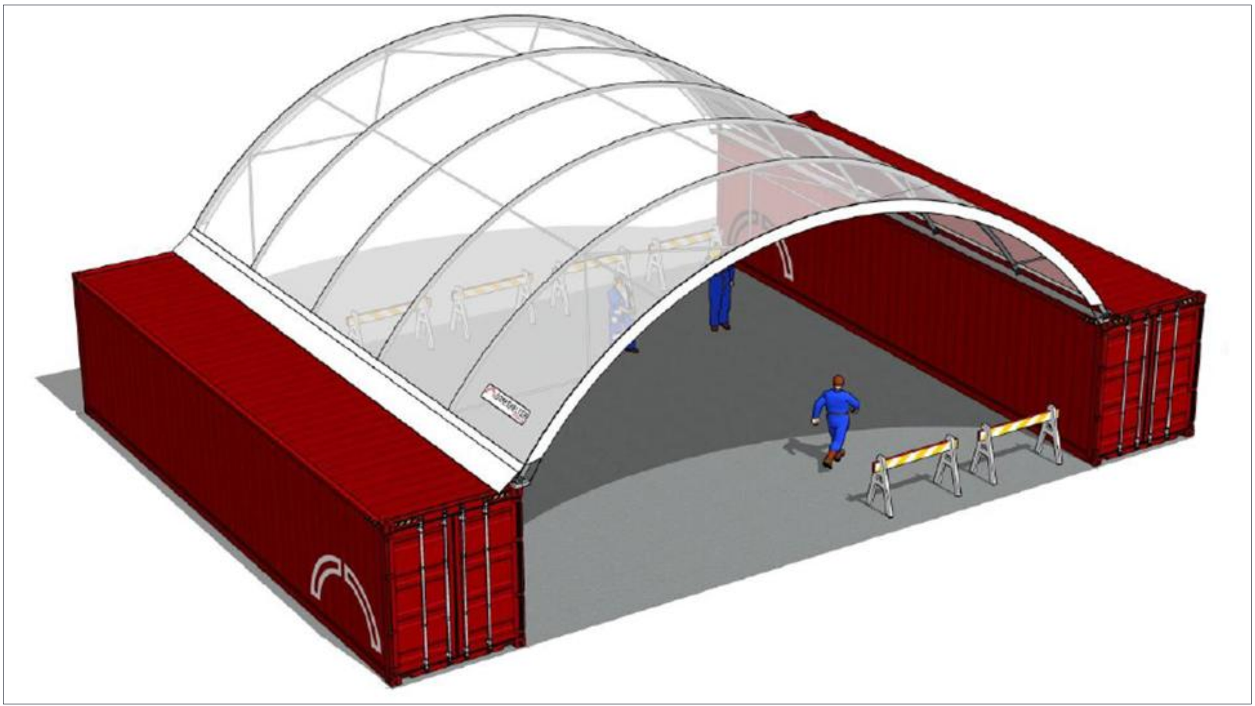

Figure 18-45: LDV workshop

The workshop's close proximity to the LDV wash bay allows for convenient washing and cleaning of vehicles prior to servicing or maintenance. Several parking bays are located outside the workshop, providing designated spaces for vehicles awaiting attention or parts. This ensures an organised and efficient flow of vehicles through the workshop.

18.2.16. Mining maintenance and repair workshop

The mining maintenance and repair workshop is a three-bay facility (see Figure 18-46 for the layout). Each bay has a primary function: maintenance, maintenance and repair, and secondary equipment. The interconnected bays facilitate the movement of personnel, parts, and tools. Each bay measures 24 m x 12 m. The workshop incorporates a reinforced concrete surface bed, converted shipping containers, and container-mounted engineered fabric dome shelters.

Workshops have been designed to accommodate the following types and number of equipment at peak:

Table 18-18: Equipment types and numbers used a basis for workshop designs

Equipment [or similar]	Type	Number of
CAT 745	45t ADT	49
CAT 395	Excavator	8
CAT 140	Grader	3
CAT 730	Water bowser	3
CAT 730	Diesel bowser	2
CAT D8	Dozer	8
Komatsu WA600	FEL	1
CAT 330GC	Rock breaker	2

The depth of the bays accommodates the range of mining equipment and provides a dedicated working zone at the rear of each bay for component work. This working zone, in conjunction with the approach apron, allows for two 40t ADTs to be serviced within a single bay, providing operational flexibility.

The maintenance bay is dedicated to equipment maintenance, while the maintenance and repair bay serves a dual purpose. The secondary equipment bay features an extra-high dome roof to accommodate shovels and allow 40t ADTs to raise their dump bodies for hydraulic system work. This bay has embedded rails in the surface bed to allow tracked equipment to enter without causing significant damage.

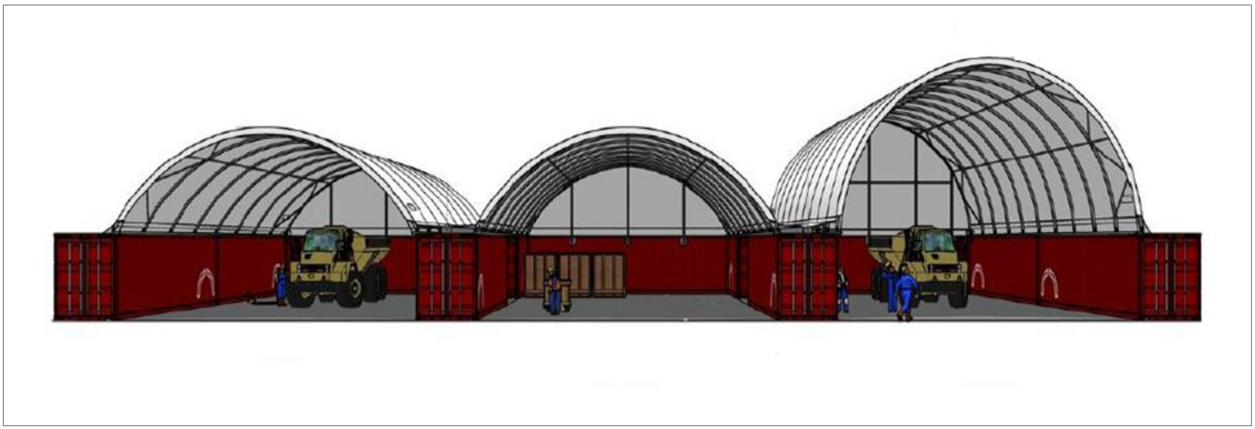

Figure 18-46: Mining maintenance and repair workshop front view

All side containers are equipped with shelving for storing parts, equipment, and tools. The rear containers have been converted into offices for the foreman, technicians, and artisans.

A number of HME parking bays are located in front of the workshop for equipment awaiting attention (see Figure 18-47). The adjacent HME wash bay allows equipment to be cleaned before maintenance or repair work begins.

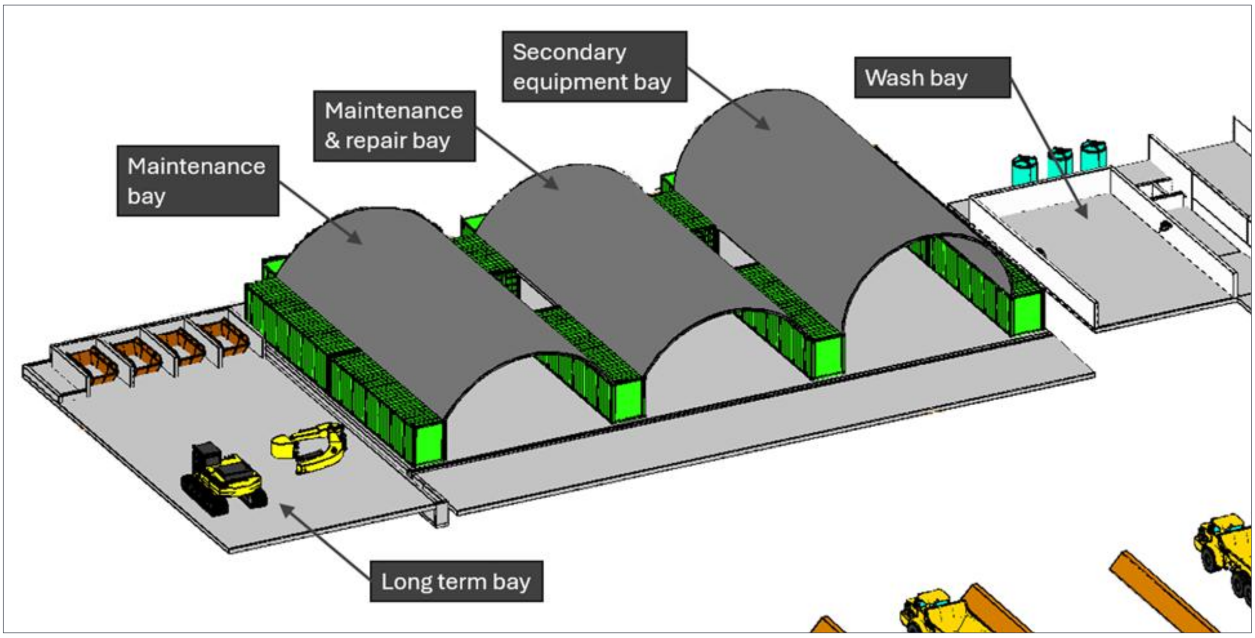

Figure 18-47: Mining maintenance area

A cut-off drain at the entrance to the three bays captures all dirty runoff and wash water, channelling it to a collection sump between the workshop and the long-term repair bay. From this sump, the water is pumped to the wash bay silt trap.

The mining maintenance workshop is not equipped with fixed or overhead cranes. All rigging will be performed using a flatbed service truck with a Hiab attachment. The width of the workshop bays allows the service truck to manoeuvre around the equipment being serviced. The workshop allows for ample space next to a truck for the tyre handler to remove wheels when suspension or drive train-related maintenance is required.

18.2.17. Long-term repair bay and waste management area

A long-term repair bay has been designated within the mining infrastructure area. See Figure 18-48 for the layout of the area. This facility comprises an uncovered, reinforced concrete surface bed designed to accommodate extended maintenance activities, thereby optimising the utilisation of workshop bays. Due to space limitations, not all rebuilds and equipment repairs can be performed in the workshops. The long-term repair bay will handle these overflow activities.

The reinforced concrete surface of the bay facilitates effective cleaning in the event of hydrocarbon spills, ensures adequate drainage, and supports the use of jacks during repair operations. Embedded steel rails allow tracked equipment to be moved onto the surface without causing significant damage. This is particularly important for large equipment such as shovels, which experience difficulty accessing the workshop bays.

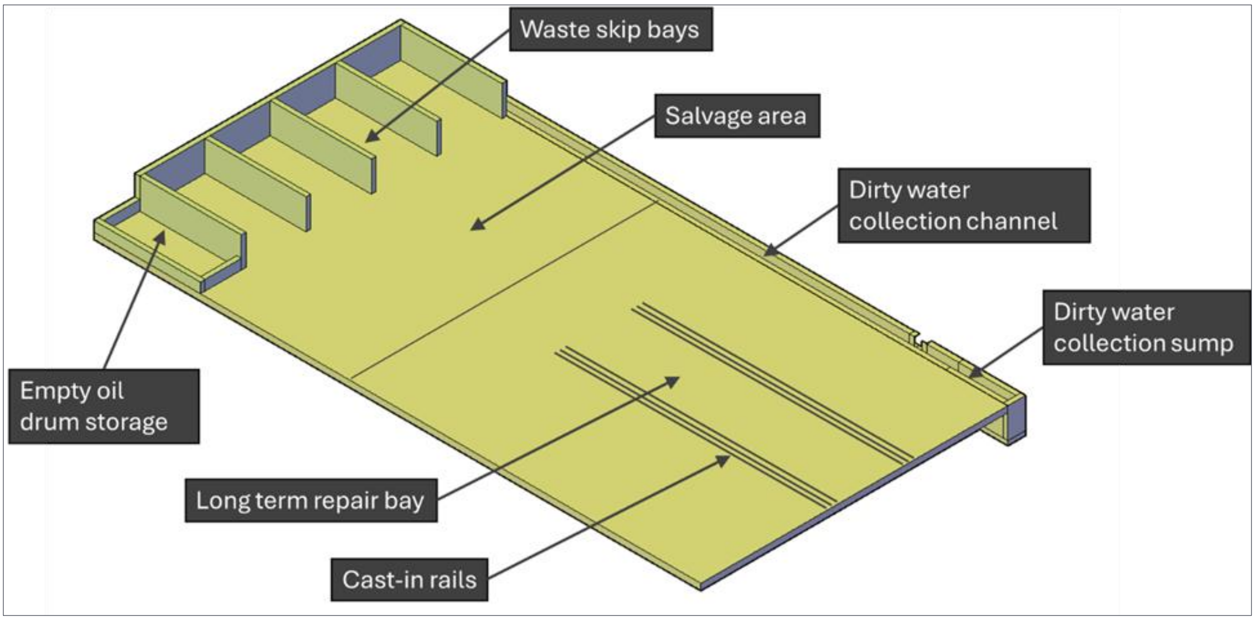

Figure 18-48: Long term repair bay and waste management area

The long-term repair bay is co-located with the salvage and waste management area. Given the high probability of hydrocarbon spills in this combined area, it is considered a "dirty" zone. The entire area drains towards a single dirty water collection sump, from which water is pumped to the silt trap at the wash bay. The design includes designated storage for four 9m³ waste skips, as well as a bunded area for the storage of used and empty oil drums. The area is accessible from the mining area to facilitate collection and off-site disposal by the appointed waste removal service provider.

18.2.18. Wash bay

The wash bay, as depicted in Figure 18-49, is designed as a dual facility to accommodate both LDVs and HMEs. Located adjacent to the equipment maintenance facilities, between the mining and general areas, it allows convenient access for both types of vehicles. LDVs can enter from the general area, while HMEs access the bay from the mining maintenance area.

Combining the wash bays into a single facility allows for centralised infrastructure, including a single silt trap, drying bed, pressure washer, and oil separator. All mechanical equipment is readily accessible for maintenance from the general area.

Both the HME and LDV washing bays are sized to accommodate the proposed fleet. The reinforced concrete structure and surface beds are designed to withstand the load of these vehicles.

While the design does not include cast-in rails for tracked equipment, conveyor belts can be used to facilitate the washing of such vehicles if needed.

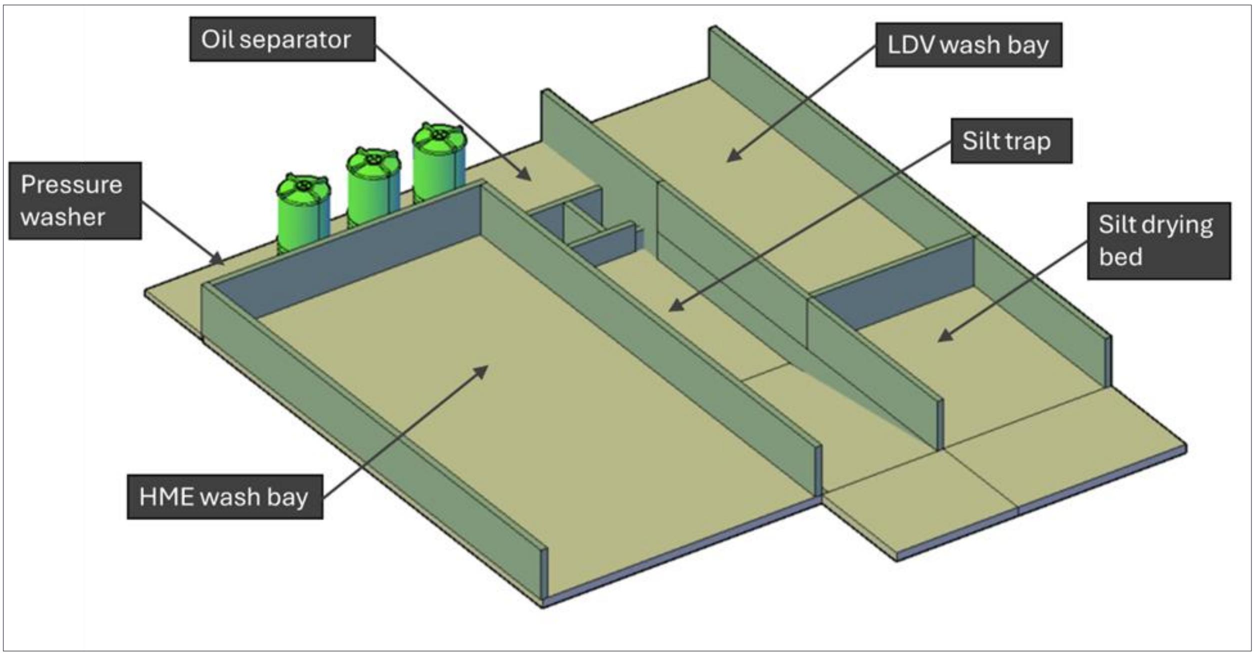

Figure 18-49: Combined wash bay arrangement

Dirty water from the washing bays will drain into a silt trap. Once the water has been desilted, it flows into a conventional over-under flow oil trap, which captures most of the oil.

A mechanical oil separator skims the oil from the surface of the water in the oil trap. The separated oil is stored in a tank for collection and responsible disposal. The clarified water from the separator is circulated back into the system for washing. Any excess clarified water from the oil trap is also returned to the pressure washer feed tank.

The wash bay is equipped with a high-pressure washing system that supplies handguns in both bays. This system reduces both the time and water required to wash equipment. As mentioned, the water is recirculated and topped up with raw water when necessary. The pressure washing system, consisting of two Cat Model 3535 pumps, delivers 130 litres per minute at 50 bar to the handguns. Figure 18-50 shows the proposed pressure washer, hose reels and handguns.

Figure 18-50: Bestline pressure washer and handguns

Excess water in the system which cannot be reused will be pumped to the PCD from where it can be abstracted and used for dust suppression.

18.2.19. Tyre management

The tyre workshop at the central facility is equipped to provide a service to the whole operation. All tyre replacements and repair work will be completed at the central facility. Complete wheel units will be transported to the remote location for fitment when required.

The tyre facilities were sized for the equipment operating on site. The typical tyre sizes considered for the project are shown in Table 18-19.

Table 18-19: Tyre sizes for typical equipment

Applies to machines	Tyre Size	Bridgestone description	Aeolus description
140K Grader	17.5-25	VSDL *1 L5 D2A	AL419 AE L4
A30G B25E ADT	23.5R25	VLTS MS E4	AE417 TL AE E-4**
Caterpillar 745 ADT	29.5R25	VLTSZ *2 E4 E2A	AL53 TL AE L-5**
Cat 980L Loader	29.5R25	VSDL *2 L5 D2A	L3 AL37 TL AE E3**
Cat 988G Loader	35/65 R33	VSDL *2 L5 D2A	AL53 TL L-5***
Cat 992K Loader	45/65R45	VSDL *2 L5 D2A	

The facility is planned to be managed and operated by a specialised tyre service provider. The service provider will be responsible for the following:

- Manage tyre life of all equipment (monitor and track) (primary, secondary, services, support fleet, light duty vehicles and busses)
- Supply of tyres and consumables
- Stock keeping and planning
- Replacement planning
- Tyre repairs on site
- Tyre fitment on site
- Daily tyre inspections and condition monitoring
- Provide tools and equipment (including tyre handler, press etc. for all activities associated with tyre management on site
- Provide all labour, management personnel and services required to effectively provide the services outlined in this request
- Contract duration of five years with option to extend based on contractor performance.

The planned facilities at the central site consist of a workshop, changing bay, inflation bay and storage.

18.2.19.1. Tyre workshop building

Similar in construction to the engineering workshop, the tyre workshop comprises a reinforced concrete surface bed, converted 40-foot shipping containers, and a container-mounted engineered fabric dome shelter an indicated in Figure 18-51. A single bay, approximately 12 m x 12 m, provides the primary work area for the fitment of tyres and wheel repairs.

One of the shipping containers has been fully converted into a parts store, ensuring readily available components for maintenance and repairs. The other container is divided into two distinct areas, an office for the responsible foreman and a dedicated tool store. This configuration provides both administrative space and secure tool storage within the workshop itself. The tyre workshop is comprehensively equipped with the necessary tools, compressors, jacks, ladders and supports, supplied by the service provider, to enable safe and efficient tyre replacements and repairs. All tyre repair and maintenance work will be completed at the central tyre workshop.

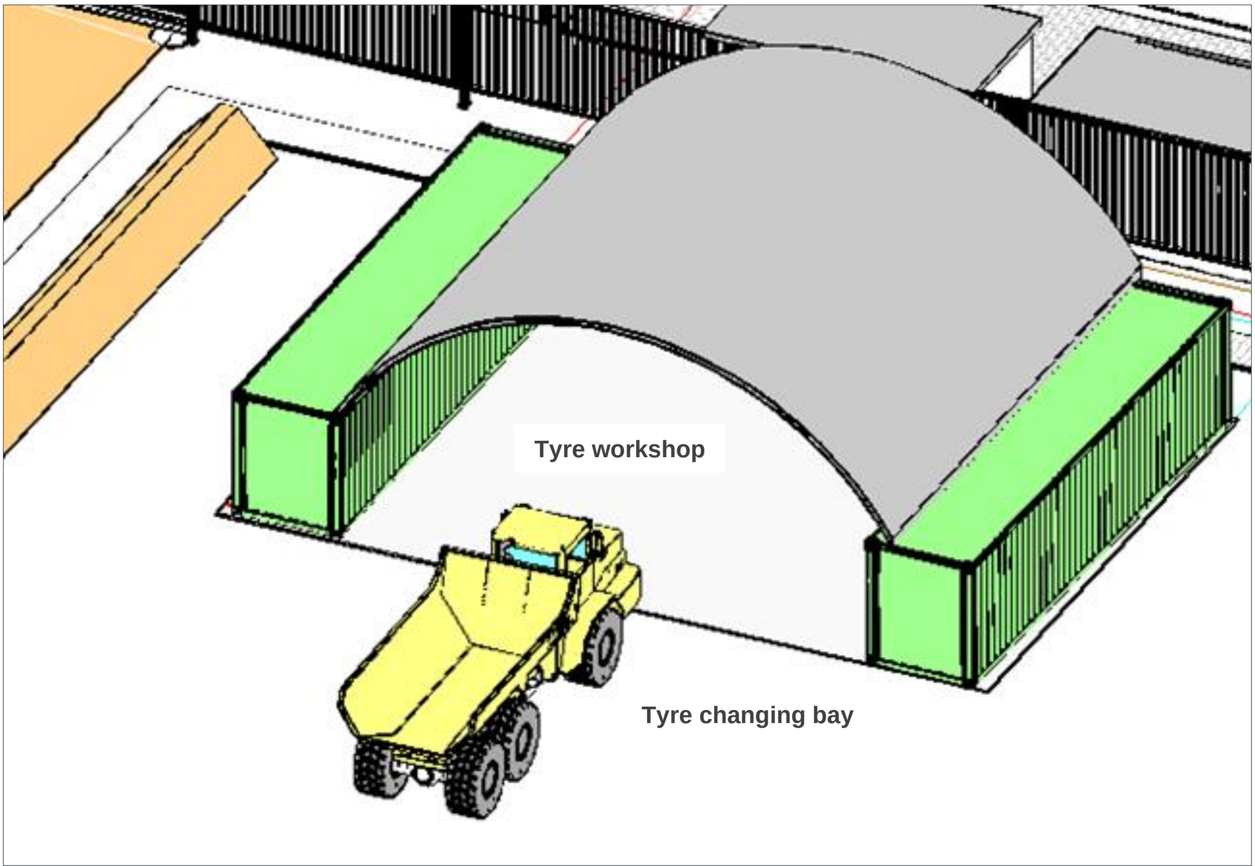

Figure 18-51: Tyre workshop and changing bay

18.2.19.2. Tyre changing bay

The tyre changing bay is a prepared dedicated flat area on the terrace with no civil construction. Tyre changes will be done using a dedicated tyre handler machine to ensure safe working conditions. Sufficient space was allowed at the tyre changing bay for the movement of the tyre handler around the equipment in their parked position.

18.2.19.3. Tyre inflation bay

The tyre inflation bay is a dedicated area on the terrace with no civil construction as depicted in Figure 18-52. The tyre inflation bay has berms around three sides to ensure the safety of personnel during inflation. No persons are allowed in the area while tyres are inflated.

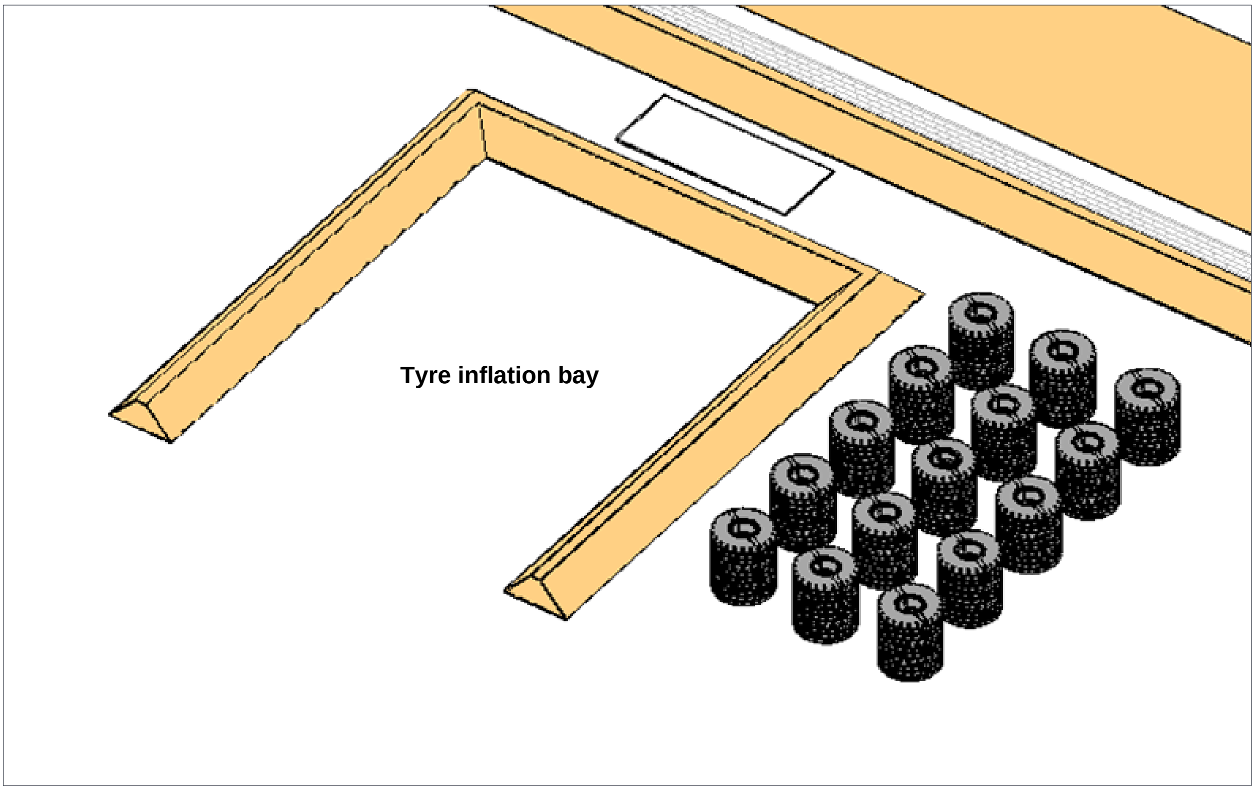

Figure 18-52: Tyre inflation bay and storage

18.2.19.4. Storage

The storage area is sized for storage of tyres for all the sites. The size of tyres can be stacked up to three levels without damaging the tyres for long term storage. The stock keeping quantities will be managed by the service provider. The storage area does not have any civil construction and consist of only a prepared surface. Provision was made for fire hydrants in close proximity of the tyre storage for emergencies.

18.2.19.5. Support equipment

Tyre fitment at the central facility will be done by a dedicated tyre handler as depicted in Figure 18-53. The tyre handler attachment will be supplied, maintained and operated by the service provider. Allowance was made for the conversion of wheel loader (Cat 960 size) as a carrier.

Figure 18-53: Tyre handler for central facility

Wheel assemblies will be transported to the remote locations using a flatbed truck with a tyre handler attachment where field support is required.

Figure 18-54: Field service truck

18.2.19.6. Request for quotation and recommendations

A request for quotation was issued to two providers for the tyre services, both submitted a proposal.

Both tender submissions were reviewed and found to be technically and financially compliant offers for the specified scope of work. The tenderers have a significant focus on safety with good practices and commitments to ensuring a safe working environment. The offers included labour complement and equipment to support the mining two-shift system.

18.2.20. Support equipment

The mining operation is supported by additional services rendered to the operations. Support mobile equipment is allowed for in the project to ensure that support services can be rendered and consist of:

- Flat bed with a crane behind the cab for general maintenance activities such as equipment maintenance and transportation of spares and components
- A 100 t hydraulic neck lowbed for the relocation of tracked equipment for operational and maintenance purposes
- A 3 t forklift for loading and offloading of delivery trucks and housekeeping
- A field service truck for the maintenance of equipment in the field.

18.2.21. Bulk fuel storage

Figure 18-55 shows the area that has been allocated for the bulk fuel storage and fuel management service provider for the establishment of their facilities. The area is easily accessible for the delivery of fuel. Allowance has been made to supply services, including sewage, potable water, service water and power to the area. Upon finalisation of the design, the integration of dirty water shall be considered.

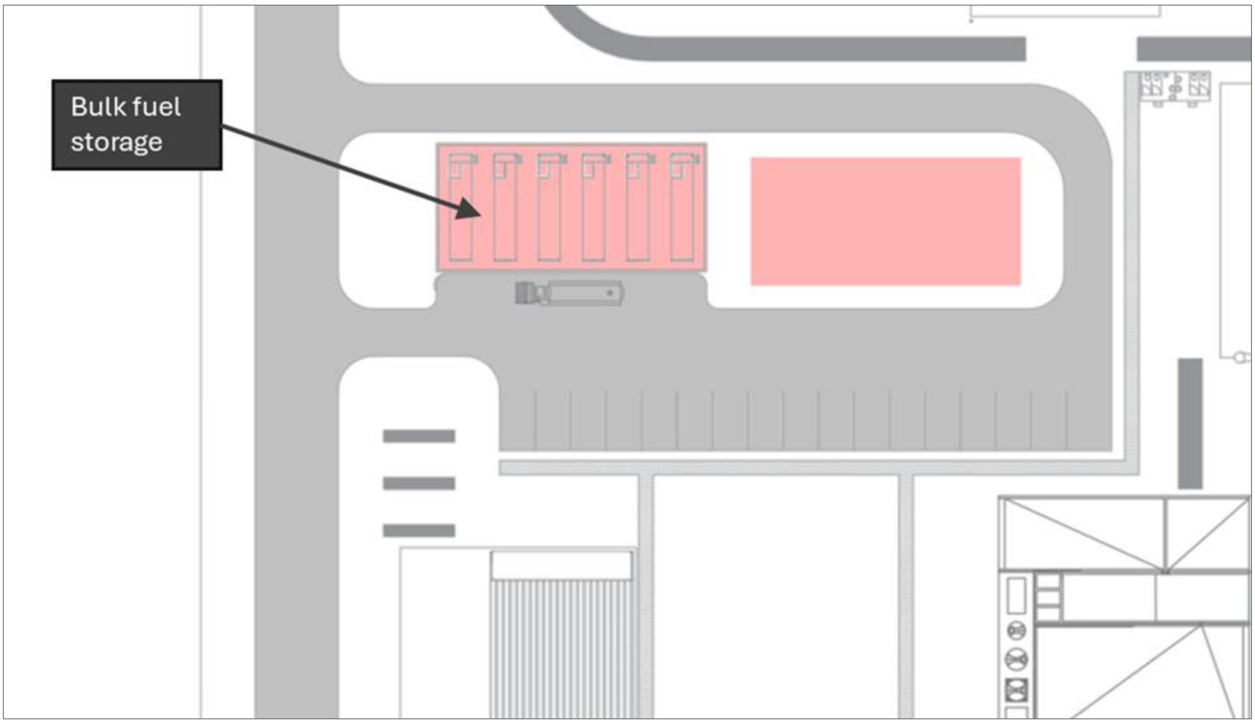

Figure 18-55: Bulk fuel storage area

18.2.22. Refuelling and service station

A two-bay service station has been allocated within the design. This station will facilitate daily service checks and refuelling, functioning as a drive-through facility easily accessible from both the hard stand area and the main haul road. The fuel supply contractor will be responsible for establishing and managing this facility; therefore, its final design and operation will be detailed during contract negotiations.

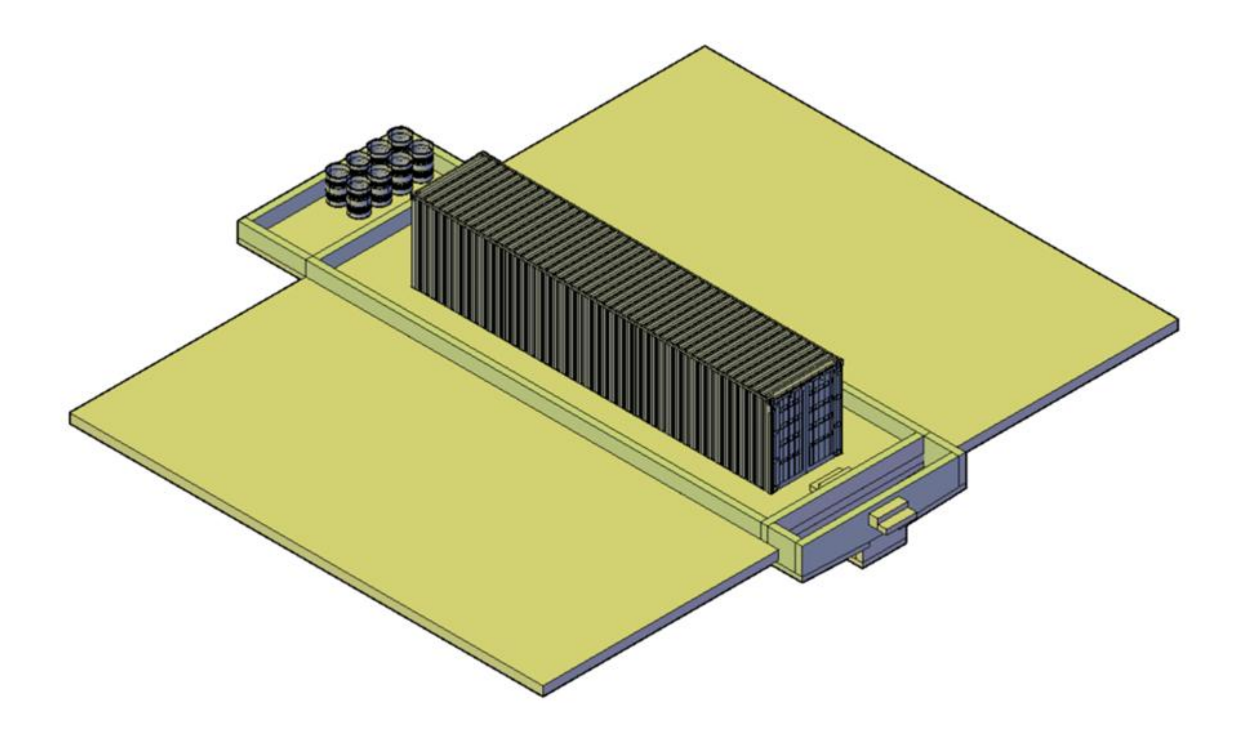

Figure 18-56: Refuelling and service station proposed layout

The proposed service station comprises a central bunded area for fuel and lubricant storage and two concrete service bays. Fuel and lubrication dispensing units are installed, maintained and managed by the fuel service provider.

Concrete surface beds are recommended for these bays due to the high potential for hydrocarbon spills. This surface type allows for easy cleaning and effective runoff management, in accordance with environmental management procedures. Runoff from this area will be contained and treated as contaminated. Provision has been made to pump this contaminated water to the silt trap at the wash bay for further processing.

While currently depicted as uncovered in Figure 18-56, it is strongly recommended that the final design incorporate a canopy to ensure uninterrupted operation during inclement weather. A canopy will minimise rainwater accumulation in the service area, thereby reducing the volume of contaminated water requiring treatment.

18.2.23. Explosives magazine

The project's explosives management strategy involves outsourcing this service. As the facility requirements were not fully defined during this study, a representative explosives magazine location has been reserved. This location was selected due to its remoteness, ensuring no permanent infrastructure lies within the 500 m blast radius. The location allows the pit's blasting radius to overlap with the magazine's. While situated within the mine's perimeter fence, the facility will be separately fenced with controlled access, in compliance with the Explosives Act.

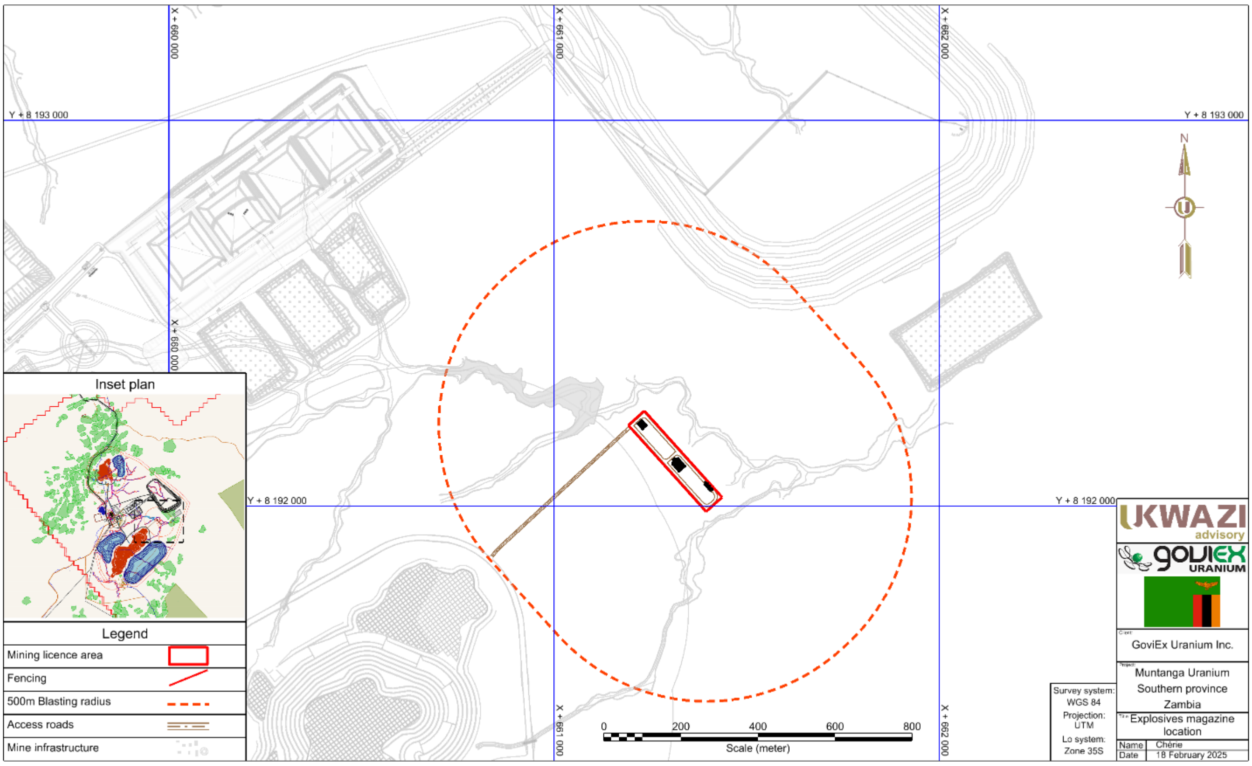

Figure 18-57: Explosives magazine

In the next project phase, the required explosive quantities should be finalised, and the corresponding blast radius requirements re-evaluated. As shown in Figure 18-57, the final design of the spent ore dump, completed late in the project, positions its toe within the 500 m radius of the explosives magazine. This proximity requires review and resolution prior to implementation.

The capital cost estimate includes provisions for the access road, site preparation, and fencing. Further development and equipping of the facility will be the responsibility of the appointed service provider.

18.2.24. Compressed air

A rotary screw air compressor size GA 37 @ 216 cfm with air receiver of 1.5 m^3 was selected for usage for the mining workshops. An allowance for pipe reticulation has been made to supply the compressed air in all workshops. Compressed air piping and distribution headers will be galvanised and provided with suitable filter/ lubricator sets, air pressure gauge, isolating valves and water drainage valves. Headers will be installed at approximately 5° to the horizontal for water collection and drainage.

18.2.25. Electrical

The Muntanga central mining area will use a 1 MVA mini substation to provide power to the main distribution boards. The detail of the design is contained in the following sections.

18.2.25.1. 11kV supply

A 95mm^2 three core XLPE SWA cable will be installed from the plant 11 kV substation to the mini substation. The plant substation is discussed in section 18.4 of this report. The estimated load requirement is 800 kVA. The fault level rating of the cable dictates the cable size instead of the cable's ampacity.

18.2.25.2. 11kV/400V Mini substation

The minisub will be an outdoor unit consisting of a 11 kV ring main unit ("RMU"), a 1000 kVA 11 kV/400 V transformer and a low voltage ("LV") feeder pillar. The feeder pillar will consist of a 2000 A moulded case circuit breaker ("MCCB") for the incomer and seven x miniature circuit breakers ("MCB") for distribution. The MCBs are all three phase units raging in size from 125 A to 300 A.

The RMU will be used to distribute 11 kV supply to the overhead line ("OHL") that will supply power to the PCD pumps.

The mini substation location is indicated on the layout drawings.

18.2.25.3. Distribution boards

Main distribution boards ("MDB") will be used to provide electrical power to different areas of the plant. These MDBs are 400V three phase units and are sized to allow for the loads that are assigned to them. During the design logical areas were identified where the electrical loads were grouped and connected to the MDBs. The MDBs were placed to optimise cable lengths considering the position of the larges loads and accessible locations.

For Muntanga Central, six MDBs will be used. Refer to the following drawings for more information regarding the location and loads of the MDBs.

- GUI534-M-E000-001 sheet 1 to 8 - SLDs
- GUI534-M-E000-002-1 - Muntanga layout drawing.

18.2.25.4. Cables and cable routing

All LV cables will be Steel Wired Armoured ("SWA") PVC Fire Retardant ("FR") and will be installed on vertical cable racking as far as practical. The cable racking is illustrated on the layout drawing and was designed with the shortest routes in mind while refraining from blocking access to operational and maintenance areas. The cable lengths are measured along the indicated routes with and allowance for the final runs.

18.2.25.5. Motor starters for pumps

A centralised form 2b motor control center ("MCC") will be used to control the pumps and will be installed in the workshop. If practical the MCC and relevant workshop MDB can be combined. The detailed design will investigate this option. No allowance was made for field isolation panels.

Field starters were allowed for in the case of the remote pump used at the PCD. This field starter will be an outdoor unit supplied form a pole mounted transformer. The field starter will be located close to the pump.

18.2.25.6. Lighting and small power

An allowance for lights was made by placing light fittings on the layout drawings. Different light fittings were used based on the application. In the open areas highmast lights were used. These installations are best suited for parking bays and areas used by the ADTs.

Highbay lights are used in the workshop areas, they are best suited for indoor use where good quality lighting is required. Bulkhead lights are installed outside along walkways and where light is required for operations and maintenance at night. Bulkheads are normally installed on a nearby structures or light poles where suitable structures are not available. In certain applications floodlights will be used. During detailed design the positions of the different lights will be confirm and a lighting study conducted.

The bill of quantity ("BOQ") made allowance for a standard type of installation for all lights, which will include Pratley boxes complete with terminals.

Switched socket outlets ("SSOs") and welding socket outlets ("WSO") are allowed for in areas such as the workshops and other areas that may require either a SSO or WSO or both. The SSOs and WSOs are indicated on the layout drawings.

No lights and SSOs were allowed for in the offices or change houses. Any lights and small power requirements for the offices and change houses will form part of their scope. The main supply to the offices and change houses are allowed for in the relevant MDB. It is recommended that solar lighting options be considered in the next stage of work.

18.2.25.7. Earthing

Materials for a standard earthing installation were allowed for in the BOQ. No earthing studies or earth resistivity test were complete as part of this phase of the project.

18.2.25.8. Pollution dam control pumps

Allowance was made for two field motor starter panels to control the PCD pumps. The motor starters will be powered by appropriately sized pole mounted transformers located close to the pump stations. An OHL will supply the transformers. The motor starters will be locally controlled by level switches and will make use of an intelligent motor starter relay to protect the motor.

18.2.25.9. Overhead lines

A wooden pole 11 kV OHL will be constructed between the mini-substation and the two PCD pump stations. The line will consist of mink conductor and will have dropout fuses to provide the required protection against fault conditions. The OHL is estimated to be 3 000 m long and is costed with all materials and labour included.

18.2.25.10. Solar installation

Allowance was made in the BOQ for a small solar installation complete with batteries that could operate independently. This installation will be located at and provide electrical power to the explosive magazine.

18.2.26. Water storage and reticulation

18.2.26.1. Potable water

A water reticulation network is required on site for potable water, process water and wastewater. The water reticulation system must be an easy-to-operate, reliable, economic and convenient system with a high level of reliability to deliver water in quantity and at the points where it is consumed. The system must be installed at minimum expense without sacrificing its design life and ongoing Capex or excessive maintenance costs.

Potable water for Muntanga will be coming from the plant. The potable water will be supplied to the offices and kitchen, messing facility, ablution facility, change house and diesel storage area. The potable water quality must adhere to the Zambia National Standard for drinking water.

The water taps flow rate must be 10 L/min (red book guideline). The amount of water required for Muntanga site was based 122 office-based personnel and 218 operational personnel per day, with an average demand of 50 L/p/d for office-based personnel and 92 L/p/d for operators, resulting in a demand of 26 kL per day.

Figure 18-58 depicts the process flow diagram ("PFD") of the potable water distribution for Muntanga.

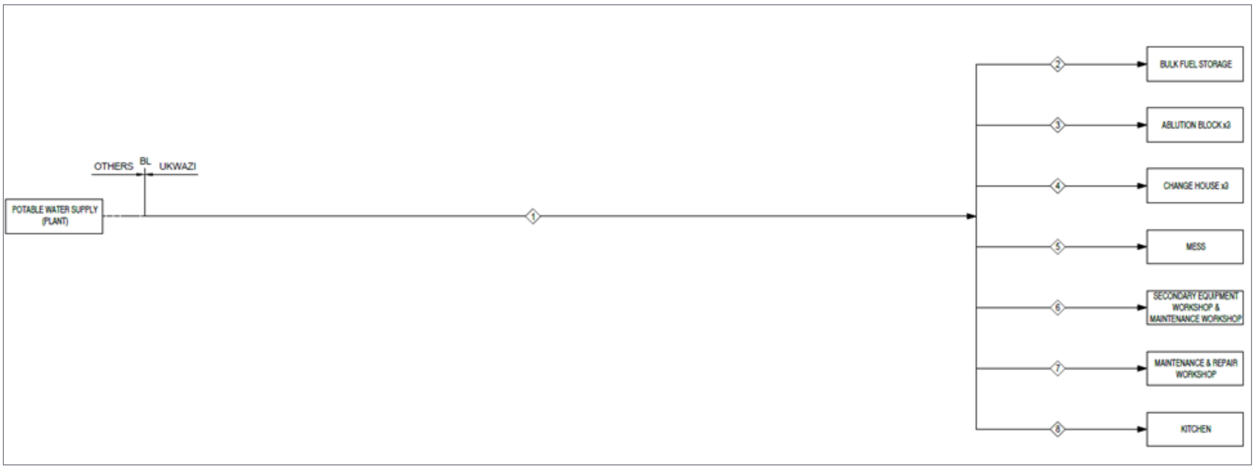

Figure 18-58: Potable water process flow diagram for Muntanga

18.2.26.2. Process water

The process water for Muntanga will be coming from the plant. The process water will be supplied to the wash bay areas, workshops, and diesel storage area. Process water is to be recirculated in a closed-loop system to conserve as much water as possible.

Figure 18-59 depicts the PFD of the service water distribution for Muntanga.

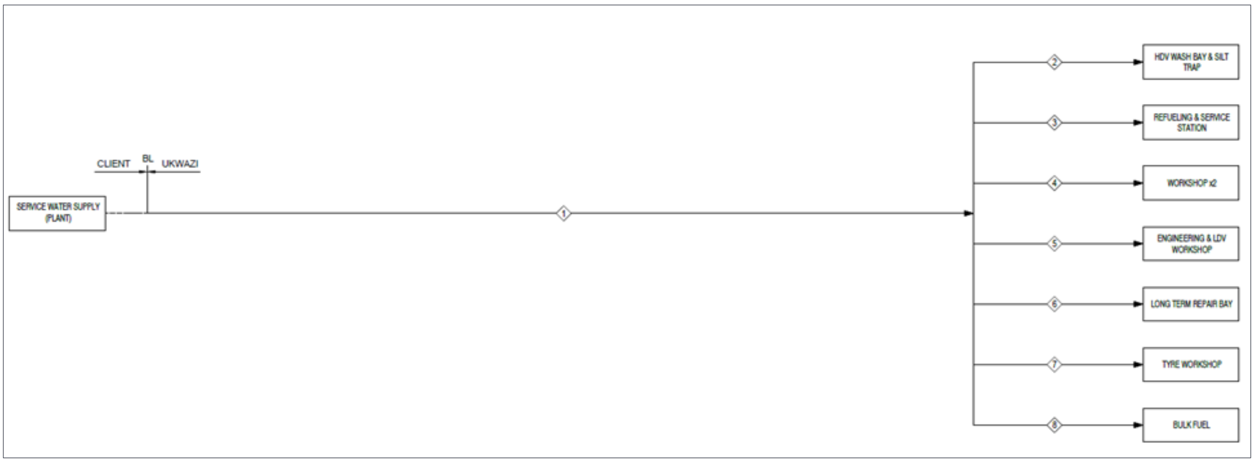

Figure 18-59: Service water PFD for Muntanga

18.2.26.3. Fire water

The fire water distribution network is independent of potable and process water networks. Fire water will be sourced from the plant. The fire water will be provided for workshops, offices and fuel station at adequate pressure of 3 Bar with all hydrants delivering 1 200 L/min. The design for hydrant fire flow will be in accordance with SANS 10090 based on the fire risk category.

Other firefighting equipment allowed for are:

- 4.5 kg STP fire extinguishers
- 5 kg CO_2 fire extinguishers
- A 50 L foam trolley.

The location of the fire hydrants was strategically placed to cover areas. The layout and the location for the fire hydrant can be seen in Figure 18-60.

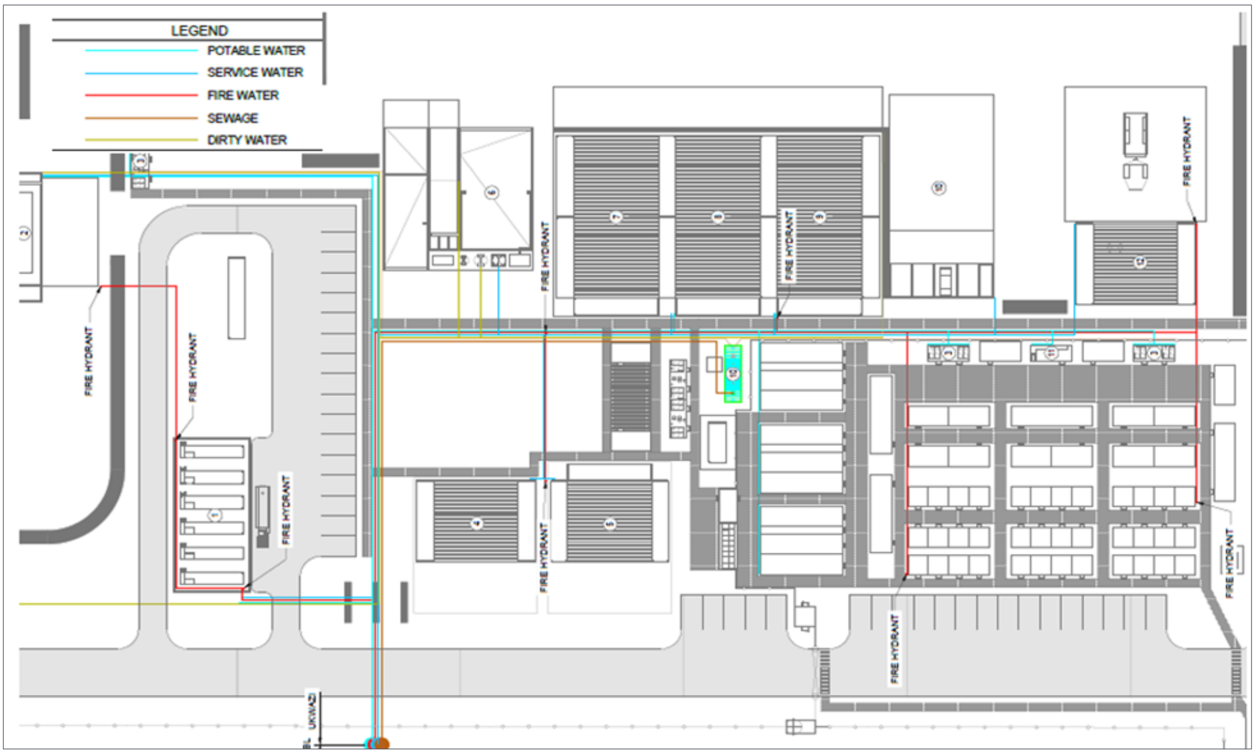

Figure 18-60: Layout and location for fire hydrants

Figure 18-61 depicts the PFD of the fire water distribution for Muntanga site.

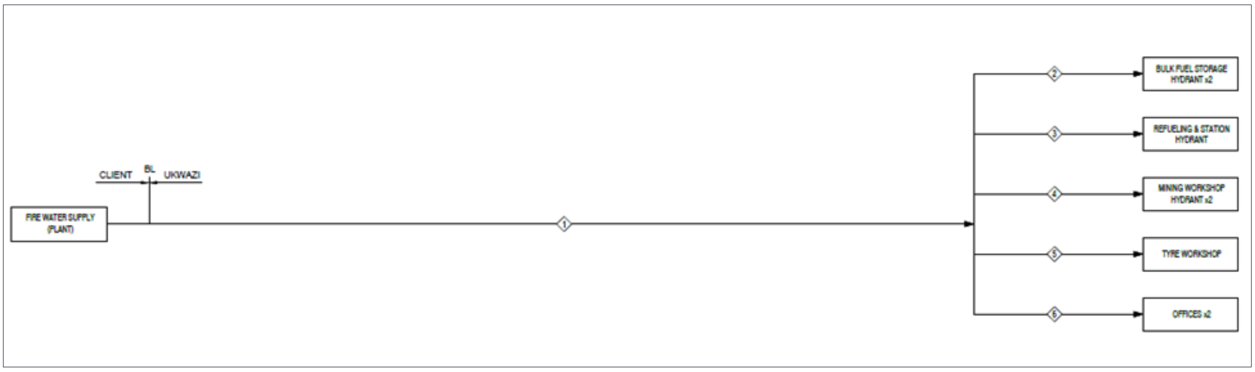

Figure 18-61: Fire water PFD for Muntanga

18.2.26.4. Dirty water

Dirty water (defined as water containing pollutants) from the workshop sump and diesel bay sump will be pumped to the wash bay sump, where it will be treated through desilting and oil separation by means of a Drizit system (or similar) before it is reused. Dirty water is to be recirculated in a closed-loop system to conserve as much water as possible. Refer to section 18.2.5 for more discussion.

Figure 18-62 depicts a flow diagram for wastewater.

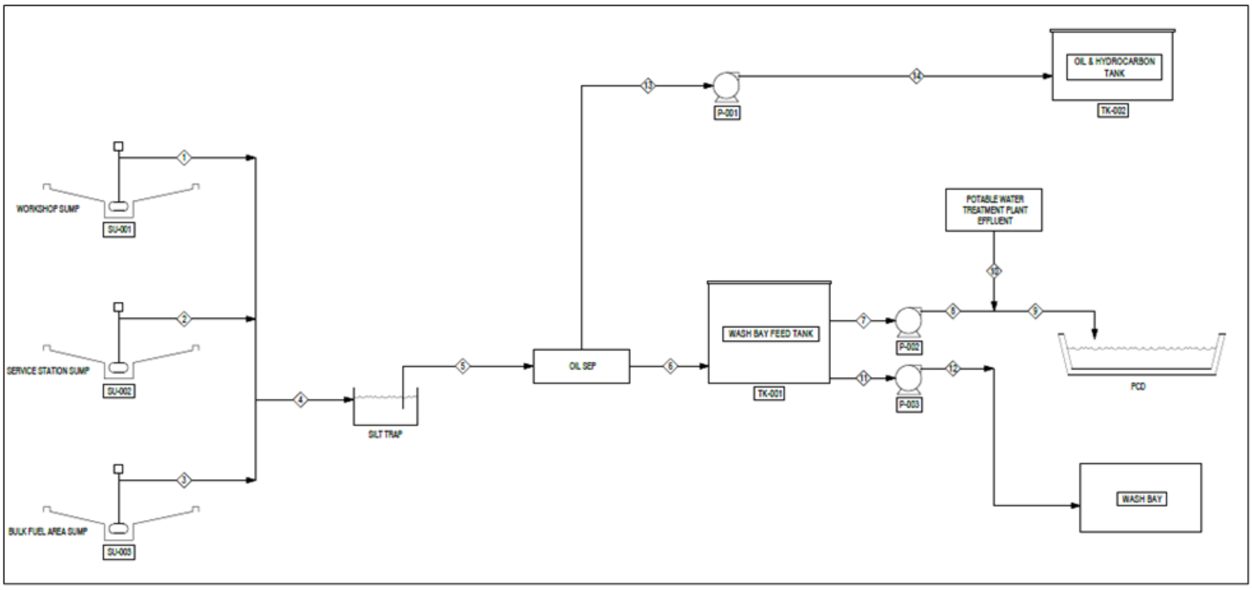

Figure 18-62: Dirty water flow diagram for Muntanga

18.2.26.5. Pipe network

Table 18-20 provides a summary of pipe network included as part of this design.

Table 18-20: Pipe network

Line routing						
	From	To	Normal flow fate [m^3/hr]	Specification	Nominal pipe size [mm]	Line length [m]
Potable water	Potable water tank	Header	15.7	HDPE	80	245
		Branch		HDPE	50	82

Line routing						
	From	To	Normal flow fate [m^3/hr]	Specification	Nominal pipe size [mm]	Line length [m]
Service water	Service tanks	Header	5	HDPE	40	267
		Branch		HDPE	25	45
Fire water	Service tanks	Header	90	HDPE	100	263
		Branch		HDPE	50	100
Pit sump dewatering	Site M	Pit to PCD pumping	195	HDPE SDR 11 PN 16	225	1 600

18.2.26.6. Pump selection

Table 18-21 shows a summary of the pumps included as part of the designs.

Table 18-21: Pump selected for supplying water from the tanks

Description	Supplier	Total head [m]	Absorbed power [Kw]	Selected Motor size [kW]
Dust suppression water feed pump	KSB	7.1	8.21	11
Workshop sump pump	Weir	4.7	1.50	7.5
Wash bay sump pump	Weir	4.7	1.50	7.5
Pit to PCD pumping	Weir	81.6	64.94	75

18.2.26.7. Sewage reticulation and management

A sewage treatment plant will be constructed to treat the sewerage and effluent from the ablutions, change house and kitchen for the entire Muntanga complex. The design of the sewage treatment plant is outside the battery limits of Ukwazi. The effluent will be treated to an acceptable quality to allow safe and efficient re-use of water on site.

Sewage will gravity feed into a buried sewage collection sump, from where it will be pumped to the treatment plant. The specified collection sump is a modular tank, with a capacity of 21.5 kL, therefore adequate to store one days' worth of sewage make.

A basic diagram of the process flow is shown below:

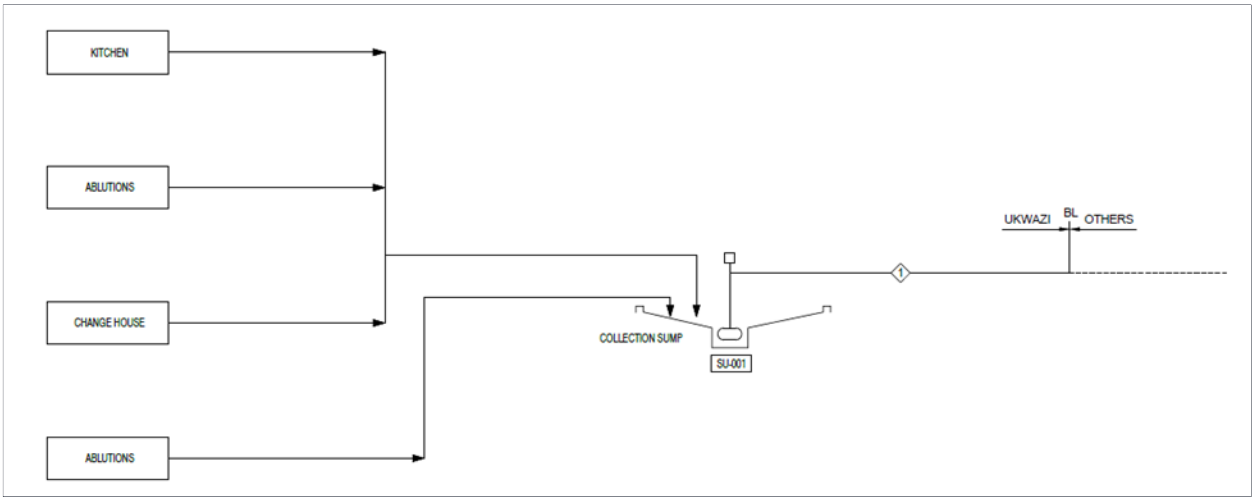

Figure 18-63: Sewage reticulation flow diagram for Muntanga

18.2.27. Communication

Communication on site is required for the coordination of a safe and productive environment. Site wide communication is achieved through the use of a very high frequency ("VHF") network on site.

VHF two-way radios are used as the main communication tool with various channels to ensure amicable traffic on the communication portals. The main communication tower is installed as part of the mining infrastructure, fully equipped with a container for power, back-up power and an office for repairs. Good coverage as shown in Figure 18-64 will be achieved on surface, but would require additional mobile units to augment the communication network for the pits at full depth. The mobile units are trailer mounted and can be relocated as mining progress.

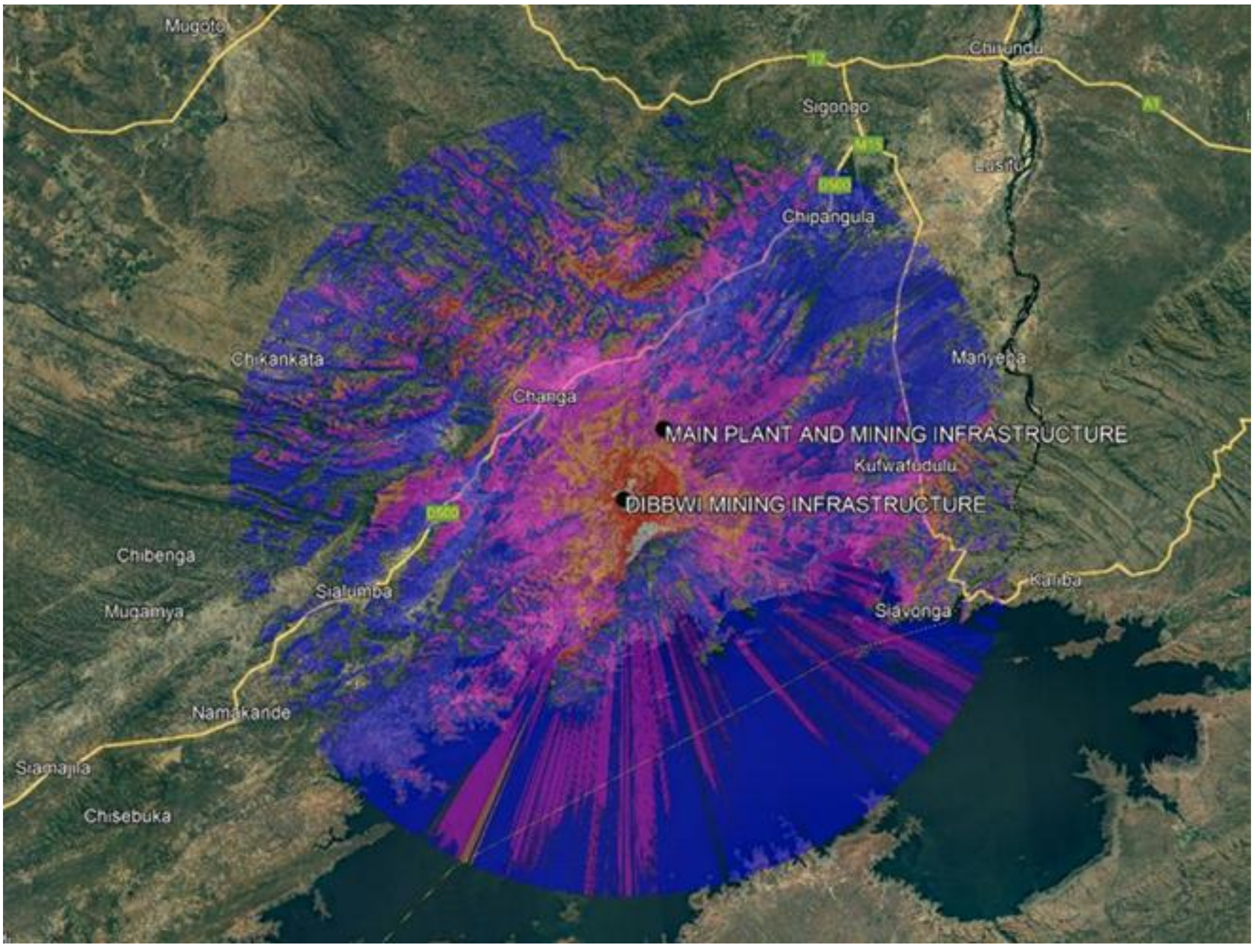

Figure 18-64: Coverage map of VFH communication

Figure 18-65 shows a typical mobile VHF trailer.

Figure 18-65: Mobile VHF trailer

The communication network includes a global positioning system “GPS” interface for the tracking of users and equipment through the Kenwood interface. A basic dispatch add-on system is available enabling additional functionality using the VHF network. KAS-20 AVL and dispatch system was included as part of the communication network and would enable basic tracking and dispatching capabilities.

The estimate is based on a quotation received from Halo technologies based in South Africa, but with operational sites in Zambia. Allowance was made for 135 mobile units installed in vehicles and additional 30 handheld units for supervisory staff and maintenance staff.

On-site support is offered by the company to maintain the equipment on site and ensuring the required up-time without impacting production.

18.2.28. Laydown and storage areas

An area has been allocated on the block plan for a laydown area. This area is approximately 40 m x 25 m. Figure 18-66 depicts the laydown area. The area is located such that it is accessible from the mining and the general area. Allowance has been made for the area to be fenced off to facilitate inventory control.

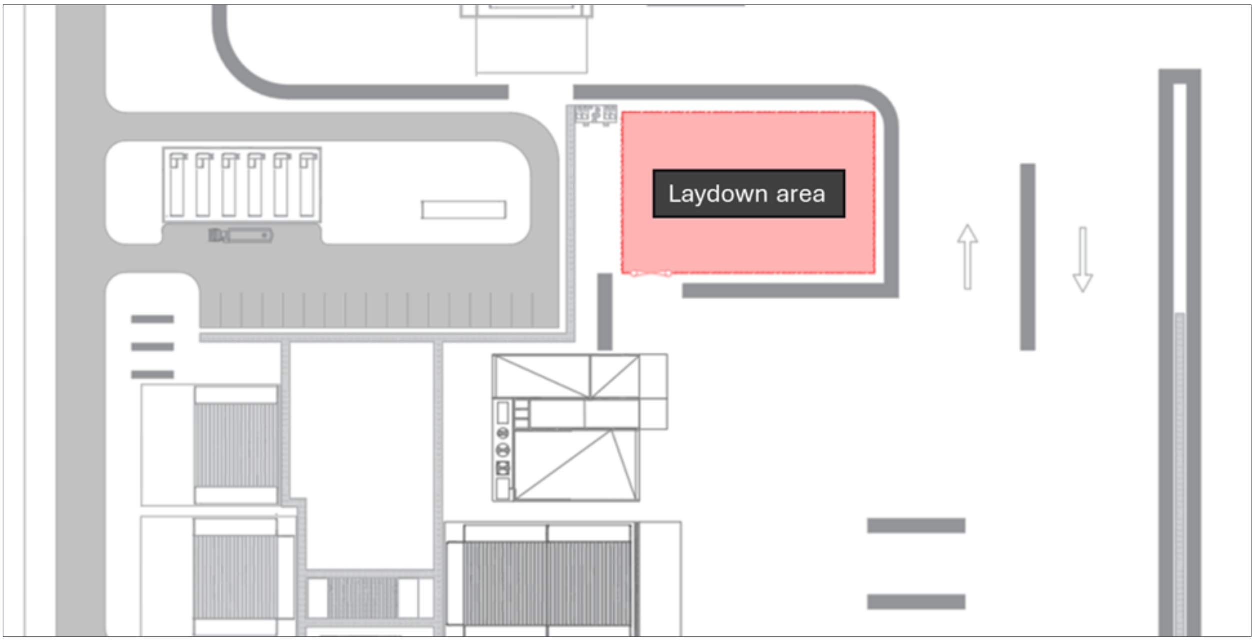

Figure 18-66: Laydown area

18.2.29. Core shed

An area has been allocated on the block plan for the storage of cores produced during drilling. No specific allowance has been made for a structure, the need for a formal structure should be reviewed in the next phase of the project.

18.2.30. Run of mine pad

As detailed in Section 16, the mining strategy requires a ROM stockpile during the initial ramp-up phase. A 150 m x 150 m stockpile area has been allocated near the primary crusher. Figure 18-67 shows the location and layout of the ROM pad. Material can be loaded and hauled to the crusher or, if suitably sized, fed directly onto the primary crusher outflow conveyor via a hopper. Water management for the ROM pad is described in the Section 18.3.

Following the geochemical specialist's recommendation the area does not require a liner or specific barrier system. The area will be cleared and topsoil stripped before stockpiling commences.

Figure 18-67: Mining ROM pad

The clearance and topsoil stripping activities will be conducted by mining operations as part of the overall mine development. Therefore, the capital cost estimate does not include a separate allowance for these preparatory works.

18.2.31. Waste rock dumps

As described in the mining section, ex-pit WRDs will be required during the mining process. Based on the geochemical tests and interpretation, the geochemical specialist informed the requirements for the barrier system and base preparation. Refer to Section 20.6 for details pertaining to the waste classification. The requirement, as provided, was that the WRD area shall be cleared from vegetation and the topsoil stripped, prior to the placement of waste materials. Water management on the WRDs are as described in Section 18.3.

The clearance and topsoil stripping activities will be conducted by mining operations as part of the overall mine development. Therefore, the capital cost estimate does not include a separate allowance for these preparatory works.

18.3. Water management

This Chapter includes the following water-related aspects:

- Surface water resources (**Section 18.3.1**)

- Floodlines (**Section 18.3.2**)
- Stormwater (**Section 18.3.3**)
- Groundwater Ingress and dewatering (**Section 18.3.4**).
- Water Quality (**Section 18.3.5**)
- Water Balance (**Section 18.3.6**)
- Excess Water Discharge Management (**Section 18.3.7**)
- Water Supply Scheme (**Section 18.3.8**)
- Water Monitoring Programme (**Section 18.3.9**).
- Water Supply Recommendations (**Section 18.3.10**).

A summary of each of the above tasks is provided below. Details on each of the tasks are provided in the **Water Management Report** (Parts I, II, and III).

18.3.1.Surface water resources

The Zambezi River and Lake Kariba were initially considered as surface water options; however, they were discarded due to the fact that they are shared international water resources between Zambia and Zimbabwe. In addition, the closest abstraction point from these two resources which could supply the mining areas would be much further in comparison to the Kafue River and groundwater sources, making it unattractive.

18.3.1.1. Lusithu River

The Lusithu River was identified as the only potential surface water resource option to supply the central processing operations as it is the largest river in the vicinity of the mine. Although it is an ephemeral river, it was considered in terms of capturing flows during the wet season through storage to supply the mine during the dry season. Detailed modelling was performed to analyse this river to assess the wet season flows. It is a tributary of the Zambezi River. The headwaters of the Lusithu River catchment are approximately 60 km to 80 km east of the project site, flowing westerly towards the mine, with a total catchment area of 1 964 km^2 (see Figure 18-68).

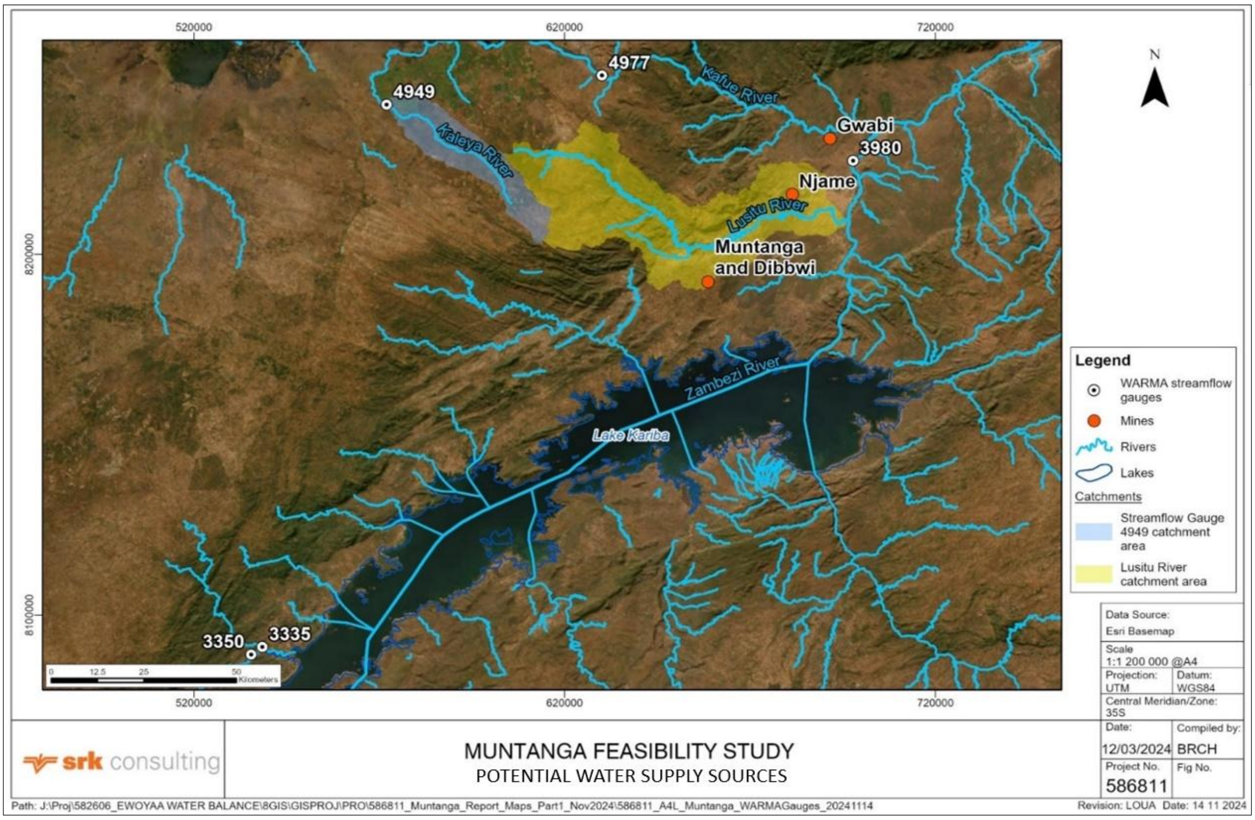

Figure 18-68: Mine location map in relation to surface water resources

A hydrological assessment was performed using the most used rainfall-runoff model in Southern Africa (the Pitman Model). The model was set up from scratch using a simulation period of 1981 to 2021 (hydrological years). CHIRPS rainfall data was used and showed large interannual variations. The wet season is from November to March, while the dry season (June to October) often shows minimal to zero rainfall. The mean annual precipitation ("MAP") of the

Lusithu River catchment is 702 mm per annum ("/a") (however; for modelling purposes, sub-catchments were delineated and modelled, varying between 674 mm/a and 736 mm/a). It is clear from the time-lapse Google Earth images (available from 1985 to current) that, as per most of the rivers and small dams in the catchment, the Lusithu River is seasonal (non-perennial).

The model is typically calibrated using observed streamflow data; however, no such data is available for the Lusithu River. Therefore, modelling was based on available knowledge of catchment characteristics (such as vegetation and soils) using previous studies/reports, and visual observations from site visits. The area is largely natural and not generally impacted by anthropogenic factors, such as dams, large abstractions (cities), commercial irrigation, etc. The calibration process also included a comparison of model parameters for similar catchments (MAP, mean annual evaporation ("MAE"), terrain, soils etc.) and other Zambian studies near the project area which used the Pitman Model. The relevant parameters for these model setups were collated, and the most appropriate parameters for the Lusithu River were selected. The model was further adjusted to account for the high variability in flow, specifically for the lack of flow in the dry periods of the year.

The modelling results showed that the mean annual runoff ("MAR") at potential abstraction points to supply Muntanga/ Dibbwi and Njame with water are 58 million m^3/a and 78 million m^3/a, respectively. However, these average MAR results are exceptionally skewed due to the higher flows in the wet seasons and almost no flow in the dry seasons, rendering it an unfavourable water supply resource for the mine.

When the rainfall from the climate change assessment was applied to the modelling, using the worst-case scenario (a decrease of 3.4 %), the results showed a minor but insignificant decrease in runoff.

18.3.2. Floodlines

Floodline delineation was carried out for the rivers traversing and in the vicinity of the proposed Project mining infrastructure. This task involved the identification of the rivers that will impact the proposed mining area, thereby assisting in identifying suitable areas to position the infrastructure and ensure that the identified areas are not impacted during flooding events. Floodlines were determined for the four mining areas, split as follows:

1. Gwabi mining area
2. Njame mining area
   - Njame North
   - Njame Middle
   - Njame South
3. Muntanga and Dibbwi East mining area
4. Dibbwi mining area.

18.3.2.1. Floodline determination

The largest river which may impact the Project is the Kafue River which flows in an easterly direction on the northern side of the proposed Gwabi Mine. Rivers within the Project area are generally seasonal and primarily experience surface runoff only during the wet season. Within the applicable catchments, there are rural areas with scattered villages, and areas of natural vegetation, and development is minimal.

18.3.2.2. Input data

Data required for floodline determination included the following:

- Digital Elevation Model ("DEM") in the form of 1m contours (obtained from MMG) and 2m contours (high accuracy elevation data provided by Intermap Technologies, a software company that produces the most comprehensive, homogenous, and precise elevation products at global scale) were used for the delineation of the catchment areas, calculation of the flood peaks, and to abstract cross-sectional data of the watercourses and floodplains
- Rainfall data required for hydrological modelling of catchments draining into the Muntanga Mine rivers and estimation of peak flows was sourced from the CHIRPS climatic gridded model for the period 1981 to 2019 and extended to 2022 with the daily rainfall data provided by AMC
- Land use was required for the determination of impervious areas and Manning's n-value to estimate the runoff potential of the area. The land use for the area was based on the available land cover obtained from the topographical information, Google Earth satellite data, a site visit carried out in May 2022, and the relevant literature for the different hydrological models.

Figure 18-69 illustrates the floodlines for Muntanga and Dibbwi East.

18.3.2.3. Hydrological and hydraulic analysis

Two hydrological models (the Rational Method for catchments < or = 30 km^2 and the alternative rationale for catchments >30 km^2) were used to estimate the peak flows along rivers within the four mining areas. Hydraulic modelling was carried out using HEC-RAS, a model that employs a standard backwater technique to compute the high-water level for various steady flow conditions, taking into account control structures across the watercourse. The 1:50 and 1:100-year floodlines were plotted and mapped for each mining area.

18.3.2.4. Flood remediation interventions

Flood mitigation measures such as river diversions and flow attenuation dams will be implemented along the rivers to prevent flooding of the Muntanga mine infrastructure including access and haul roads impacted during the 1:50-year and 1:100-year flooding events.

18.3.2.5. Climate change impacts

The impact of climate change on floodlines was assessed for the Northern Njame Mine catchment which served as a proxy for the entire region. The 1:50-year and 1:100-year floodlines were revised based on the existing HEC-RAS model and the estimated peak flow rates by incorporating the projected impacts of climate change. The highest increase in rainfall for the near-term scenario was chosen as input into the model, to represent the worst-case scenario of flooding. The highest increase projected for the near term is +3.9 % for the one-day maximum rainfall. The results of the climate change impact assessment on the existing floodlines showed the following:

- The 3.9 % increase in the one-day maximum rainfall resulted in a variance of <10mm flood level, which is an insignificant change in the peak flow rates
- The adjusted peak flows under projected climate change conditions resulted in an insignificant change in the floodlines and water surface elevations along rivers within the Njame Mine area
- The results indicate that projected climate change impacts will have an insignificant impact on the floodlines of all streams within the Project.

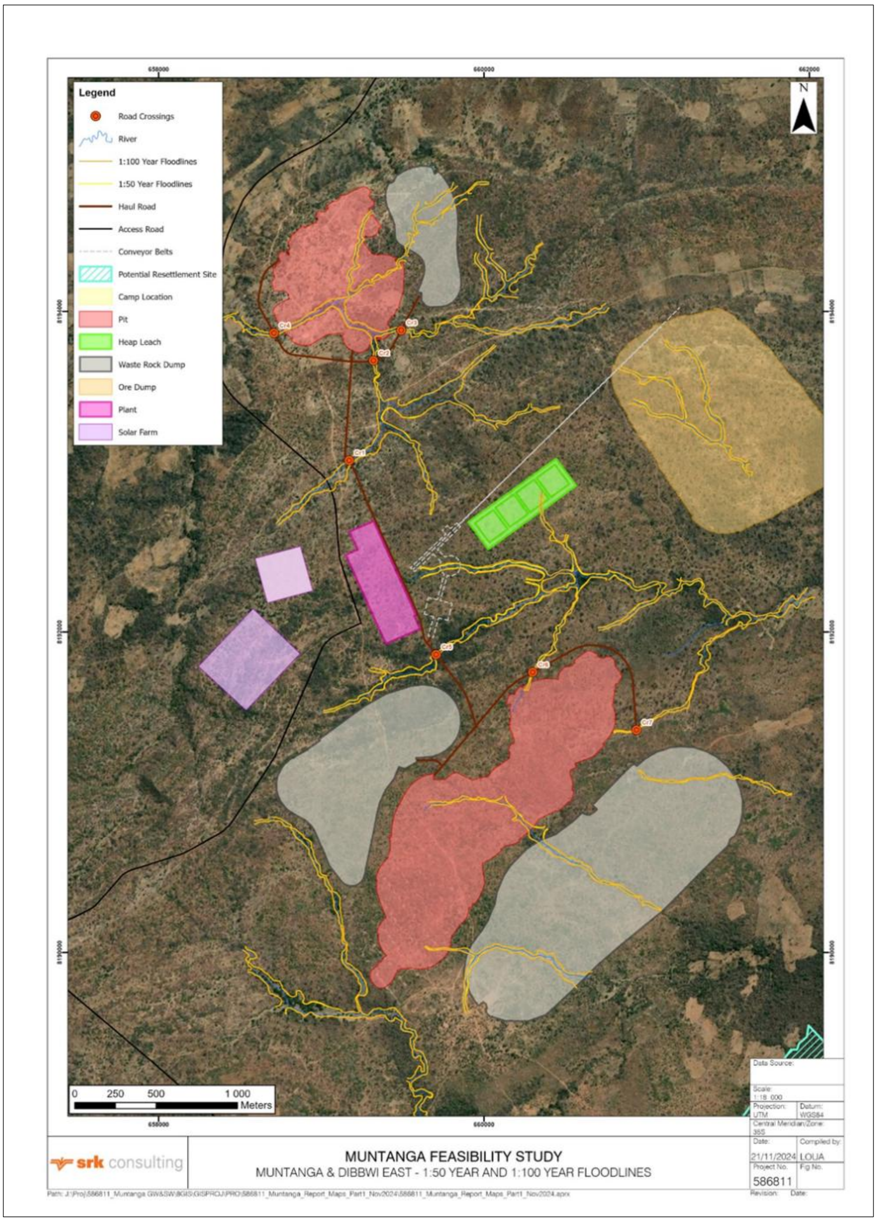

Figure 18-69: Muntanga and Dibbwi East floodlines

18.3.3.Stormwater

The stormwater management task involved the development of stormwater measures that will ensure that the proposed infrastructure and activities at the mine are not negatively impacted during flooding events. Stormwater management at Muntanga and Dibbwi East considered the latest mine layout indicating the positions of each mine's infrastructure.

The following principles were applied in the development and implementation of a stormwater management plan for the Muntanga mine:

- Contact water must be collected and contained in a system which is separate from the non-contact water system, thereby minimising the risk of spillage or seepage into clean water systems
- The stormwater management plan must be sustainable over the life cycle of the mine and different hydrological cycles
- The statutory requirements of various regulatory agencies and the interests of stakeholders must be considered and incorporated.

The stormwater management measures focussed on the following key mine infrastructure within each mine site:

- Pit
- WRD
- HLP
- Mine plant
- Haul Road.

18.3.3.1. Input data

Bentley FlowMaster, an efficient program for the design and analysis of a wide variety of hydraulic elements, such as pressure pipes, open channels, weirs, orifices, and inlets was used to size the stormwater controls and determine the channel characteristics. Data used in sizing the stormwater channels and storages that will divert runoff away from mine infrastructure to prevent damage and scouring and store the contact water from potentially contaminating components included the following:

- A DEM comprised of 1 m contours (obtained from MMG) was used in delineating the sub-catchments draining into each stormwater control structure
- Design rainfall depths estimated from the CHIRPS climatic data and the daily rainfall provided by AMC used for the calculation of peak flows for the 1:50-year and 1:100-year storm events
- Land use data to aid in the determination of the percentage of impervious areas and Manning's n-value.

18.3.3.2. Stormwater management

Stormwater management at the Muntanga mining sites will ensure that affected stormwater originating from the potentially contaminated areas (pits, WRDs, the mine It is recommended that paddocks around the WRDs be constructed to capture side slope runoff. The stormwater collected in these paddocks can either be evaporated or allowed to seep into the ground. Plants and the HLPs) are diverted by lined channels and berms and contained in pollution control dams ("PCDs")/ settling ponds. The stormwater management controls at Muntanga mine are designed to have the capacity to contain and convey a 1:100-year storm event without spilling.

All non-contact water originating from the upstream catchments and generated from unimpacted areas within each mine site will be diverted by stormwater channels, berms, and culverts to prevent mixing with the contact water from the infrastructure mentioned above. The diverted non-contact runoff will be discharged towards the existing drainage lines/ watercourses. In addition, some of the stormwater will flow directly towards the mine infrastructure and be stored in clean water ponds positioned at the lowest points for use in mining activities, or to allow for evaporation and infiltration.

The proposed stormwater controls for the mining sites are shown in Section 18.3.7 for Muntanga and Dibbwi east.

18.3.4. Groundwater ingress and dewatering

The groundwater task for the Project aimed to achieve several key objectives, as follows:

- To estimate groundwater inflows into the pits and analyse the distribution of pore pressure around the pit high wall throughout the lifespan of the mine
- To utilise historical and fieldwork data for evaluating groundwater quality and identifying water supply options for the Muntanga, Dibbwi, Dibbwi East, Njame and Gwabi pit areas
- To develop a dewatering, depressurisation and groundwater management plan tailored to the planned mining infrastructure, ensuring practicality and effectiveness.

A series of field investigations were conducted to attain these objectives. These investigations involved the installation and pumping tests of nine hydraulic test boreholes ("HTHs") and three water supply boreholes. Water samples were collected from 12 strategically selected boreholes to indicate water quality. The data collected from these field investigations were essential for understanding the aquifer characteristics and hydrogeological modelling. Two numerical models were developed to test dewatering scenarios and post-closure pit rebound. The Muntanga, Dibbwi, and Dibbwi East pits were modelled together, while the Gwabi and Njame pits were modelled separately. The results informed the development of a tailored groundwater management plan for the Project area.

A conceptual hydrogeological model was developed as a foundation for constructing a numerical groundwater model. The model included geological formations, recharge and discharge areas, hydrological features, groundwater flow and levels, and hydrogeological unit boundaries.

18.3.4.1. Hydrogeological setting

The main aquifers are characterised by undifferentiated sandstones of the Escarpment Grit Formation , classified as fissured aquifers with varying productivity. The main aquifers are characterised by undifferentiated sandstones of the Escarpment Grit Formation , classified as fissured aquifers with varying productivity. Regional geological structures, including faults and fractures, play a significant role in controlling the movement and storage of groundwater within the aquifers, as evidenced by the spatial variability in the hydraulic properties of the aquifers (Figure 18-70). The northeast faults facilitate preferential groundwater flow, evidenced by higher yields and conductivity rates in boreholes near these faults. The aquifers are primarily recharged by rainfall, with groundwater flow influenced by the regional geological structures.

For the Muntanga, Dibbwi, and Dibbwi East sites, the aquifers are hosted in the fractured and faulted Escarpment Grit Formation, with water levels ranging from 13.37 mbgl to 62.46 mbgl. Groundwater flow is generally oriented southeast at the Muntanga and Dibbwi areas and east at the Dibbwi East area. Muntanga's coarser-grained ‘Braided facies’ have higher effective porosities and more brittle fracturing, leading to higher groundwater yields. Despite the similarities between Dibbwi and Dibbwi East surface geologies, Dibbwi East has higher hydraulic conductivity and groundwater yields due to the number of faults in this area compared to Dibbwi.

The main aquifers in the Njame and Gwabi areas occur in the fractured consolidated sandstone units, which occur along the contact between the Escarpment Grit and the Madumabisa Formations. The Gwabi area potentially has a deeper aquifer in a fractured consolidated unit and some shallow aquifers in the unconsolidated interbedded Escarpment Grit Formation. The ridge on the Njame site has shallow weathering zones with fractures only in the fresher, consolidated rock, resulting in lower groundwater yields. However, the low-lying areas and fault zones on the ridge may have higher yield potential due to preferential groundwater pathways.

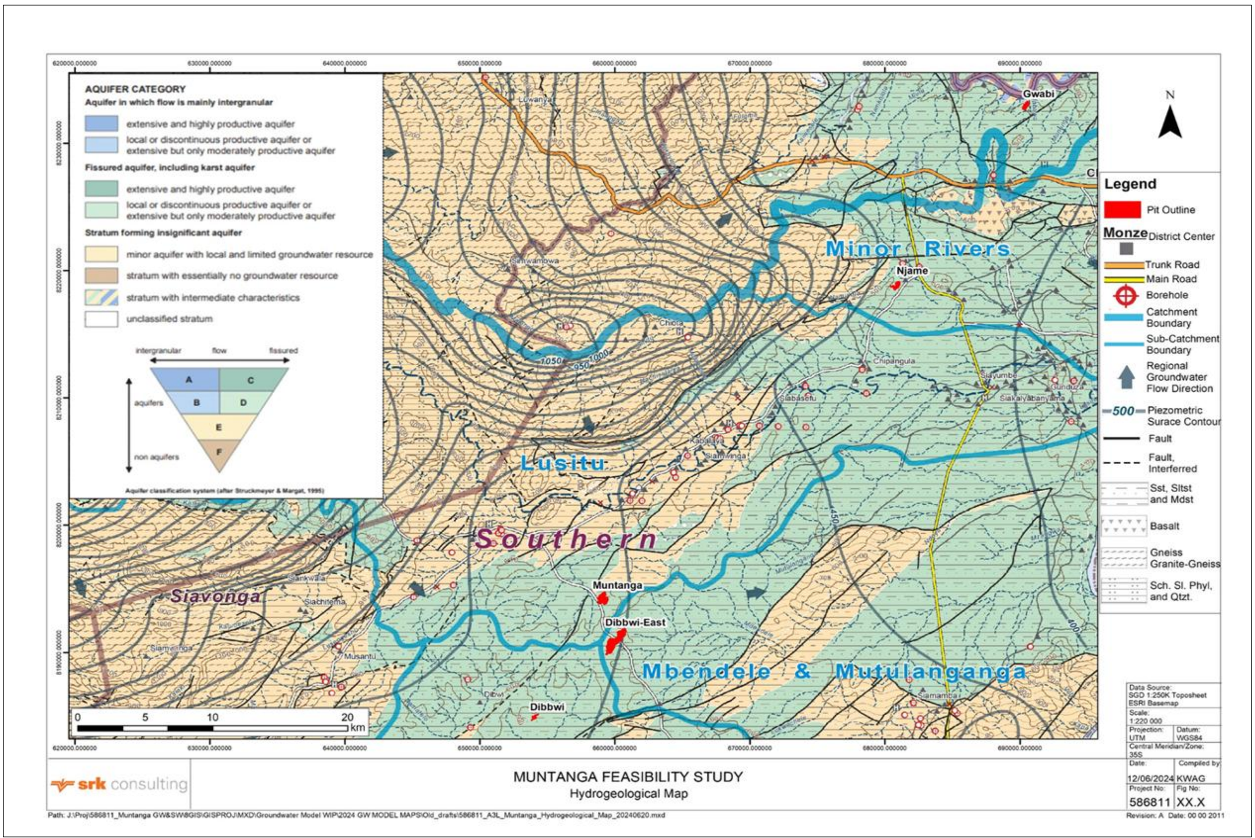

Figure 18-70: Hydrogeological map

18.3.4.2. Drilling and pumping tests

GoviEx, with remote support from SRK, drilled nine HTHs and 18 monitoring boreholes. Specifically, one HTH was drilled at the Gwabi, Njame, and Dibbwi mines, while three HTHs were drilled at both Muntanga and Dibbwi East mines. Each HTH, with a nominal diameter of 152 mm (6 inches), was accompanied by two monitoring boreholes with a nominal diameter of 102 mm (4 inches). The average borehole depth was approximately 100 m, with some boreholes reaching depths of up to 150 m.

Pumping tests were conducted to determine the hydraulic properties of the aquifers, including hydraulic conductivity, transmissivity, and storativity (Table 18-22). Following SANS 10299-4:2003 guidelines, pumping tests were conducted on all nine HTHs to determine the aquifers' hydraulic conductivity. The tests involved step drawdown tests

("SDT") with four one-hour steps, followed by constant discharge tests ("CDT") and recovery measurements. The key findings are as follows:

- The Muntanga, Njame, and Dibbwi East aquifers exhibited the highest hydraulic conductivity rates, averaging 1.3 m/day, 0.9 m/day, and 1.4 m/day, respectively. The Gwabi and Dibbwi aquifers had the lowest average conductivity rates of 0.65 m/day and 0.09 m/day, respectively
- Transmissivity values varied across the project area, ranging from 5 m^2/day to 66 m^2/day, with the Muntanga aquifer having the highest and the Dibbwi aquifer having the lowest transmissivity rates
- Borehole yields for the project area ranged from 4.1 L/s to 20.3 L/s, with boreholes near northeast-trending faults yielding the highest groundwater volumes
- The potential groundwater production from aquifers is sufficient to meet the long-term water supply needs of mines and surrounding settlements.

18.3.4.3. Groundwater levels

The static water level ("SWL") observed in the drilled boreholes varies significantly across the project site. This regional variation in SWL highlights the complex hydrogeological dynamics of the project area. The key findings are as follows:

- Boreholes at the Dibbwi-East site have the deepest SWL, ranging from 53.05 metres below ground level (mbgl) to 62.46 mbgl. In comparison, the boreholes at the Gwabi site exhibit artesian properties with the shallowest SWL at 0.0 mbgl
- The SWL at the Muntanga, Njame and Dibbwi sites ranges between 13.75 mbgl to 35.9 mbgl, 19.29 mbgl and 21.12 mbgl, and 28.35 mbgl to 32.04 mbgl, respectively
- Despite the variability in SWL, the hydraulic head as metres above mean sea level ("mamsl") values remain relatively consistent across the Dibbwi, Dibbwi East, and Muntanga sites, indicating a likely low regional groundwater gradient
- The Gwabi and Njame areas have low hydraulic heads due to their low-lying nature.

Table 18-22: Muntanga pumping test data and results

	Borehole name	MTDTH 1560			MTDTH 1576	MTDTH 1580	MTDTH 1566				MTDTH 1585	MTDTH 1583	MTDTH 1570			MTDTH 1574	MTDTH 1577
Borehole details	Depth (m)	100			70	70	100				100	100	80			70	70
	Pump depth (mbgl)	70			70	70	94				94	94	40			40	40
	Static water level (mbgl)	23.01			23.75	23.90	36.50				36.96	36.07	15.50			15.90	19.02
SDT	Steps	1	2	3			1	2	3	4			1	2	3		
	Pumping rate (L/sec)	3.2	4.1	4.5	N/A	N/A	4.9	7.6	10.6	14.8	N/A	N/A	3.7	4.8	5.5	N/A	N/A
	Total drawdown (m)	4.99	6.22	6.59			9.37	12.14	18.04	51.90			15.14	22.11	44.23		
CDT	Pumping rate (L/sec)	4.47					11.40						5.47				
	Total drawdown (m)	7.64			6.37	6.66	55.39				3.00	3.49	17.48			6.02	5.48
	Recovery duration (min)	1800			3690	1800	300				300	180	1080			840	960
Calculated hydraulic parameters	Transmissivity (m^2/d)	25			23	28	4.9				81	35	48			58.40	57
	Hydraulic conductivity (m/d)	0.449			0.386	0.496	0.098				4.276	1.755	0.900			1.718	1.948
Main lithology		Sandstone, mudstone, siltstone, gritty sandstone					Sandstone, mudstone, siltstone, gritty sandstone						Sandstone				

	Borehole name	DMDTH 1513				DMDTH 1613	DMDTH 1618	DMDTH 1609				DMDTH 1596	DMDTH 1600	DMDTH 1364			DMDTH 1630	DMDTH 1620
Borehole details	Depth (m)	130				130	120	130				130	130	140			130	130
	Pump depth (mbgl)	120				120	120	120				120	120	130			120	120
	Static water level (mbgl)	53.05				52.14	55.03	58.71				59.96	61.08	59.82			61.56	62.42
SDT	Steps	1	2	3	4			1	2	3	4			1	2	3		
	Pumping rate (L/sec)	9.5	12.6	15.0	18.0	N/A	N/A	10.0	14.6	18.1	19.9	N/A	N/A	9.95	15.80	20.03	N/A	N/A
	Total drawdown (m)	2.96	3.88	4.63	6.54			7.41	8.45	6.97	16.99			67.42	69.74	72.14		
CDT	Pumping rate (L/sec)	17.79						20.31						20.05				
	Total drawdown (m)	16.68				4.14	2.39	29.13				3.24	2.67	23.80			3.17	2.08
	Recovery duration (min)	480				360	1440	120				n/a	n/a	300			45	180
Calculated hydraulic parameters	Transmissivity (m^2/d)	34				38	58	23				86	73	23			110	150
	Hydraulic conductivity (m/d)	1.147				1.598	1.524	0.437				2.213	1.485	0.284			1.833	2.517
Main lithology		Sandstone, mudstone, siltstone, gritty sandstone						Sandstone, mudstone, siltstone, gritty sandstone						Mudstone, siltstone, sandstone, gritty sandstone				

18.3.4.4. Numerical modelling and results

Given the distance between the deposits, two separate numerical groundwater models were developed. The first model includes Muntanga, Dibbwi East and Dibbwi deposits, and the second includes Gwabi and Njame deposits.

The numerical models predict significant groundwater inflows, particularly at Dibbwi East, necessitating a robust dewatering strategy to manage water ingress effectively. Pit ingress for various predictive scenarios was simulated (Table 18-23) and the resulting inflows are graphically shown in Figure 18-71 and Figure 18-72. The maximum predicted passive residual inflows, i.e. inflows into the pits with dewatering, are summarised below:

- Muntanga – 650 m^3/d
- Dibbwi East - 2 400 m^3/d
- Dibbwi - 1 120 m^3/d
- Gwabi - 1 300 m^3/d
- Njame – 400 m^3/d.

Table 18-23: Summary of predictive scenarios

Scenario		Description
1	Predictive simulations with pits only	Baseline case
2	Predictive simulations with pits and dewatering boreholes	3 x Dewatering boreholes @ Muntanga 3 x Dewatering boreholes @ Dibbwi East 1 x Dewatering borehole @ Dibbwi 1 x Dewatering boreholes @ Gwabi and @ Njame
3	Predictive simulations with pits and additional dewatering boreholes	2 x Dewatering boreholes @ Muntanga 9 x Dewatering boreholes @ Dibbwi East 3 x Dewatering boreholes @ Dibbwi 4 x Dewatering boreholes @ Gwabi 3 x Dewatering boreholes @ Njame
5	Predictive simulations – post-closure	No pumping post-closure

In all cases, inflows are generally at low and manageable levels with dewatering. Only towards the end of mining does an increase occur due to the declining effectiveness of dewatering boreholes as a consequence of head decline. The actual response of the groundwater table to dewatering during operations must be used to inform management of the system and also maintain inflows to within a target limit.

As a result of the dewatering, a zone of drawdown will be created around each pit.

- Dibbwi East pit experiences the largest drawdown due to its larger extent and depth compared to the other pits. The cone of drawdown is extensive but largely within the mining area (Figure 18-73). At the periphery of the cone of drawdown, there may possibility of reducing water levels and yields in neighbouring communities. The risk however is low
- The cone of drawdown is localised around the Muntanga pit, but eventually, the Dibbwi East drawdown will also encompass this area
- Dibbwi pit is generally localised within approximately 1km of the pit, with moderate to low drawdowns of about 6 m
- The zone of drawdown for the Gwabi pit is restricted to the immediate vicinity of the pit, extending close to the Kafue River. There may be some moderate impact on adjacent fish and agricultural farms, with drawdown expected to be less than 5 m for neighbouring users
- The Njame cone of drawdown is confined to the immediate vicinity of the pit, with minimal expected impact on neighbouring farms and communities.

The groundwater intercepted will be used for mining and process plant needs, and excess water will be released into the environment via a dam. A general guideline limit of less than 1 000 m^3/d as groundwater contribution to pit sumps was targeted to limit the sump size requirements in the pit. Ex-pit dewatering boreholes were therefore simulated to identify the number of boreholes required, likely yields and passive residual inflows into the pits. This information allowed for a detailed water balance to be constructed and make-up water to be determined.

<table>
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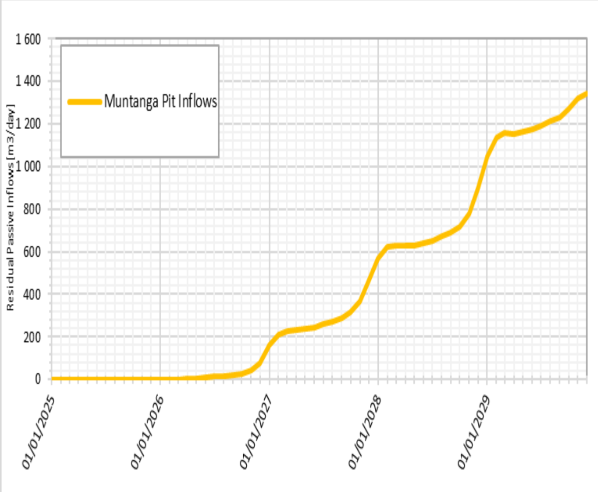
<td>
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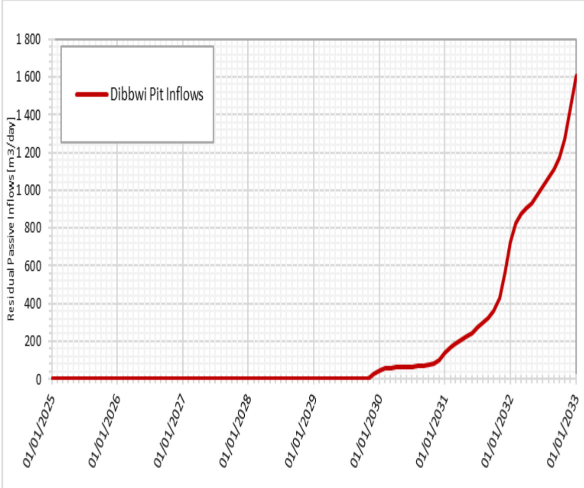
<td>
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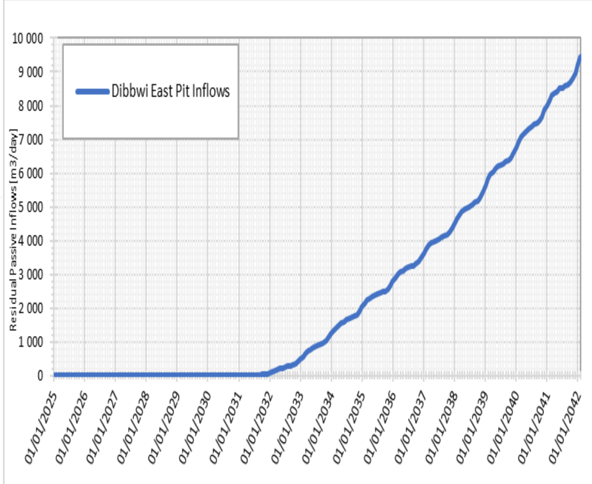
<td>
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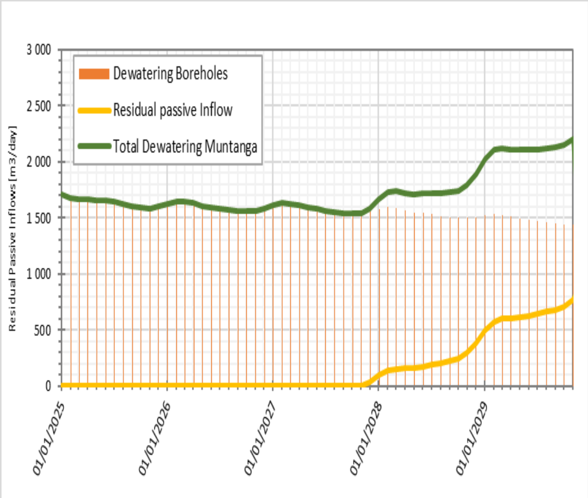
<td>
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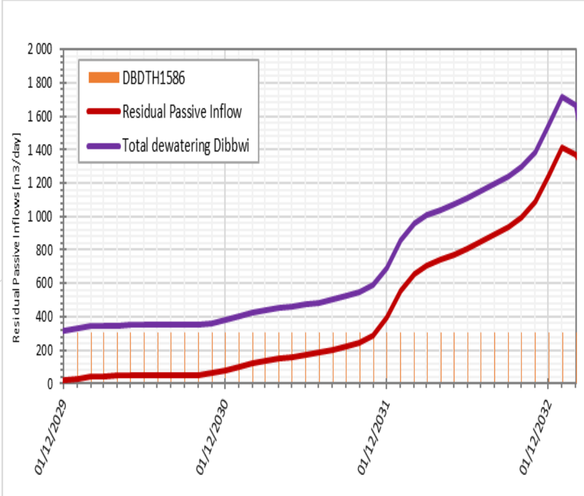
<td>
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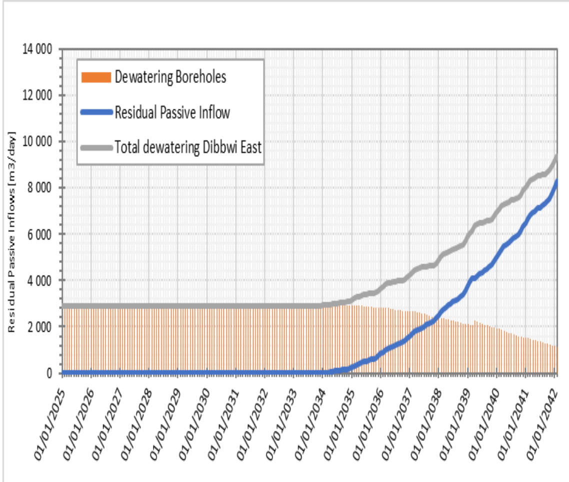
<td>
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Scenario 3

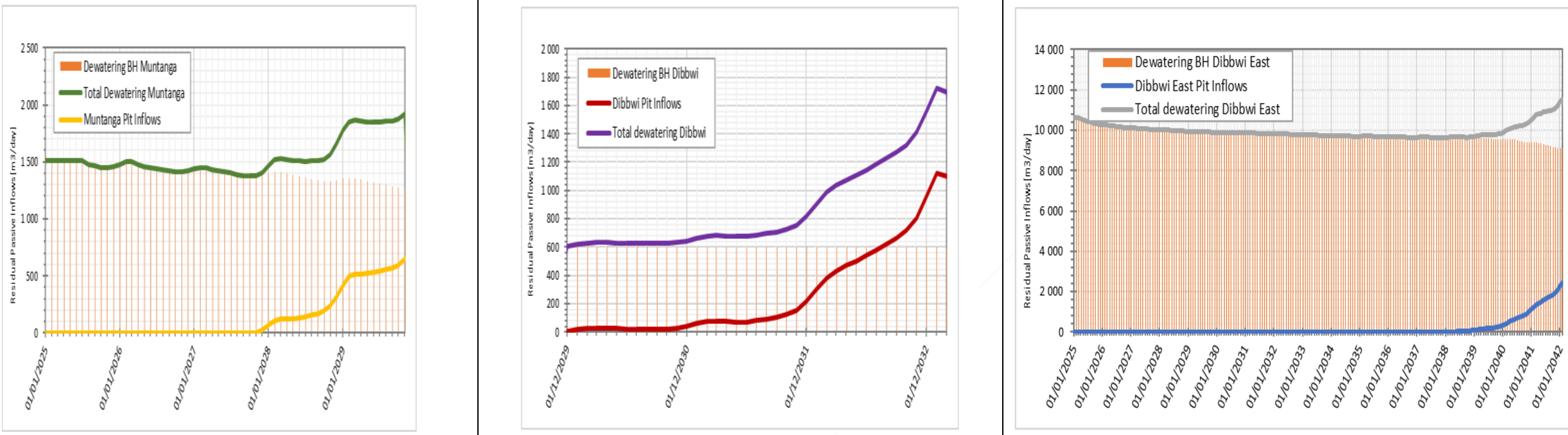
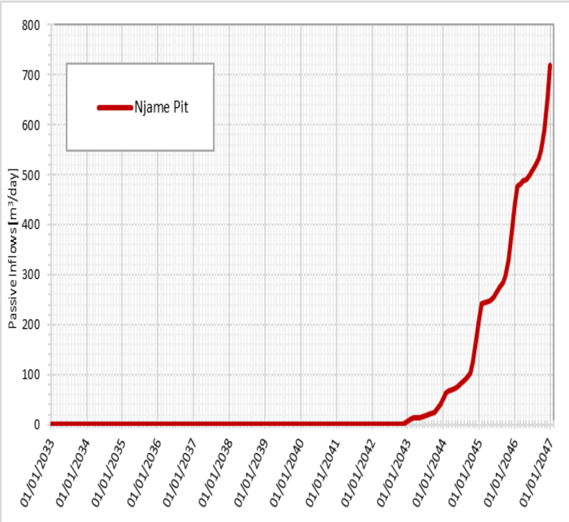
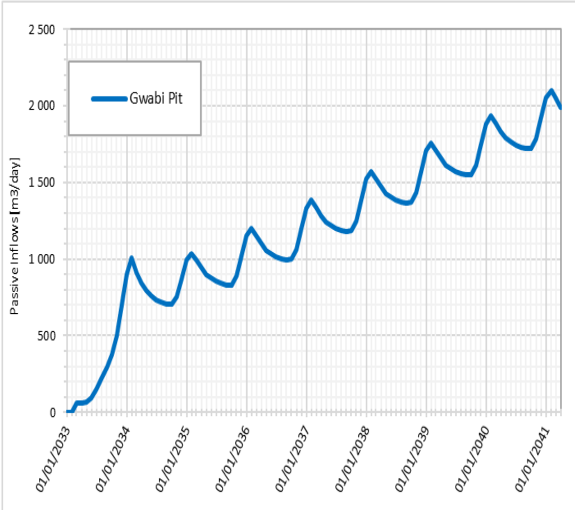


Figure 18-71: Pit inflow estimates for various scenarios for Muntanga, Dibbwi and Dibbwi East

<table>
<tr><th>Site</th><th>Njame</th><th>Gwabi</th></tr>
<tr><td>Scenario 1</td><td>
</td><td>
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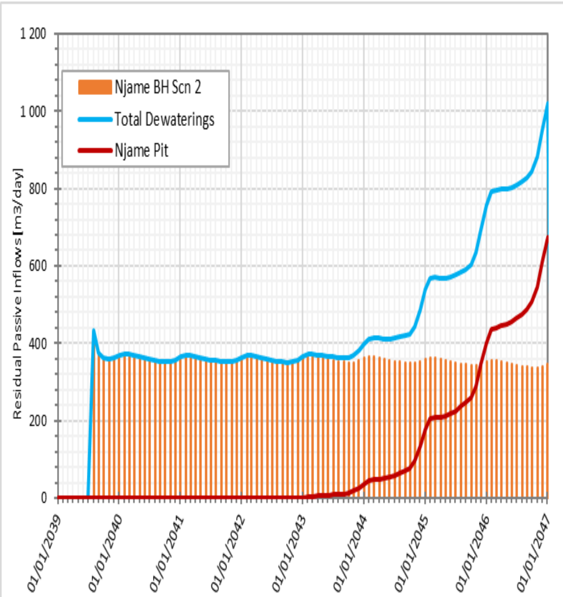
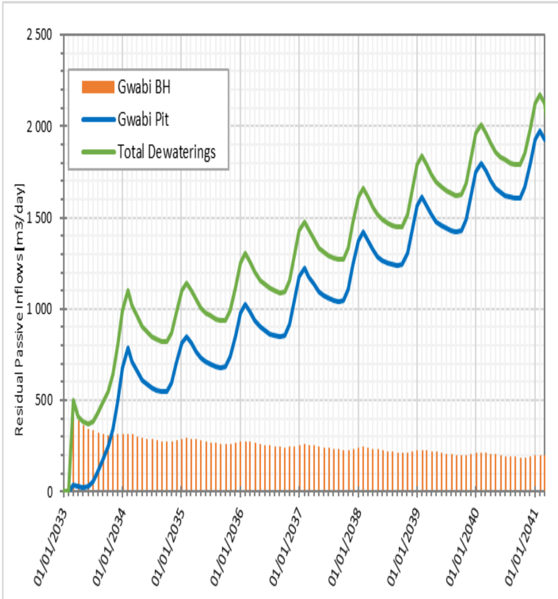
<tr><td>Scenario 2</td><td>
</td><td>
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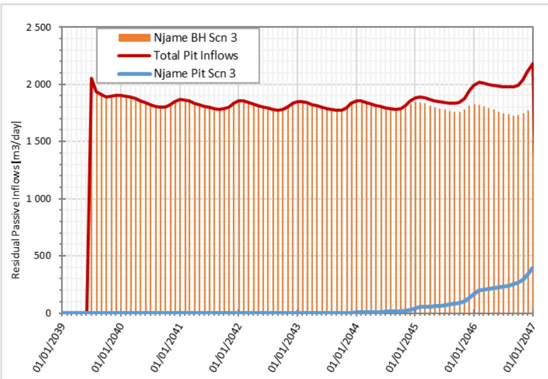
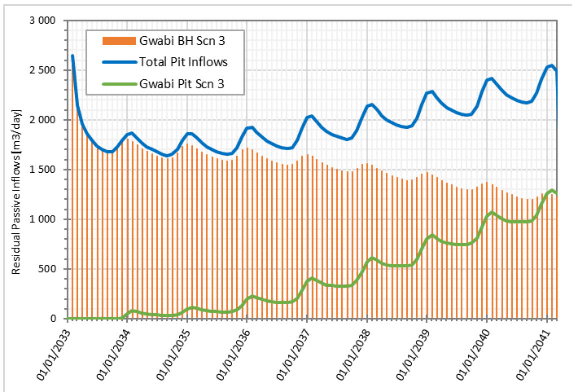
<tr><td>Scenario 3</td><td>
</td><td>
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</table>

Figure 18-72: Pit inflow estimates for various scenarios for Njame and Gwabi

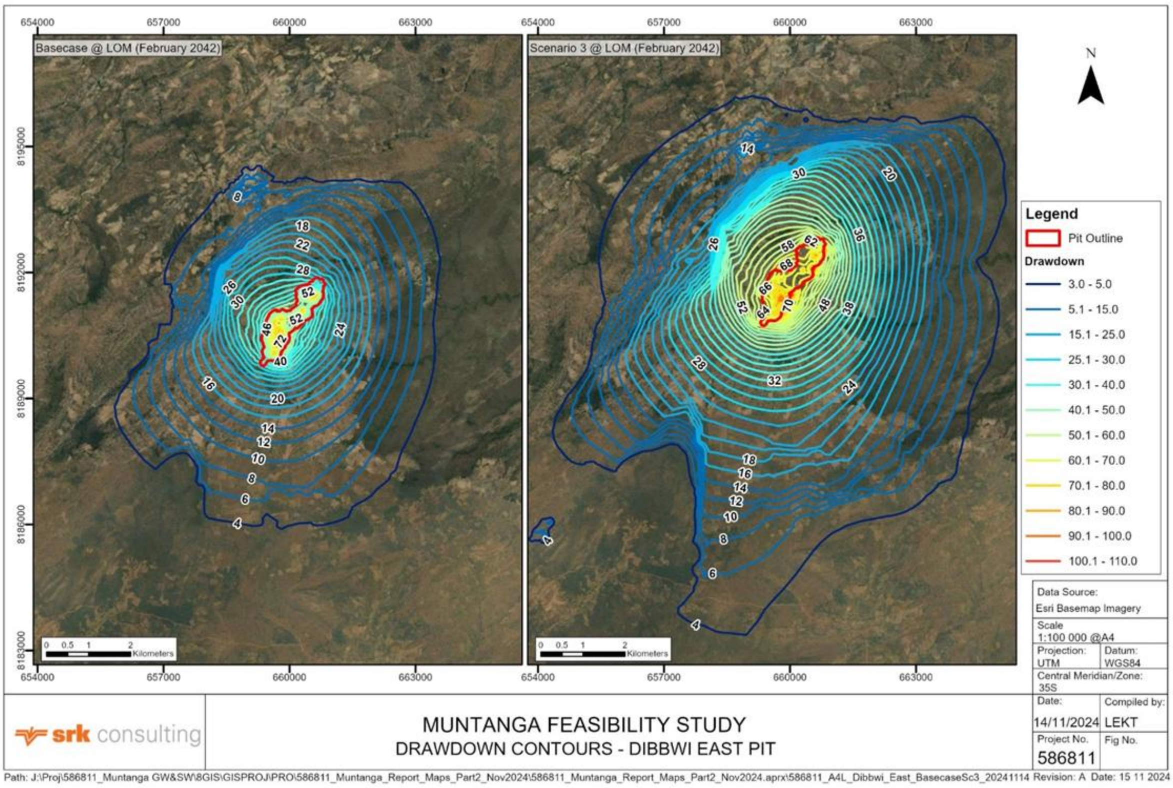


Figure 18-73: Drawdown contours – Dibbwi East Pit

18.3.4.5. Dewatering of the pits

The dewatering plan must be continuously assessed, updated and refined based on monitoring data, changes to mining plans and schedules, and improved understanding of the conceptual hydrogeological model, especially the role of the various structures and lithological variability, recharge rates and additional hydrogeological characterisation. As per the relevant environmental and mining permits, the conceptual hydrogeological and numerical models will need periodic revision to synthesise the information. The groundwater monitoring and numerical analysis will be used to improve dewatering operations, which can materially impact mining costs and slope failure risk.

As per the numerical modelling results, pumping boreholes will be strategically drilled in addition to the already drilled HTHs around the mining sites to keep the working area dry and safe. Based on the numerical model, a dewatering scheme summarised in Table 18-24 is proposed. The yields, pump size and yields are indicative and will need to be confirmed during detailed design and construction. The scheme will need to be refined before construction, using baseline monitoring data and improved geological information, detailed (monthly or bi-annual) pit shells and schedules, and revised hydrogeological conceptualisation. Recharge rates will be a key component of any revised estimate.

Table 18-24: Dewatering borehole scheme

Dewatering borehole	X	Y	Z	Depth	Max. pumping rate		Pump size
	m	m	mamsl	m	m^3/day	m^3/sec	[kW]
Muntanga							
MTDTH1566	659 002	8 194 545	493	100	259	0.00300	5.5
MTDTH1560	658 482	8 193 817	465	100	1 240	0.01435	22
Dibbwi East							
DMDTH1364	659 845	8 190 244	471	140	640	0.00741	11
DW1	660 427	8 191 772	433	150	1 178	0.01363	26
DW2	659 949	8 191 163	453	150	1 178	0.01363	26
DW3	659 351	8 190 557	468	150	1 178	0.01363	26
DW4	659 833	8 190 083	438	150	1 178	0.01363	26
DW5	660 046	8 190 514	453	150	1 178	0.01363	26
DW6	660 558	8 191 135	443	150	1 178	0.01363	26
DW7	660 831	8 191 567	433	150	1 178	0.01363	26
DW8	659 318	8 189 988	456	150	1 178	0.01363	26
Dibbwi							
DBDTH1586	654 286	8 185 216	463	120	302	0.00350	5.5
DW10	654 187	8 184 757	458	130	302	0.00350	5.5

The mine dewatering team must maintain the dewatering system and ensure all boreholes are pumped continuously and at a rate that propagates the maximum drawdown. Any pump failures must be addressed immediately. Therefore, the mine dewatering team will need to ensure:

- Daily inspection of all dewatering boreholes is carried out
- Relevant equipment - pump spares, replacement pumps, discharge pipes etc. are readily available and
- Relevant skilled personnel to undertake repairs and install replacement pumps and equipment are available.

The effectiveness of the dewatering system must be reviewed at quarterly intervals. This is imperative as any unaccounted pore pressure build-up and increase in inflows will have an impact on mining operations.

Once mining is complete, groundwater ingress and direct precipitation will lead to pit lakes forming. Significant post-mining inflows are expected at Dibbwi East (2 600 m^3/day to 2 900 m^3/d) due to the local cone of drawdown persisting. Other pits, such as Muntanga and Dibbwi, will have lower inflow rates, varying seasonally. In all cases, the pit lakes, due to groundwater infilling, are unlikely to decant and lake stage will stabilise below the pit crest. The water level rebound in the pits will vary, with some pits experiencing a slow rebound due to the persistence of a cone of drawdown. For instance, the Dibbwi East pit is expected to reach a steady-state elevation of 540 mamsl approximately 30 years after mining ceases.

The modelling results underscore the importance of strategic dewatering to manage groundwater inflows and maintain safe mining conditions. The models provide a framework for optimising borehole placement and refining water management strategies as more data becomes available. The potential impacts on local communities and the environment necessitate ongoing monitoring and adaptive management.

18.3.5.Water quality

Following the pumping tests, groundwater samples were collected in December 2023 from the nine HTHs and three additional points: DMD1807, CM046 and MC083 (Figure 18-74 and Figure 18-75. The samples were submitted to Elements Laboratory and NECSA Laboratory and were analysed for:

- Elements Laboratory - pH, EC, Total Alkalinity, TSS, Cl, SO_4, P, Ca, Mg, Na, K, Al and metal scan
- NECSA Laboratory - Gross alpha, beta U, Th, Ra, and Po.

A second set of samples were collected in May 2024 by GoviEx. AMC collected surface water samples from the Kafue River and the Kasungu, Lusithu and Machinga streams in May 2023. AMC sampled community wells in May 2023. Given that the drinking water supply boreholes tap the same aquifers, the water quality data was evaluated against the drinking water standards set out in the Zambia Bureau of Standards (“ZABS”) Drinking Water Standards (“ZDW”) and the ZABS Effluent Discharge Standard Limits (“ZED”). Samples collected from the rivers and streams were compared to the Kafue Catchment (“KC”) guideline standards.

18.3.5.1. Surface water quality results

The results were evaluated to establish compliance with the KC guidelines limits. The following results are highlighted:

- The lower reaches of the Lusithu and Kasungu streams have a greater sediment load, which is reflected in TSS levels ranging between 164 mg/L to 266 mg/L. This is likely after storm events as these streams mainly flow only after storm events
- The Lusithu stream was recorded as having low salt and hence TDS concentration levels, with marginally elevated Fe concentrations. There is no discernible change in water quality along the stream.
- The Kasungu stream was found to have elevated levels of metals, Al, Fe, Pb and Mn but with neutral pH and salt concentrations within the catchment limits. This result would need verification, and the river should be resampled and form part of regular monitoring
- The Machinga stream has high concentrations of NO_3-N (6.53 mg/L) and Na (25.7 mg/L), which are slightly above the KC guideline limits of 6.0 mg/L and 20 mg/L, respectively.

18.3.5.2. Groundwater quality results

The results were evaluated to establish compliance with the ZDW and ZED standard guidelines limits. The following results are highlighted:

- Fe, Mn, SO_4 and electrical conductivity (“EC”) exceed the recommended ZED guideline limits across all deposits (Figure 18-76)
- The SO_4 level across all deposits except at Muntanga is relatively elevated by comparison with the other salt ion concentrations
- Muntanga has overall better water quality in comparison to the other deposits, with an EC of less than 347 µS/cm. Boreholes at Muntanga, Dibbwi, and Njame have lower concentration levels, which gradually increase sequentially from Muntanga to Dibbwi, Dibbwi East, and Njame, with a steep increase at Gwabi, especially at the GWDTH 1657 borehole (Figure 18-76). The artesian water at the GWDTH 1657 borehole is likely deeper, older water
- The water quality at Gwabi is poor, with elevated levels of SO_4, Na, Ca, Mn and Fe. The HTH boreholes drilled into the orebody seem to be of poorer quality than nearby community wells. The poor water quality may be related to its low elevation, local agricultural activity and the influence of the orebody geology
- Most trace elements are within the ZED and ZDW guideline limits except for Manganese and Iron. Fe-rich goethite is known to host a significant portion of uranium mineralisation in the project area. The P and Mn concentrations are high because the uranium mineralisation in Karoo deposits is often associated with phosphorus and manganese mineralisation (Cairncross, 2004)
- According to the World Health Organization (“WHO”) guidelines, the ingestion of Mn-rich water does not pose a significant threat to consumer health. However, elevated phosphorus levels in groundwater can potentially lead to adverse health effects, particularly kidney diseases.

There are some discrepancies between the two sample sets (December 2023 and May 2024), and careful sampling and laboratory analysis are required.

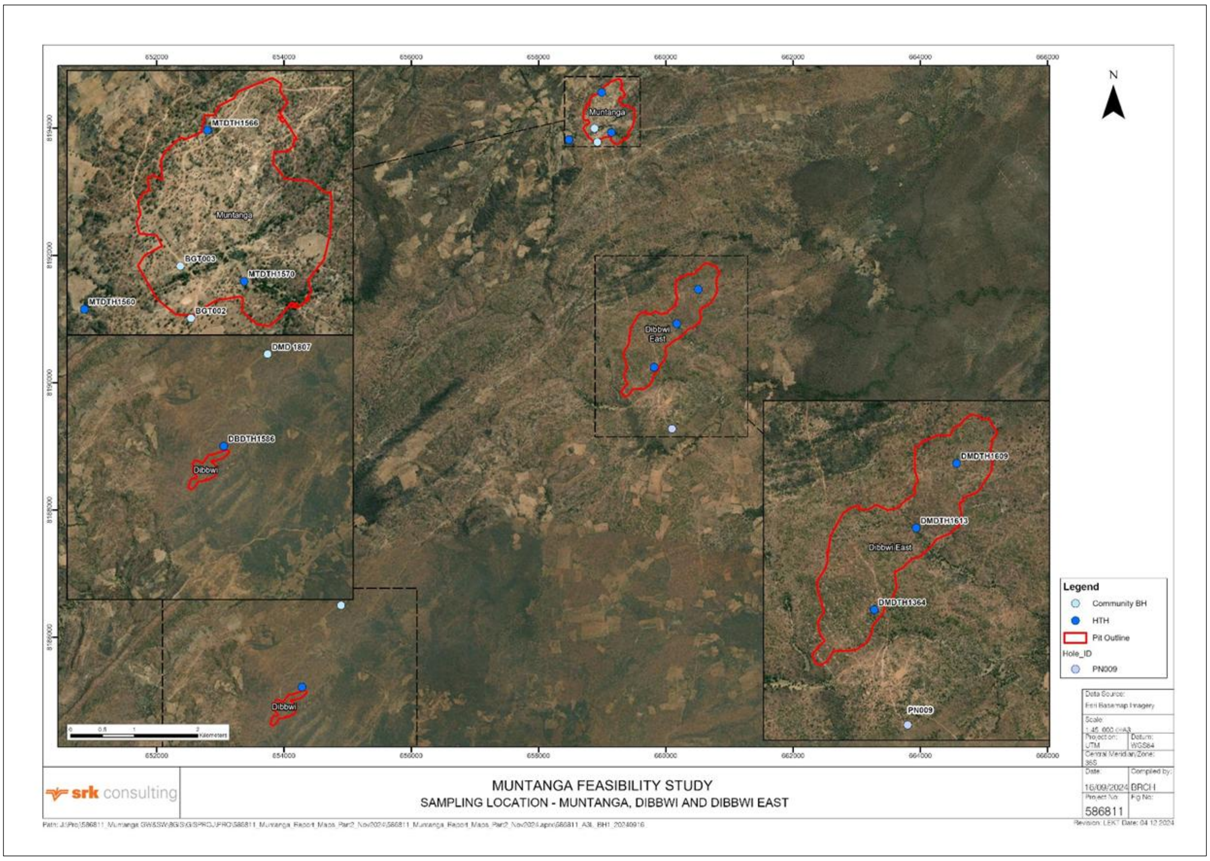


Figure 18-74: Sample location – Muntanga, Dibbwi and Dibbwi East

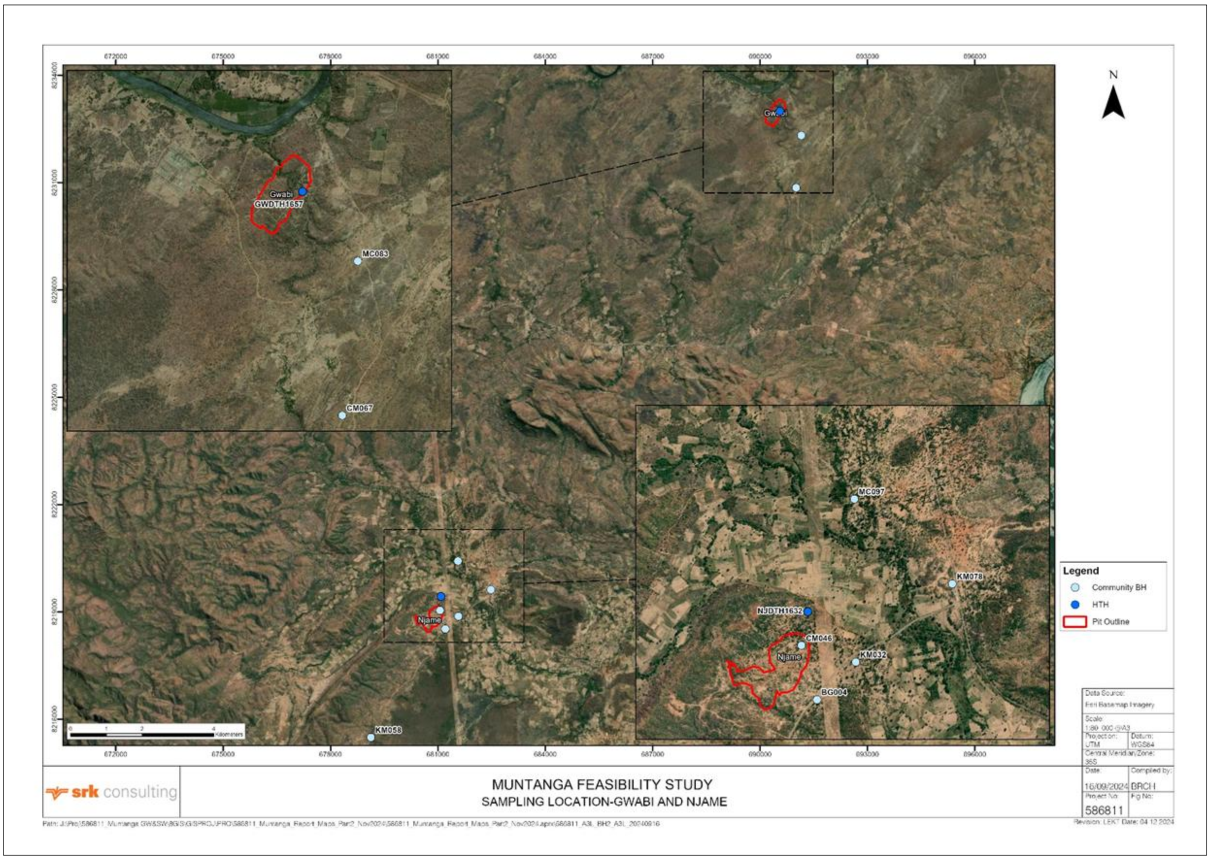


Figure 18-75: Sample location – Gwabi and Njame

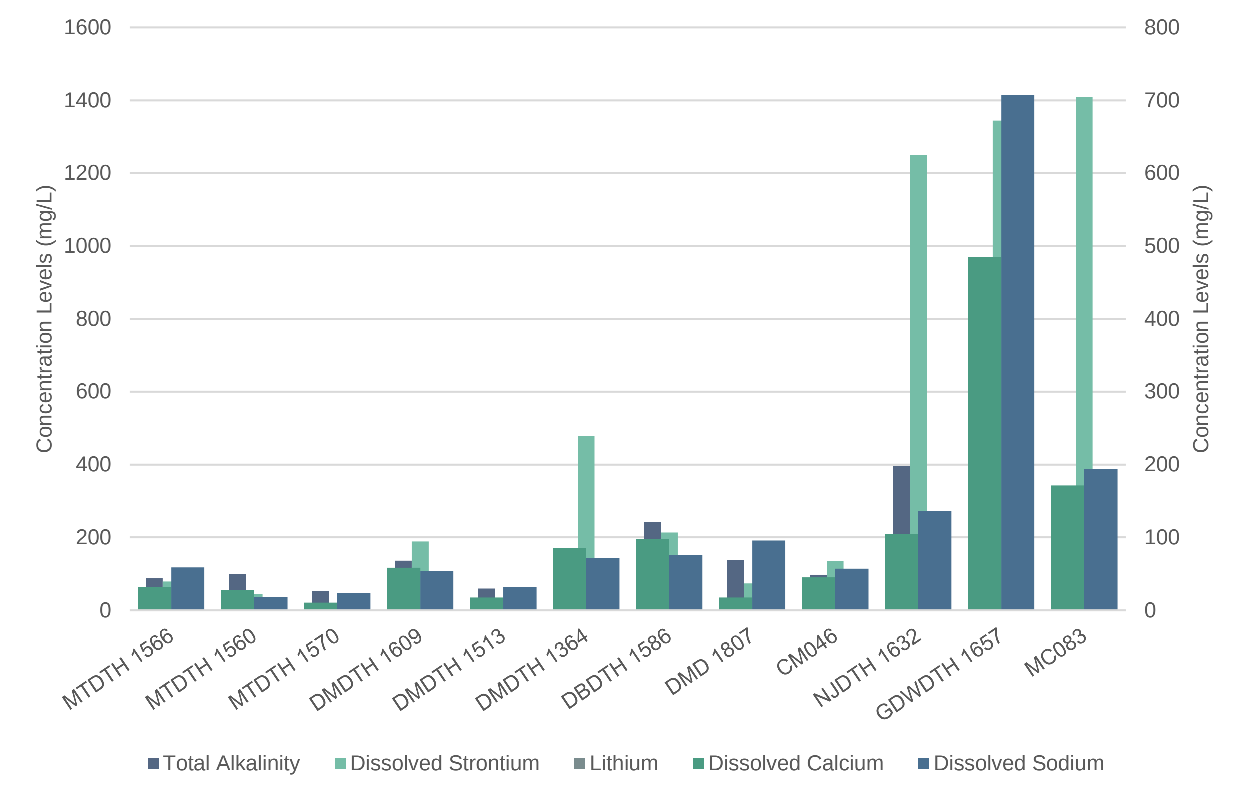


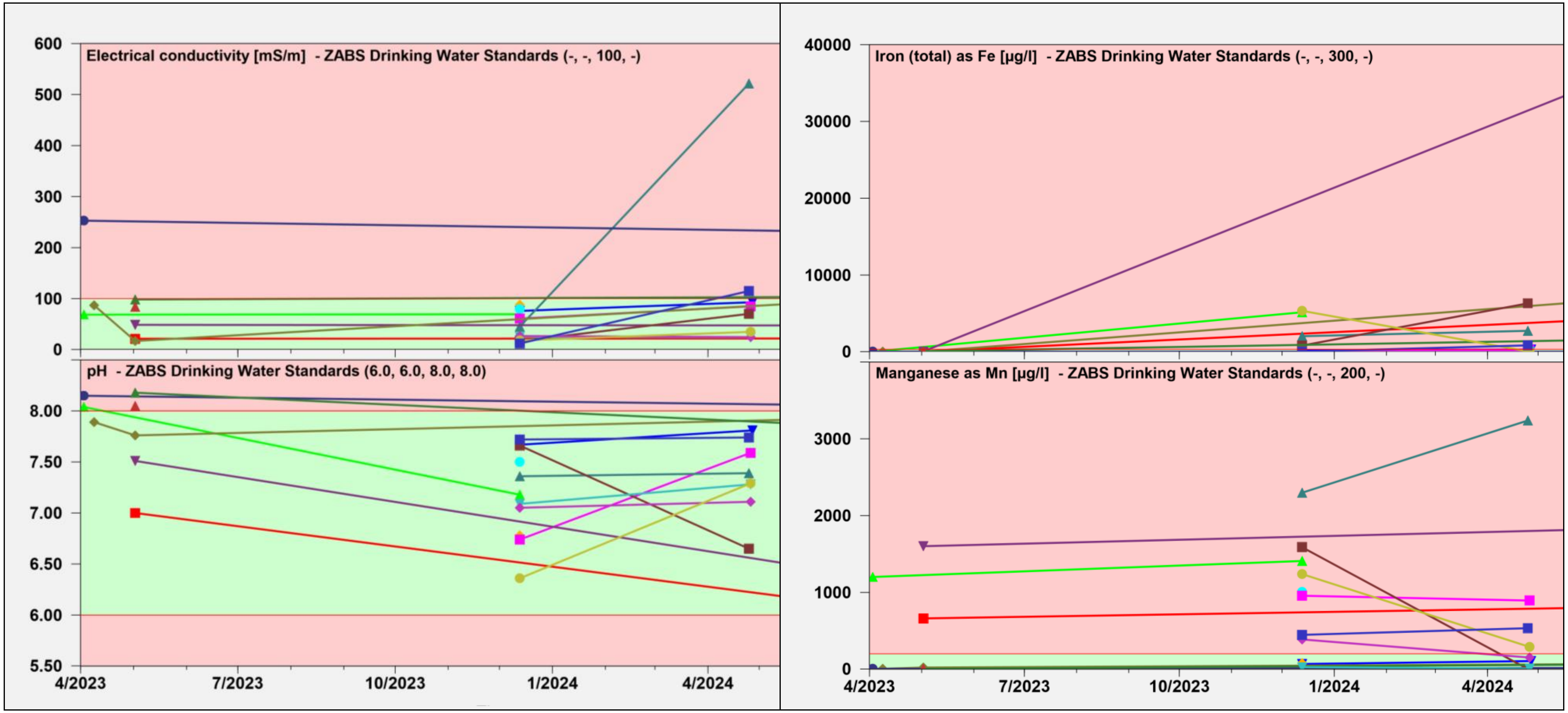
Figure 18-76: Regional trends for five of the analysed parameters from the sampled boreholes

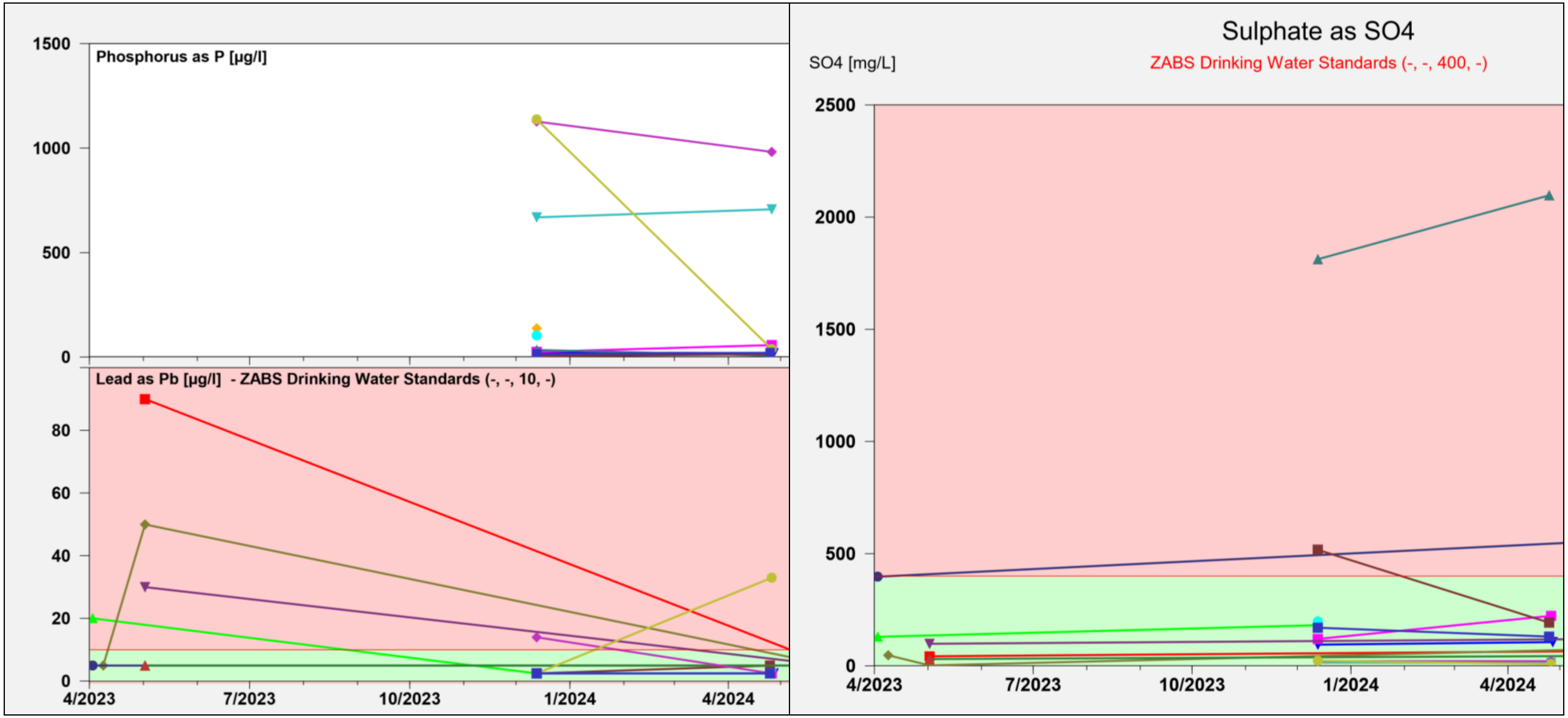
18.3.5.3. Radioactive parameters analysis

The following results are highlighted:

- Appreciable gross beta activity levels were measured in the groundwater across all the project deposits and are from the decay of uranium
- Uranium and its daughter product ^{226}Ra are at elevated levels, notably at Njame (^{234}U: 71.1 and ^{226}Ra: 1740 mBq/L), Gwabi (^{234}U: 211 and ^{226}Ra: 2490 mBq/L) and Dibbwi East (^{234}U: 234 and ^{226}Ra: 3270 mBq/L)
- The concentration levels for the other radioactive parameters, such as ^{232}Th, are relatively low across the entire project site.

The observed trend in radioactive levels mirrors that of certain physicochemical parameters, such as EC and total alkalinity, and some metal concentrations. This correlation suggests groundwater quality is due to the geochemical interaction with the orebody.


Electrical conductivity [mS/m] - ZABS Drinking Water Standards (-, -, 100, -)
pH - ZABS Drinking Water Standards (6.0, 6.0, 8.0, 8.0)
Iron (total) as Fe [µg/l] - ZABS Drinking Water Standards (-, -, 300, -)
Manganese as Mn [µg/l] - ZABS Drinking Water Standards (-, -, 200, -)
600
500
400
300
200
100
0
8.00
7.50
7.00
6.50
6.00
5.50
40000
30000
20000
10000
3000
2000
1000
4/2023
7/2023
10/2023
1/2024
4/2024



Note: The data points in regions highlighted in red fall outside the Allowable Guideline limits and green falls within the Allowable Guideline Limits.

Figure 18-77: Groundwater physicochemical, trace elements and SO_4 concentration levels across the Project site

18.3.6. Water balance

A water balance has been prepared to estimate the volume of excess water and the water requirements to sustain the plant. Groundwater supplies the entire water requirement for the mine. The water balance should be used as a tool to inform site planning and to optimise the water management on site. Activities carried out to complete the water balance included the following:

- Collection of data required to set up the water balance for the Gwabi, Njame, Muntanga and Dibbwi East and Dibbwi mine sites. The following data was obtained from GoviEx and other sources
  - Layout plans for the five mine sites
  - Rainfall data
  - Evaporation data
  - Proposed water usage
- Development of process flow diagrams at the four sites (Muntanga and Dibbwi East have been combined into one)
- Preparation of water balance for each site using Goldsim software.

18.3.6.1. Methodology

The steps taken to compile the water balance for the mine sites are as follows:

- Identification of each site water circuit and development of a schematic flow diagram
  - This task involved compiling flow diagrams for each site showing the water process flows
  - The purpose of the block flow diagram is to identify the water streams required to be modelled in each water balance
  - The water balance for each mine site was modelled in Goldsim software as a daily mass balance using flow estimates for each water stream
  - Constants used in similar mining applications were used where necessary in the model.

18.3.6.2. Data input into the Goldsim model

The following key steps summarise the construction of the water balance model at each site:

- The water balance components were constructed and linked according to the process flows in the block flow diagram
- The input data (rainfall, evaporation, proposed water usage) was collated and inserted into the model and
- The model was calibrated to simulate realistic flow rates for the proposed mine layout plan at each site.

18.3.6.3. Mine layout plans

The mine layout plan for the Muntanga and Dibbwi East mine sites indicating infrastructure considered in the water balance model is shown in Figure 16-1. The infrastructure considered includes the following:

- Pit mining area
- Plant
- WRD
- PCD
- Borehole excess water dams
- Pregnant pond
- Barren pond
- Rinse pond
- Spent ore stockpile
- Spent ore dam
- Stormwater
- Heap leach.

18.3.6.4. Rainfall and evaporation

The CHIRPS daily rainfall record for the period 1981 to 2021 was downloaded for Gwabi, Njame and Muntanga, Dibbwi East and Dibbwi sites, each record extended to 2022 with the daily rainfall data for the period 2020 to 2022 obtained from AMC to achieve a reasonable long rainfall record for the project. The monthly mean monthly rainfall (generated from the CHIRPS data and AMC 2020 to 2022, daily rainfall record) and evaporation used in the development of water balance for each mine site are plotted in Figure 20-4.

18.3.6.5. Proposed water usage

Data estimated based on the planned mine infrastructure at each mine was used to define the behaviour of the model and the boundaries defined. The boundaries for the four mine sites were defined according to where the internal streams leave the system and where the external streams enter the system. The boundaries include the following:

- Rainfall into various parts of the system
- Evaporation from various parts of the system
- Seepage from various parts of the system
- Potable water used in each system
- Groundwater entering the mine workings at various parts of the system
- Water discharge from the heap leach and stormwater systems
- Estimated sewage to be pumped from each sewage sump.

18.3.6.6. Groundwater flow and dewatering simulation

The inflow into the pits was simulated in the numerical groundwater model using MineDW. The predictive model used the pit depths to predict the estimated groundwater influx into the pits. Pit inflows for the five sites from the groundwater model were extracted into Microsoft Excel ("MS Excel") and input into the water balance model. The borehole dewatering at each pit was simulated in the numerical groundwater model and was extracted into MS Excel as input into the water balance model.

18.3.6.7. Water balance results

18.3.6.7.1. Muntanga and Dibbwi East

The potential total excess water from Muntanga and Dibbwi East mines including the water used at the plant are shown in Figure 18-78. This water results from ex-pit dewatering boreholes and groundwater inflow of both the Muntanga and the Dibbwi East pits. The excess water from the pits can be stored in the excess water dams for reuse or released into the environment depending on the water quality.

It is assumed that water from the excess water dams is re-used in the plant. The makeup water is then calculated based on the estimated plant water requirements less the water stored in the excess water dams and the contact fissure water from the pits. Due to borehole dewatering and pit inflows, no additional make-up water is required.

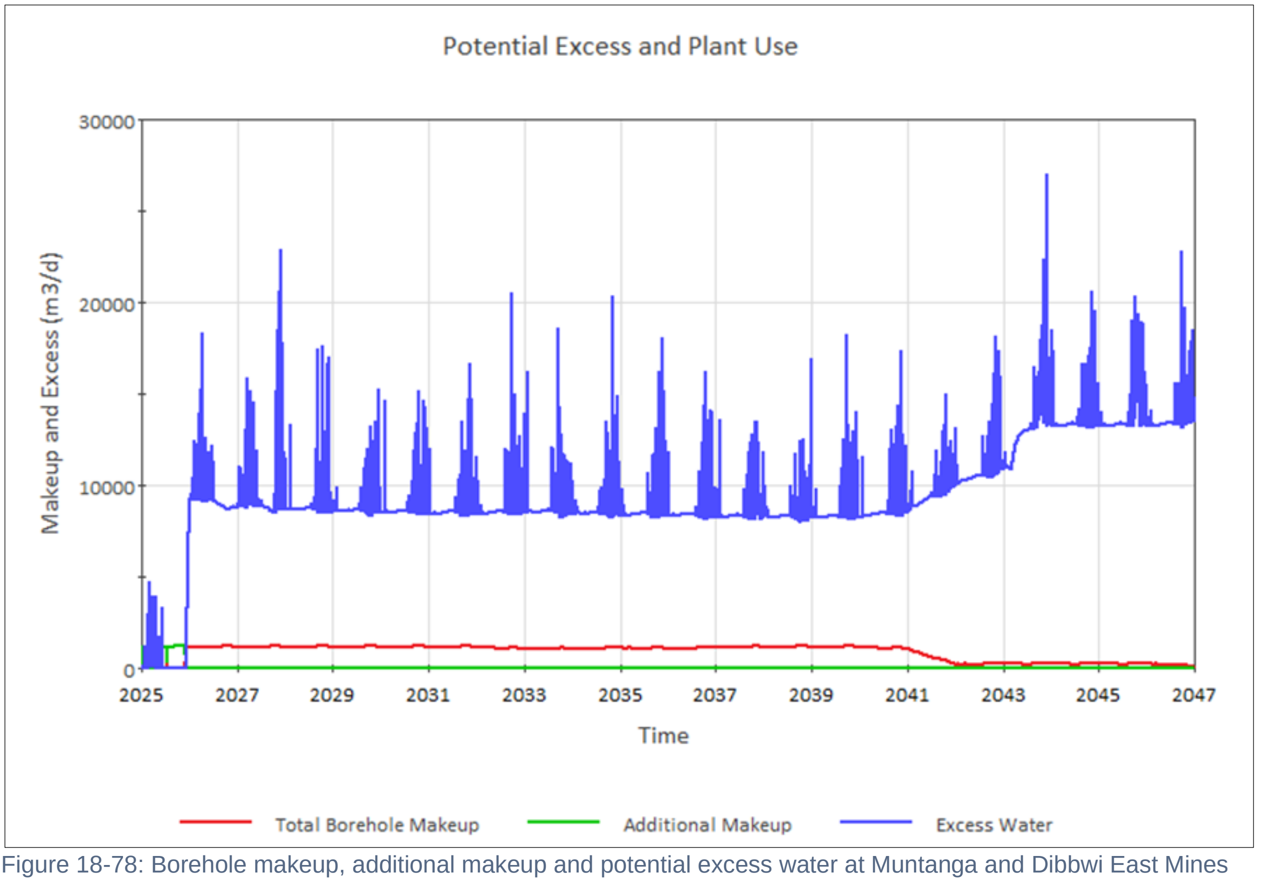


Figure 18-78: Borehole makeup, additional makeup and potential excess water at Muntanga and Dibbwi East Mines

18.3.6.8. Water balance conclusions and recommendations

The following can be concluded from the water balance study for the Gwabi, Njame, Muntanga and Dibbwi East and Dibbwi mines:

- No additional makeup water is required and
- Excess water can be used at the plant as makeup water or released to the environment depending on quality.

The following is recommended:

- Continual monitoring of the inflow volumes into the pit
- Monitoring of the dewatering volumes
- Metre water used at the plant
- Consider the potential use of excess water for additional dust suppression.

18.3.7.Stormwater water discharge management

Several non-contact (clean) surface water infrastructures were assessed as part of the design for each site. From this, the following was decided:

- Clean surface water diversion channels will divert clean water around mining infrastructure, where necessary
- Clean water storage ponds around the mining pit of each site will intercept clean surface water flowing towards the pit and provide a means of water storage, which will then be purposed for other uses within each site
- Upstream attenuation dams will be required for the Gwabi site to allow for the collection and storage of surface water originating upstream of the site, which would be available to the surrounding communities if deemed safe
- Attenuation and excess water dams will provide a means of water collection and storage, from where water is required for operational purposes is pumped.

A summary of the infrastructure considered as part of the design for each site is listed in the sub-sections below:

- Muntanga and Dibbwi East
  - Grassed, clean surface water diversion channels (1 m to 1.5 m bottom widths and 0.5 m to 1.5 m depths)
  - One clay-lined, clean water storage pond (capacity of 2 800 m^3)
  - One attenuation dam (capacity of 85 000 m^3)
  - Two excess water dams (capacities of 7 400 m^3 and 24 000 m^3)
  - One diversion berm (length of 23 m and height of 5 m)
  - 227 WRDs paddocks (50 m x 25 m x 1 m depth per paddock).

Figure 18-79 illustrates the general surface water infrastructure arrangement for each Muntanga and Dibbwi East.

18.3.8. Water supply and ex-pit dewatering scheme

Dewatering of pits from boreholes and associated piping required to pump the water to the excess water dams of each site was designed. The extracted water will meet the operational requirements however if the boreholes are not able to supply sufficient water for operational activities, additional water will be pumped from the excess water dams. Potable needs will be directly from the dewatering boreholes. Submersible pump configurations to pump water from each borehole to the excess water dam are also included as part of the infrastructure installations. A summary of the infrastructure considered as part of the design for each site is listed in the sub-sections below:

- Muntanga and Dibbwi East
  - 11 boreholes (total drilling depth of 1 540 m with borehole depths ranging between 100 m to 150 m)
  - The following HDPE pipes (total length of 12 570 m), consisting of the following pipe diameters and associated lengths
    - 50mm OD Class 10: 1 045 m
    - 110mm OD Class 10: 2 508 m (inclusive of 1 068m potable water pipeline)
    - 125mm OD Class 10: 963 m
    - 160mm OD Class 10: 1 106 m
    - 200mm OD Class 12: 3 003 m
    - 250mm OD Class 12: 333 m
    - 315mm OD Class 12: 513 m
    - 90mm OD Class 10: 1 070 m
    - 110mm OD Class 10: 640 m (potable water pipeline)
    - 125mm OD Class 10: 1 385 m

Figure 18-79 illustrates the general surface water infrastructure arrangement, and the borehole location and associated piping arrangement for Muntanga and Dibbwi East

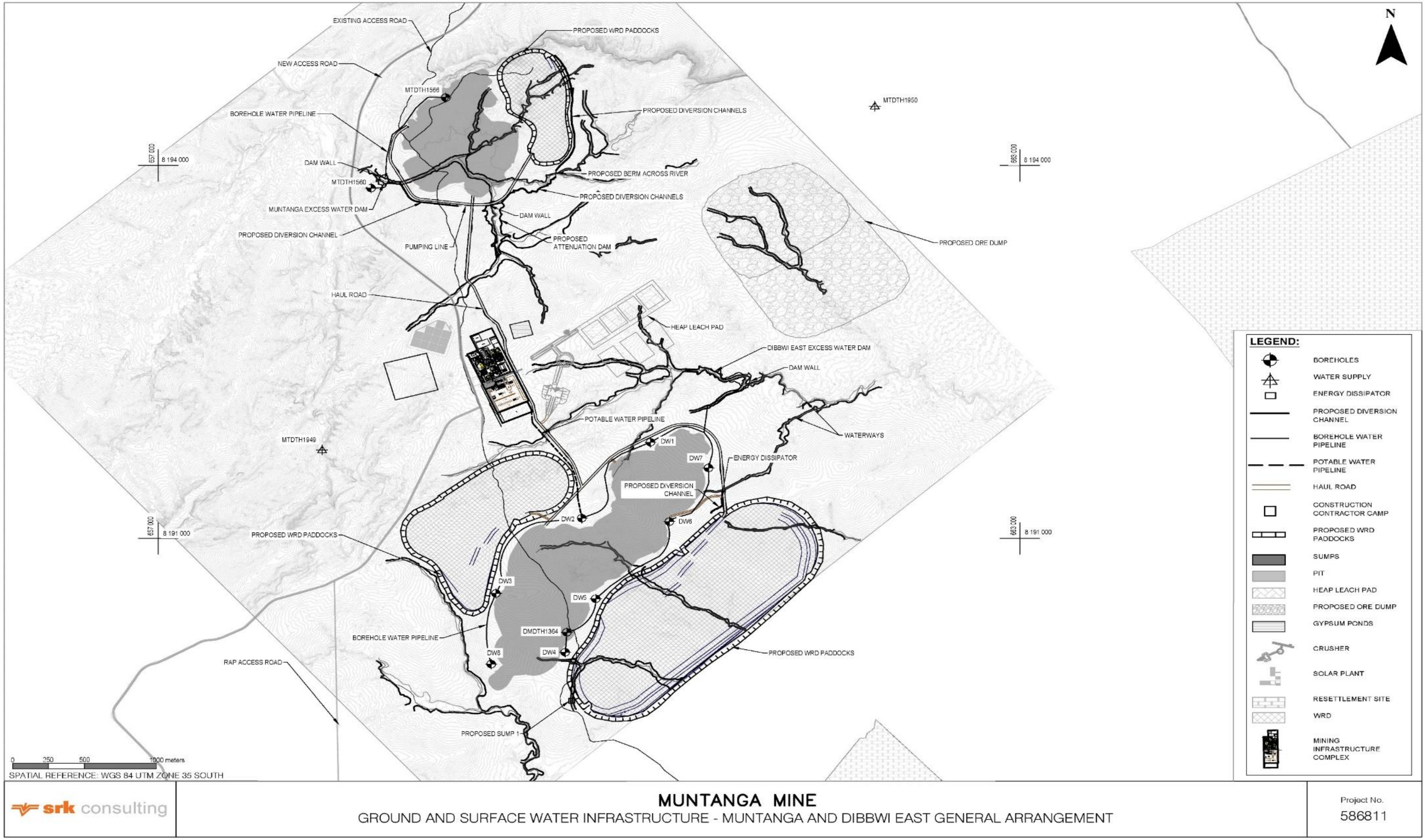


Figure 18-79: Ground and surface water infrastructure - Muntanga and Dibbwi East general arrangement layout

18.3.9.Water monitoring

Given the potential contamination risks from radioactive and heavy metals polluting the water resources and the need to reduce contact water so that excess water can be safely discharged, a comprehensive water management plan is recommended. The water management plan will ensure that a duty of care is taken, and dewatering and stormwater management are effective. It minimises groundwater contamination and impacts from mining activities and helps maintain the baseline conditions of the aquifer.

The water management plan will include monitoring water quality and quantity by routine recording and analysis of water discharges, water levels and quality.

18.3.9.1. Dewatering monitoring

Besides environmental compliance groundwater monitoring points, standpipe piezometers and vibrating wire piezometers are recommended at high-risk slopes around the pit. SRK recommends that automatic continuous data loggers be installed in these monitoring boreholes.

- Groundwater levels are to be measured at pumping and non-pumping boreholes to evaluate the amount of drawdown caused by the pit dewatering
- Daily pumping rates from dewatering boreholes must be recorded. Sump discharge should be recorded. Water samples will be collected for chemical analysis at least bi-annually
- Based on the baseline groundwater quality, parameters that exceed the guideline limits will have to be monitored in addition to standard monitoring parameters as required by the relevant environmental and mining permits
- The monitoring data collected must be used to determine the effectiveness of the dewatering system and to identify the build-up of undesirable pore pressure and aquifer response to recharge events.

18.3.9.2. Groundwater level monitoring

To assess the effectiveness and impacts of dewatering, the water levels of the aquifers around the mining areas will be routinely monitored. Additionally, high groundwater levels increase pore pressure, reducing slope stability and posing a risk of pit wall or tailings failure.

18.3.9.3. Monitoring database

A sound database management system is proposed, which not only stores the data but enables interpretation and monitoring of the effectiveness of dewatering and compliance with environmental regulations. The purpose of the database management system should be to record all monitoring and related information, such that complete, accurate records can be extracted easily and quickly and requiring minimal processing.

The database records stored should include the following:

- Climatic information e.g. daily rainfall, evaporation etc.
- List of monitoring point names with co-ordinates, including elevation
- Complete maps showing the location of the monitoring points – these might be in a number of formats i.e. pictures (jpeg files), GIS (shp files) and CAD (dwg files)
- Logs and borehole construction details
- Borehole pumping records, including rates, details of pump decommissioning etc. An annotated graphical depiction of this will aid future interpretation and numerical modelling
- Water level and pressure head data – values converted to mamsl and annotated graphs to display temporal changes in water levels or head for each point
- Inflow rates – accurate measurement of all major inflows and graphical depictions of these
- Sump discharge rates - graphical depictions of these with annotation
- Chemical and radiogenic data - QAQC must be carried out on data and statistics determined per analyte, such as the average, standard deviation and 90th percentile.

18.3.9.4. Stormwater

All channels, berms, and containment facilities will be inspected and serviced regularly to ensure the design capacity and integrity are maintained. Stormwater control measures will be kept clear of debris and silt.

18.3.9.5. Water balance

The mine should maintain and update a comprehensive water balance that includes inflows and outflows of groundwater for the five mining sites.

18.3.10. Summary

Flood mitigation measures such as river diversions and flow attenuation dams will be implemented along the rivers to prevent flooding of the Muntanga mine infrastructure including access and haul roads impacted during the 1:50-year and 1:100-year flood events.

Stormwater from unimpacted catchments upstream of mining will be diverted to drainage lines using stormwater channels, berms, and culverts to prevent mixing with the contact water. Stormwater from potentially contaminated areas (the mine pits, the HLPs, and the plant areas) will be collected in lined channels and discharged into proposed PCDs. It is recommended that paddocks around the WRDs be constructed to capture side slope runoff. All stormwater measures are designed for 1:100-year storm events. The stormwater infrastructure will include:

- Grassed, non-contact (clean) surface water diversion channels
- Four x clay-lined, clean water storage pond with a total capacity 21 400 m^3
- Five x excess water dam with a total capacity of 67 400 m^3
- Five x attenuation dams with capacities ranging between 14 000 m^3 and 85 000 m^3
- 317 WRD paddocks
- One diversion berm (length of 23 m and height of 5 m).

Surface water resources close to the Central, Dibbwi, and Njame pits are limited as most streams and rivers are ephemeral such as the Lusithu River. The Zambezi River and Lake Kariba were discarded as water supply options due to the distance from mines and the likely onerous regulatory approval process. Kafue River will be a reliable water resource, as it has high flows and therefore a high assurance of supply, specifically to supply water to the nearby operations at Gwabi.

Passive residual groundwater inflow into the pits will be kept ideally within 1 000 m^3/d, so that in-pit sumps and contact water volumes are kept to a minimum. Ex-pit dewatering boreholes will be strategically drilled in addition to the already drilled boreholes around the mining pit to keep the working area dry and safe (Table 18-25).

Table 18-25: Dewatering boreholes

Project area	Total number of dewatering boreholes	Passive residual pit inflows [m^3/d]
Muntanga	2	650
Dibbwi East	9	2 400

The groundwater analysis and water balance (both detailed in Part II of the Water Management Report) have shown that there will be sufficient groundwater on-site from the dewatering process to meet the mining requirements, without the need to bring water in from external sources to the sites. The water balance has considered:

- Rainfall into various parts of the system
- Evaporation from various parts of the system
- Seepage from various parts of the system
- Potable water use in each system
- Groundwater entering the mine workings at various parts of the system
- Water discharge from the heap leach and stormwater systems
- Estimated sewage to be pumped from each sewage sump.

Excess water will be generated and will be released to the environment depending on quality. Muntanga has overall superior groundwater quality in comparison to the other deposits, with an EC of less than 347 µS/cm. The concentration levels of most of the analysed parameters (including total alkalinity, dissolved calcium, and sodium) exhibit distinct spatial patterns across the project area. Notably, boreholes at Muntanga, Dibbwi, and Njame consistently exhibit lower concentration levels, with a gradual increase observed sequentially from Muntanga to Dibbwi, to Dibbwi East and Njame. With respect to radiogenic activity, appreciable gross beta activity levels were measured in the groundwater across all the project deposits and are from the decay of uranium.

The water management plan will include monitoring water quality and quantity by routine recording and analysis of climatic conditions, discharge and flows, water levels and quality.

18.4. Bulk power supply

18.4.1. Introduction

Muntanga and the satellite pits will be connected to the Zambian National Grid. The Zambia Electricity Supply Corporation Limited (“ZESCO”) will deliver power. Muntanga will connect via a new, dedicated connection to the Siavonga 330 kV/132 kV/33 kV substation, which is adjacent to the Kariba Dam. At Muntanga there is an option to install a solar plant and battery energy storage system in the future for the mine's needs; however, this is currently not in the FS base case.

For the FS, GoviEx and SRK utilised the services of UTILINK Limited (herein referred to as "Utilink"), an engineering, project management and energy advisory consultancy in Zambia, to form an "FS Project Team" (comprising SRK, GoviEx and Utilink) who engaged directly with ZESCO to discuss and agree the form and point of the connection with grid and is currently negotiating a Connection Agreement ("CA") and Power Purchase Agreement ("PPA"). The study and design of a renewable plant were supported by MCD-Energy ("MCD"), an international renewable energy consultancy company.

18.4.2.Bulk power supply strategy overview

The bulk power supply strategy and required infrastructure have been developed with the engagement and advice of ZESCO and the project is summarised below and illustrated in Table 18-26.

- At Muntanga (Central), construction of a 39 km, 132 kV single circuit OHL and Muntanga 132 kV/35 kV/11 kV substation connecting to the Siavonga substation
- Construction of a 11 kV switchgear supplied by the grid feeding the site's electrical distribution system
- GoviEx has yet to determine if a solar plant will be constructed prior to first production; however, a 9 MWp solar plant and 2 MWh battery energy storage system (to manage solar-grid integration) was studied and cost and remains an option which, if executed, would reduce the average tariff by 20 % and payback is expected in around five to six years at current grid and technology pricing
- Diesel generator plant ("DG") is constrained to support critical loads only.

The procurement strategy is shown in Table 18-26.

Table 18-26:Bulk power procurement strategy

Package	Details	Units
1	Grid Connections (all)	EPC / Transfer to ZESCO
3	Solar PV and Battery Energy Storage System	EPCM / Owner Operate
4	6.6 kV Switchgear	EPCM / Owner Operate

18.4.3.Zambia Electricity Supply Corporation Limited and the Zambian National grid

ZESCO is the Zambian national power utility owned by the Government of the Republic of Zambia. ZESCO's mandate is to supply electricity and energy solutions within Zambia and the Sub-Saharan region. Their operations include generation, transmission, distribution, and supply and they own and operate a number of hydropower stations with a combined power generating capacity of more than 2 900 MW. ZESCO has very recently approved and encouraged the connection of third-party-owned solar generation capacity on the grid with an option to "net metre" energy to the grid. They supply power to mining operations under a PPA, which, amongst other things, sets the fixed and variable components of the tariff and provides preferential customer status for security and quality of supply. During the FS, GoviEx and the FS technical team engaged with ZESCO's Transmission, Finance, and Planning and Projects Departments.

18.4.3.1. Zambian National Grid

Around 80 % of Zambia's electricity is generated from hydropower, with other sources being thermal. The Government of Zambia is diversifying generation sources with Zambia having immense potential for power generation through solar, wind, geothermal, and further hydropower development. Zambia is a member of the Southern African Power Pool ("SAPP") with interconnecting transmission routes with Congo, Tanzania, and South Africa. In normal years, Zambia is a net exporter of power to the SAPP; however, the dependency on hydropower represents a risk to the security of supply as evidenced by the country-wide load-shedding seen in 2024 due to the impact of lower-than-predicted reservoir levels in early 2024. As a result, and which is well documented, Zambia has initiated a drive to diversify supply sources (solar, thermal, wind) and strengthen the regional interconnectors to ensure sufficient reserve power can be procured via SAPP.

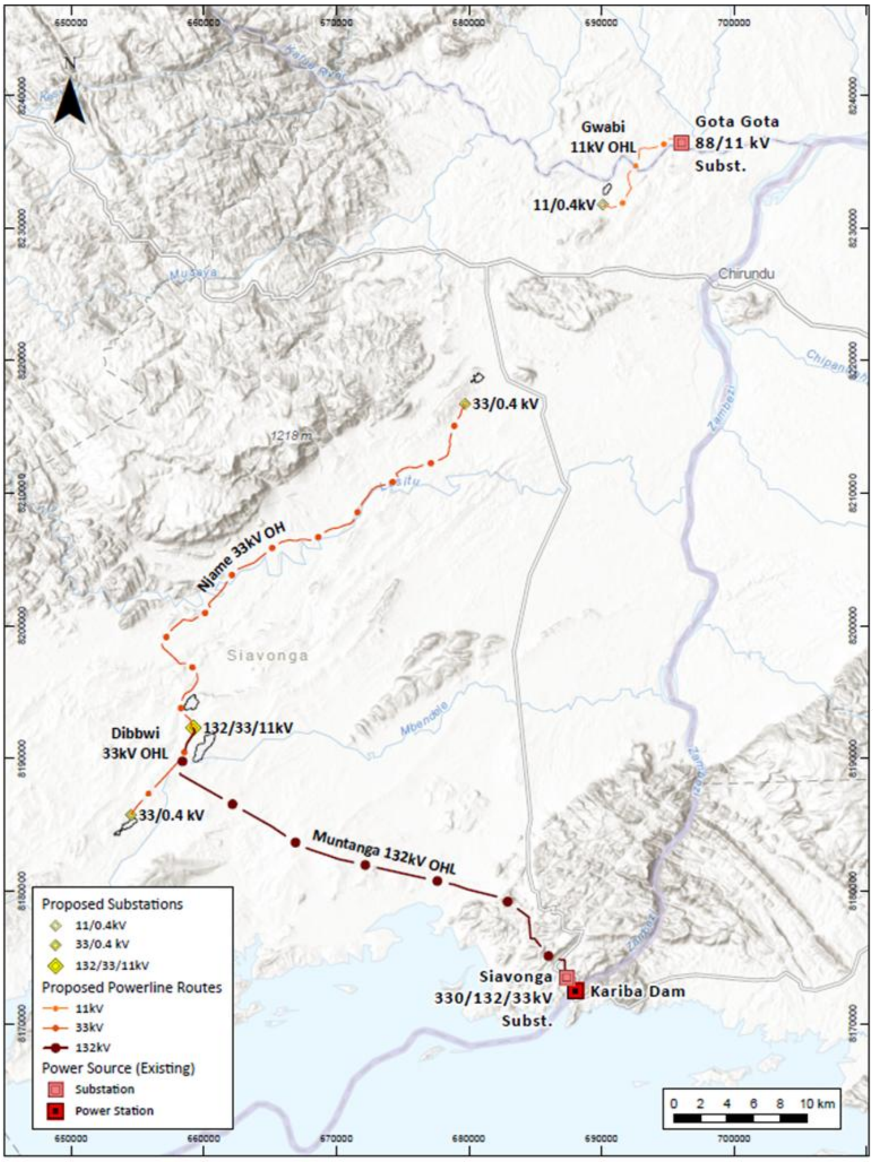


Figure 18-80: Layout of proposed power connections

18.4.4.Zambia Electricity Supply Corporation Limited agreements and documentation

18.4.4.1. Meetings

The FS project team held a number of key meetings with ZESCO representatives from the planning and projects department, transmission department, and the Chirundu and Siavonga regions including:

- November 2022 (general discussion)
- October 2023 and November 2023 (connection options and grid connection study)
- June 2024 (update and connection agreement) and
- October 2024 (update and connection agreement).

18.4.4.2. Grid connection (impact) study

Further to discussions, ZESCO formally advised GoviEx of the ZESCO proposal for connecting the project. This was undertaken by ZESCO in the form of a grid connection (impact) study ("GCS"). Within the GCS, ZESCO studies a number of options for connections at different voltage levels to understand and confirm available power and suitability.

18.4.4.3. Non-disclosure agreement

GoviEx and ZESCO signed a non-disclosure agreement ("NDA") in Q1 2024.

18.4.4.4. ZESCO site visit and grid connection study discussion

Together with ZESCO representatives, GoviEx and Utilink visited the proposed sites of connection at Gota-Gota and Siavonga and discussed requirements with ZESCO to inform the FS design. At this time, Utilink discussed the outcomes of the GCS with ZESCO and together, ZESCO and the FS project team agreed on an overall strategy. In summary:

- The proposed strategy for supply Muntanga and Dibbwi was agreed as per the ZESCO proposal (a 132 kV connection to Siavonga, a 132/33/11 kV substation at Muntanga and a 33kV connection to Dibbwi)
- The initial proposal to connect Gwabi via a 33 kV line from the existing and operational Gota-Gota 88/11 kV substation, requiring an extensive upgrade to the substation, was discussed and superseded by an alternative proposal for an 11kV connection to the Gota-Gota substation requiring a reduced level of upgrades to facilitate such a connection
- The initial proposal to connect Njame via an 11 kV transmission line to the 33 kV substation at Gwabi was discussed and superseded by an alternative proposal to connect Njame to the proposed Muntanga 132/33/11 kV substation.

18.4.4.5. Connection agreement

A draft connection agreement has been provided by ZESCO and is being reviewed by GoviEx. The connection agreement does not mandate GoviEx to construct the power supply infrastructure. The connection agreement formalises the following:

- Formally agree on the point and form of connection
- Confirms the capacity to be made available by ZESCO at the point of connection and
- Confirms the approach to project implementation.

18.4.4.6. Power purchase agreement

GoviEx has requested a draft of a PPA from ZESCO and is awaiting a formal response.

18.4.5.Strategic studies

To bridge the PEA, various strategic studies were conducted during the development of the FS. During the study key design basis inputs, such as the load list, were continuously updated with the final inputs used for FS design (as presented in Section 18.4.5.4).

During the Phase 1 strategic study, multiple supply options were assessed (and discussions commenced with ZESCO) about a possible grid connection. ZESCO undertook a grid impact study and advised on proposed connection points, which were then reviewed by Utilink, and counterproposals made and agreed where necessary. Later, during Phase 2, the potential for a solar plant at Muntanga was reassessed. The viability of a relocatable diesel-solar-battery plant to supply power to the satellite projects was also revisited.

18.4.5.1. SRK site visit

The following observations and confirmations were made in relation to Muntanga and Dibbwi:

- No power lines exist in the Muntanga and Dibbwi areas
- There was an understanding that ZESCO had previously considered developing a transmission line into the area from Siavonga to support local development as far as Kanyama
- The obvious potential voltage levels are 33 kV and 132 kV. Other voltages would require additional voltage transformation.

The following observations and confirmations were made in relation to Njame:

- An 11 kV line is being constructed along the D500; however, it is understood that capacity is limited, and the line is being used to supply local schools and villages
- There is no potential for a local connection to the 330 kV lines that constrain Njame to the east (see Figure 18-81).

Figure 18-81: 330 kV transmission crossing the D500 just east of the Njame area

The small solar plant (10 x 545 Wp panels) at the Muntanga camp was observed and it was hoped to provide proxy information for a future solar farm to compare to global irradiance databases (Figure 18-84).

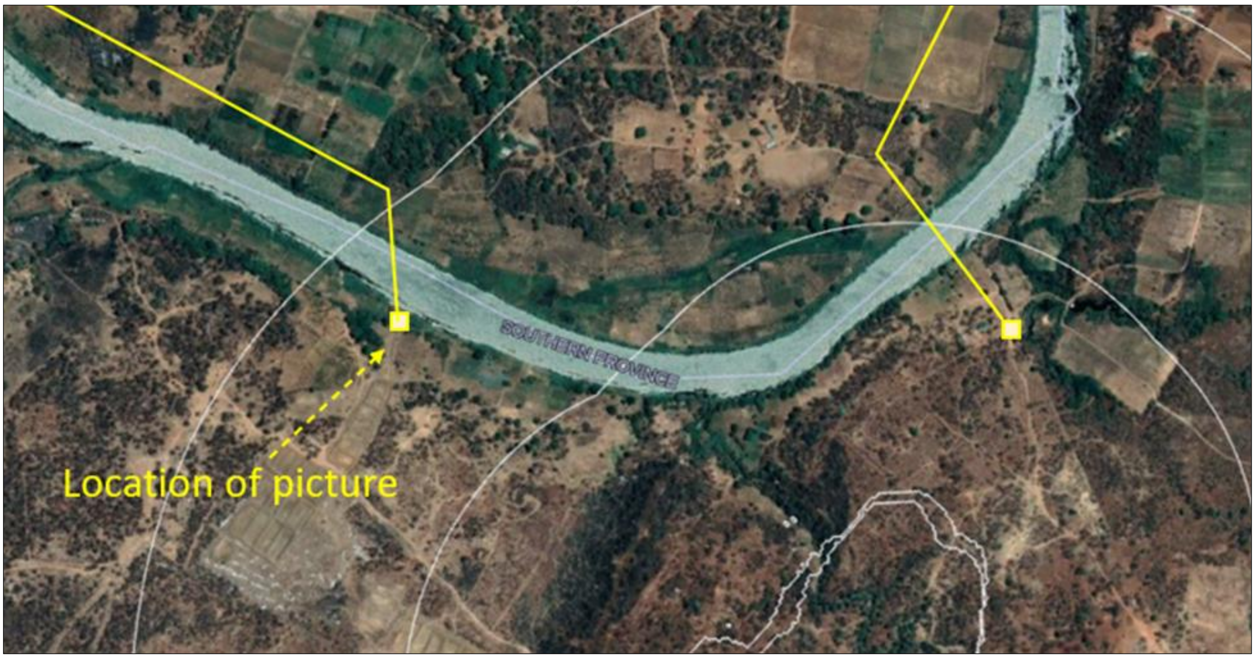


Figure 18-82: Location of 11 kV line terminus at Palabana fish farm

Figure 18-83: 11 kV line terminus at Palabana fish farm

Figure 18-84: GoviEx Muntanga camp ground-mounted solar panels

18.4.5.2. Phase 1 strategic study: Muntanga grid connection

The Phase 1 study assessed multiple options for supplying the project sites with power and was undertaken at a conceptual level of study with Capex and Opex costs estimated at a Class 5 (scoping/ concept) level of estimate. From a number of potential options for Muntanga, the following were analysed at a tariff level of 10 USD c/kWh and 7 USD c/KWh assuming the grid tariff is similar for all transmission voltage levels:

- Grid connection at 33 kV only, no renewables, 100 % grid
- Grid connection at 132 kV only, no renewables, 100 % grid
- Grid connection at 33 kV, 5 MWp Solar FT, 100 % grid
- Grid connection at 33 kV, 5 MWp Solar SAT, 100 % grid
- Grid connection at 33 kV, 5 MWp Solar FT, 2MWh Battery Energy Storage System (“BESS”), 100 % grid
- Grid connection at 33 kV, 10 MWp Solar FT, 2MWh BESS, 100 % grid
- Grid connection at 33 kV, 5 MWp Solar SAT, 100 % grid, Net metering
- Off-grid Diesel, 5 MWp Solar, 2 MWh BESS, 100 % grid
- Grid, Diesel, 5 MWp Solar, 2 MWh BESS, 70 % grid
- Grid connection at 132 kV, 5 MWp Solar SAT, 100 % grid.

The tariff levels chosen for the assessment were drawn from a mixture of previously published ZESCO tariff rates and benchmarks from negotiated PPAs with ZESCO or Copperbelt Energy Corporation (“CEC”). These indicated the tariff is likely to be in the range of 7 USD c/kWh to 10 USD c/kWh (at pre-2023 exchange rates). For the purposes of this analysis, a placeholder tariff of 10 USD c/kWh was agreed with a lower 7 USD c/kWh as a comparison.

The tariff only gets confirmed once the CA is agreed and PPA negotiated. Although ZESCO publishes rates for users <5 maximum average demand (“MVA”), all mining projects must negotiate a PPA. For user with a demand of >5 MVA, previously this attracted a “bulk demand” tariff; however, more recently ZESCO made it a requirement for those >5 MVA to negotiate a PPA. During a litreature search, SRK found some previously published rates for >5 MVA which indicated a 2024 tariff of approximately 8 USD c/kWh at a current exchange rate.

Capital costs were estimated and covered supply and installation and were considered to be scoping level with an accuracy of up to +50/-25 %. Cost inputs for operating/ reinvestment costs for renewable energy plants as presented in Table 18-27.

Table 18-27: Estimated operating/ reinvestment costs

Details	Total capital cost [USDM]	
Solar maintenance cost	3	USD/kWp/annum
Battery maintenance cost	5	USD/kWp/annum
Solar reinvestment cost	20 %	panels per annum
Batteries	None	Life of project only 10 years

For the analysis, an assumption was made that the connection agreement and PPA ensure the grid supply from ZESCO is stable and of sufficient quality to ensure any outages do not adversely impact planned plant operating hours (i.e., grid availability is assumed to be >98 %). Therefore, back generation is only required for critical loads and no battery energy storage system for off-grid plant switching is required. The results are presented in Figure 18-85 and Figure 18-86.

Table 18-28: Inputs for Phase 1 study (superseded by later FS design criterion)

Item	Units	Value	Source / Notes
LOM	Year	16	Project assumption
Muntanga:			
Continuous demand*	kW	4 950	Project assumption
Consumption	GWh	43	Project assumption
Satellites:			
Continuous demand*	kW	1 150	Project assumption
Consumption	GWh	10	Project assumption
General:			
Grid availability	%	>98	ZESCO
Diesel (delivered to site)	USD/L	1.3	GoviEx
Litres per kWh for DG	L/kWh	0.27	Industry benchmark
Solar irradiation (Peak)	kWh/kWp/y	1 750	Solar PV simulation software

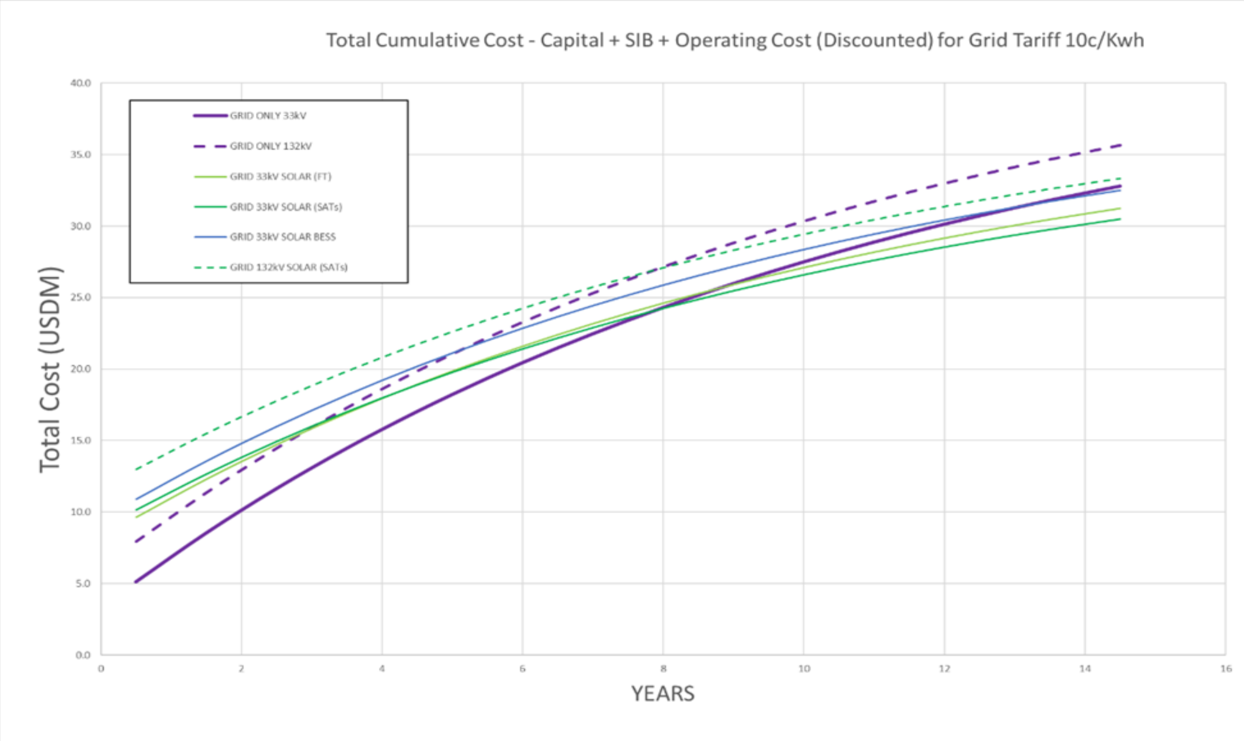


Figure 18-85: Total cumulative cost (discounted) for selected options comparing grid + solar options at 10 USD c/kWh

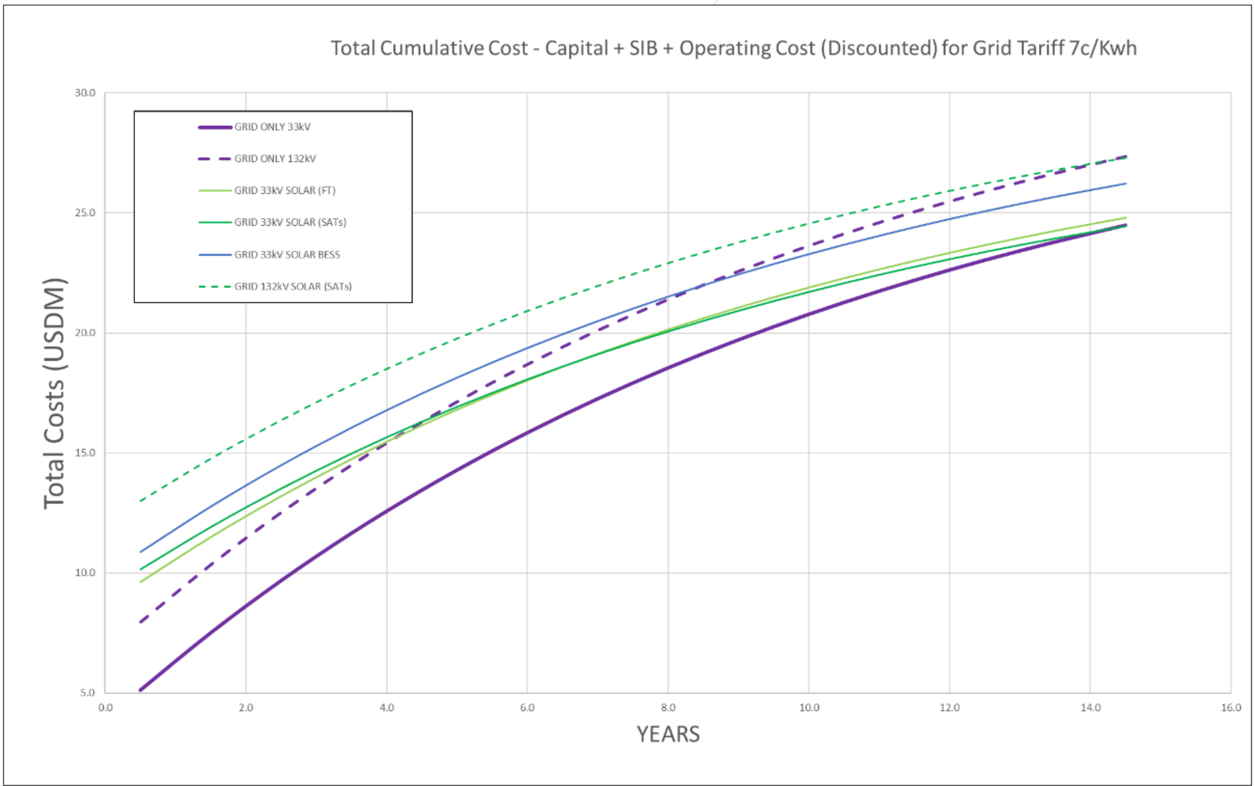


Figure 18-86: Total cumulative cost (discounted) for selected options comparing grid + solar options at 7 USD c/kWh

Figure 18-85 shows the "Total cumulative cost (discounted) for selected options comparing grid + solar options at 10 USD c/kWh" focusing on selected grid and solar options. The key conclusion shown in Figure 18-85 is that for either transmission voltage (33 kV or 132 kV) when assuming the same tariff of 10 USD c/kWh, a 5 MWp SAT solar plant pays back around year 8. Figure 18-86, shows "Total cumulative cost (discounted) for selected options comparing Grid + Solar options at 7 USD c/kWh". This shows that when the tariff is reduced, the annual savings are reduced, and it takes longer for the solar plant to pay for itself. On a discounted basis, the payback is around year 14.

From the Phase 1 study, the following conclusions and recommendations were drawn:

- A grid connection providing a stable power supply is the optimal solution for the project
- Power from an off-grid solution will be very expensive relative to a grid connection
- Either voltage level (33 kV and 132 kV) is preferred when compared to off-grid
- The impact of solar plants will largely depend on the eventual agreed tariff. The "breakeven" point is around 7 USD c/kWh with progressively faster payback seen as the tariff increases
- The USD tariff price will depend on the ZESCO negotiations, the Energy Regulation Board ("ERB") advice, and the ZMW/ USD exchange rate. The grid connection voltage may also have some impact, but it is currently impossible to predict this
- Obtaining budget pricing for a 5 MWp solar plant may be prudent to obtain values for the FS if required.

Connection options for the satellite plants were also reviewed. At the time of the study, the satellite plants were to be exploited in sequence and concurrently with Muntanga. Each satellite plant had a LOM of roughly five to six years.

The main options assessed were:

- Grid connection
- Relocatable off-grid solution and
- Grid connection with the relocatable solar plant.

The following conclusions and recommendations were drawn:

- A grid connection represents the lowest-cost option
- An off-grid thermal hybrid plant would be extremely expensive. The impact of a reduced grid availability changes this quite significantly, therefore wider grid availability needs to be monitored (if required in the long run, the inclusion of diesel engines to provide 100 % backup would be a simple and relatively low-cost addition to the project)
- If a re-deployable solar plant is constructed at Dibbwi and relocated to Njame and Gwabi, the analysis shows this is not cost-effective at a grid price of lower than 11 USD c/kWh.

18.4.5.3. Grid impact study ZESCO and Utilink review

The ZESCO grid impact study was presented to GoviEx via an online meeting and the summary of the ZESCO proposed grid connection strategy for GoviEx was provided in a letter format. The study itself was not provided.

For Muntanga, ZESCO assessed a connection at 132 kV and 33 kV. It was established the existing 33 kV grid was unable to support the proposed development, and the connection would need to be at the 132 kV transmission voltage. An intermediate transformation to 66 kV was considered suboptimal and likely to result in a similar level of Capex.

18.4.5.4. Muntanga solar plant

As recommended during the Phase 1 power strategic study, a solar plant has been further considered during the FS and a final sizing and trade-off was undertaken once the FS load list was defined for Muntanga (see Section 18.4.6). By centralising the processing plant at Muntanga and trucking ore from the satellites to the processing plant, the agreed peak energy demand is approximately 6.3 MW. Assuming a 24/7/365 operation, this would mean an energy requirement of around 55.2 GWh per year.

The proposed solar plant would be located close to the mine site and the main incoming overhead line ("OHL") substation from Siavonga to the southwest of the Muntanga Road (see Figure 18-87) and will feed in behind the metre at the 11 kV level.

At approximately 16º south of the Equator with an estimated solar irradiation of 1 770 kWh/kWp/year to 1 785 kWh/kWp/year (GSA, Worldbank Global Solar Atlas), a 9 MWp solar plant facing north, north-east would generate approximately 16 GWh per year (28 % renewable penetration of the total energy demand).

Figure 18-87: Proposed Muntanga 9MW solar plant layout and location

The ground-mounted solar power plant consists of locally sourced concrete bases (Zambia) or steel frames (South Africa) installed and fixed in the ground in a northerly facing aspect with the exact location and angle to be determined in the detailed engineering design. Energy is generated using monocrystalline solar PV panels (575 Wp or higher) with 36, 250 kW decentralised string inverters. The interconnecting cables between the solar panels and the DC/AC inverters and from the inverters to a combiner panel are to be provided. A connection from the solar AC combiner panel to the mine MCC panel in the switch room is in place. Ideally, the mine MCC panel will consist of fully motorised breakers feeding the individual loads in the processing plant. The entire solar power plant including switchgear will be surrounded by an appropriate fence including service access gates with security lighting and CCTV, as required.

A 2 MWh BESS is proposed for the site (Figure 18-88). The 2 MWh BESS is included to manage the transitions from solar to grid and vice versa but can also mitigate any fluctuations in grid supply. The BESS is a self-contained unit, typically housed in a 20’ ISO container, a DC combiner panel, battery inverter units (PCS), an AC combiner panel, and a step-down/step-up transformer.

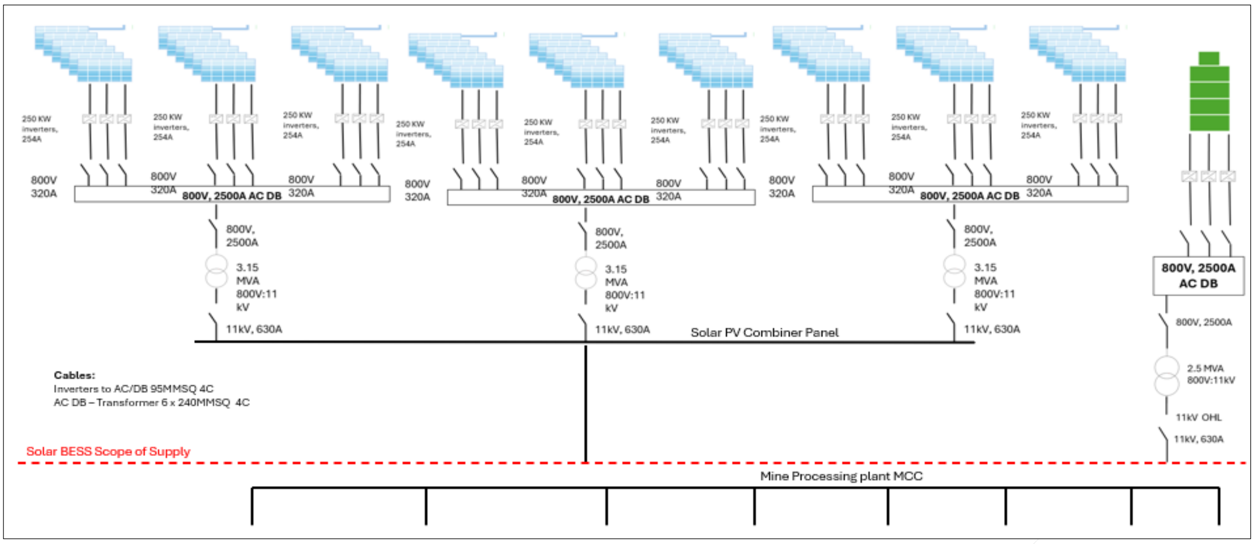


Figure 18-88: Preliminary single line for 9MWp solar and 2MWh BESS

The expected Capex cost for the 9 MWp solar plant and BESS at current market rates, is USD8.0 million. The expected savings generated by the solar plant are calculated at USD1.4M per annum, giving an estimated payback on the renewable energy plant of around 5.5 years. The solar power plant has an expected life of 25 years and the BESS ten years to 15 years, so the reinvestment costs over the LOM are limited to smaller electrical components, such as inverters and possibly some panels.

With a grid connection at the mine, the estimated cost/ kWh including the Capex cost of construction of the OHL and based on a LOM of 20 years, the LCOE is USD0.114 /kWh.

To reduce costs and control the technical aspects of the project closely, an EPCM approach to this project was assumed with GoviEx procuring the major components directly, managing the EPC contractor for construction and a LSTK delivery of the BESS.

Costs for GoviEx-supplied equipment are based on 2023/24 market rates. Materials purchase preference would be:

- Frames, mounting structures and cables: Zambia, SADC or East Africa
- Panels and inverters: China
- Electrical switchgear: sourced either locally, in South Africa or in India.

A preliminary commercial offer was received for installation works required for the solar power plant (a quotation for installation of 6 MW was sought and adjusted for 9 MW and 10 MW scenarios).

18.4.6.Feasibility study power demand and project load profile

18.4.6.1. Power demand

The electrical demand developed throughout the study with a number of iterations and updates from November 2023 to August 2024. A summary of the finalised electrical load list for the Muntanga site is presented in Table 18-29.

Table 18-29: Muntanga (central) area finalised electrical load list

Item	Nominal average [running]	Nominal average [running]	Maximum average demand	Maximum demand
	kW	kVA	kW	kVA
Processing plant	5 915	6 767	7 394	8 469
MMA	209	209	209	209
Mine camp	60	60	60	60
Dewatering	173	203	173	203
Water supply	23	27	23	27
Other	638	727	786	897
Total	**7 018**	**7 993**	**8 645**	**9 865**

(Source: M7610-E831-001 - Electrical Load List- Rev C.pdf (SGS, 2024), 31372 - FS Electrical Load List_Consolidated_v2)

18.4.6.2. Load profile

The overall project load profile of for the Project is presented in Figure 18-89 .

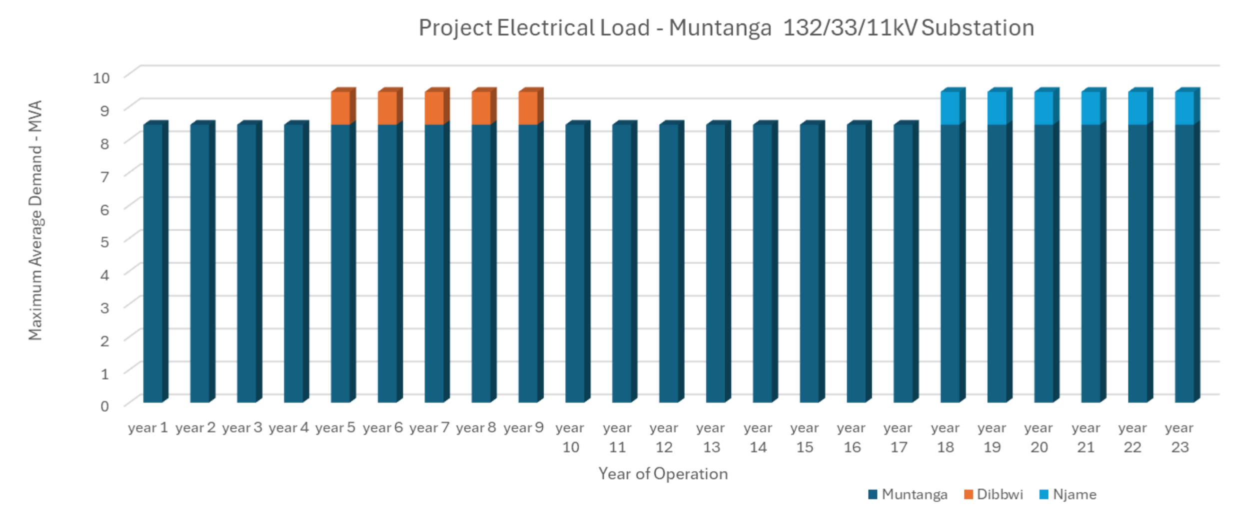


Figure 18-89: Muntanga and connected satellite projects overall project load profile MVA

18.4.6.3. Supply strategy/ Distribution voltage

The grid connection study and subsequent discussions determined the design basis for the form of the power supply infrastructure.

- Muntanga supply via 132 kV connection to Siavonga
- Site distribution at 11 kV as requested by the processing plant engineers.

18.4.7.Feasibility study power supply infrastructure

18.4.7.1. Muntanga power supply

The details of the power supply scheme are as follows with the single line diagrams presented in Figure 18-90 and Figure 18-91:

- Install a 132 kV line switch bay at the existing Siavonga 330/132/33 kV substation complete with line protection, metering and SCADA
- Construct a 39 km 132 kV single circuit steel monopole line from Siavonga 330/132 kV substation to Muntanga using 150 mm^2 ACSR conductor (ZESCO preference) and line post insulators. The 150mm^2 ACSR is deemed small and a 250 mm^2 ACSR conductor is recommended
- At Muntanga, construct a 2 x 30/15/15 MVA 132/33/11 kV substation complete with protection, metering, SCADA and Optic fibre telecommunications
- Install 5 panel SF6 insulated 11kV indoor switchboard (2I + 2F +1BS) and 7 panel SF6 insulated 33 kV indoor switchboard (2I + 4F + 1BS)
- Lay 2 x 300m long cable circuits, each of six runs of 1C x 300 mm^2 XLPE 11 kV cable (two per phase), from the Muntanga 132/33/11 kV substation 11 kV switchboard to the Muntanga Process Plant 11 kV switchboard.

Tariff metering will be on the 2 x 11 kV incomer panels and installation will be done by ZESCO.

To minimise the transformation cost, the Muntanga 132 kV transformers are of the “three-windings” (“3-winding”) type, providing 11 kV supply for Muntanga. During normal system conditions, one 3-winding transformer will work as a 132/11 kV transformer while the other will work as a 132/33 kV transformer. In the event of an outage of either transformer, the available transformer will then provide both 33 kV and 11 kV. The alternative, a conventional 2-winding transformer solution, would require four transformers to achieve the same level of security of supply.

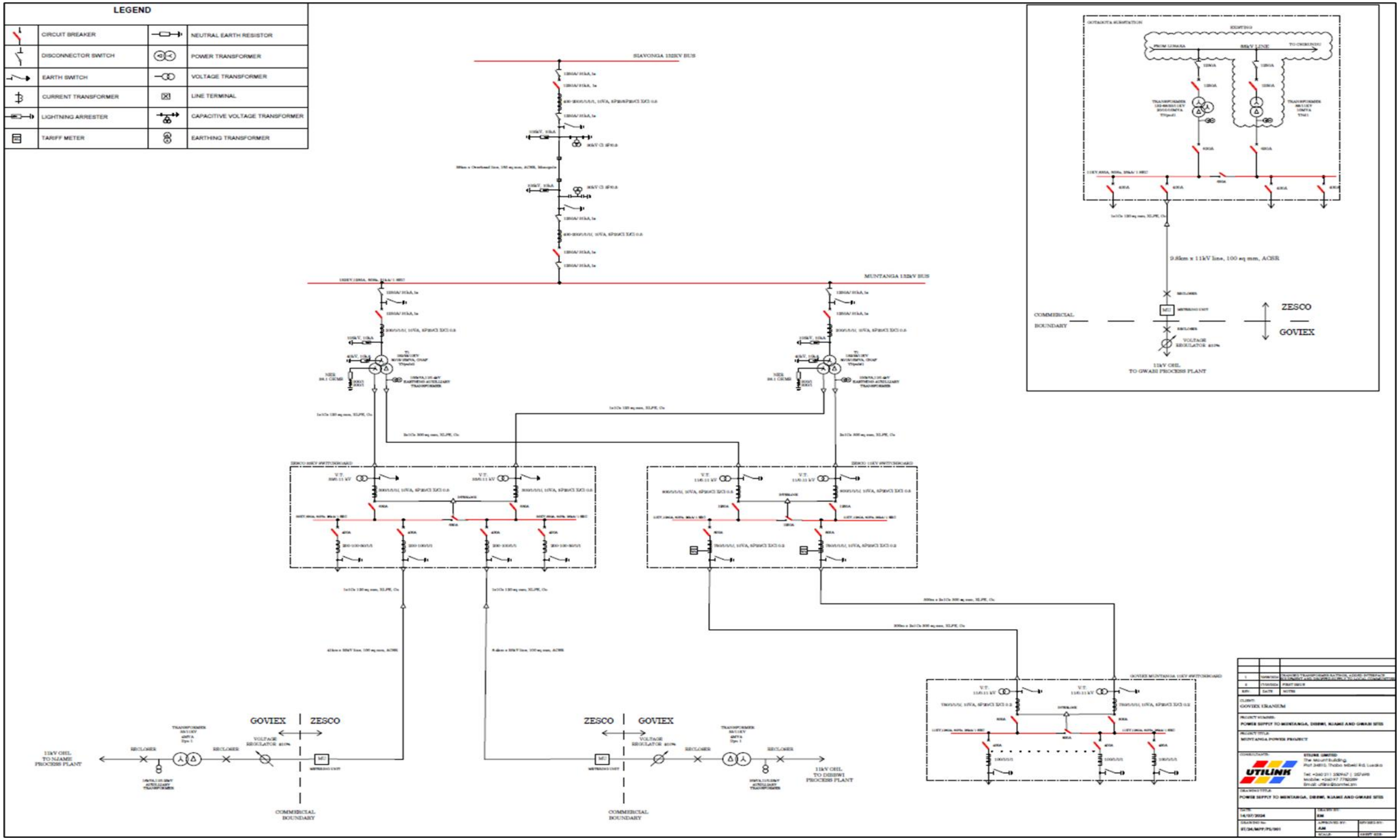


Figure 18-90: Single line diagram for overall power supply scheme to all sites

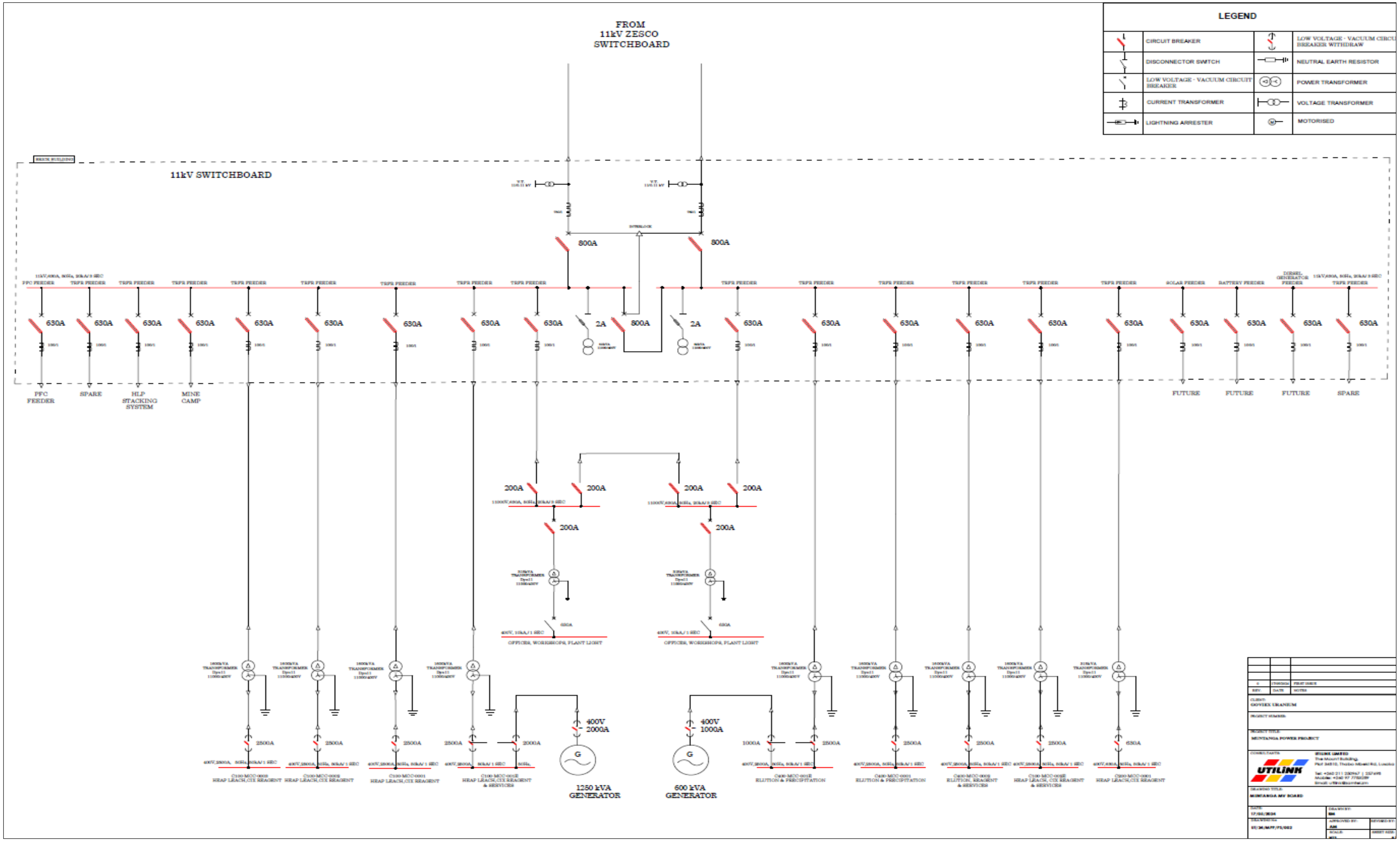


Figure 18-91: Single line diagram for 11 kV "MV Board"

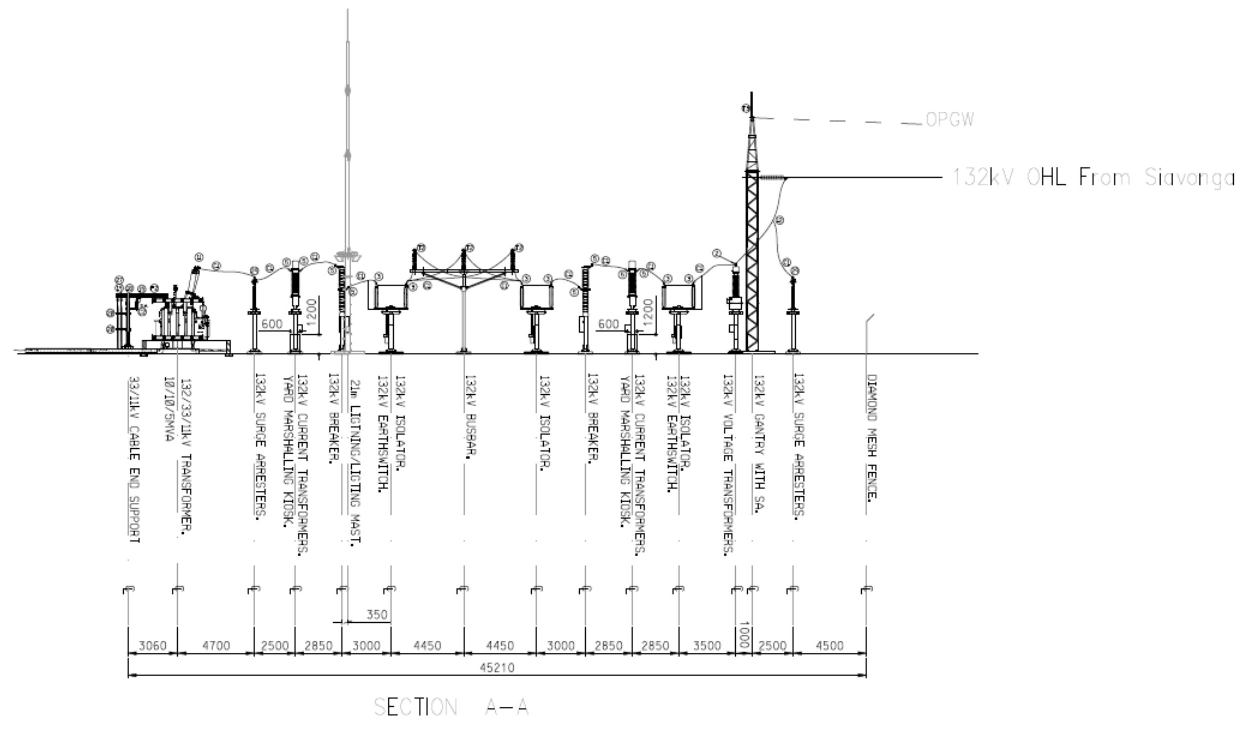


Figure 18-92: Elevation through Muntanga 132/33/11 kV substation

18.4.7.2. Zambia Electricity Supply Corporation Limited/ GoviEx interface

At the ZESCO/ GoviEx interfaces at Gwabi, the local voltage regulators and 33/11 kV substations (at Dibbwi and Njame) are placed on the GoviEx side of the commercial boundary. This is recommended as GoviEx will be in a much better position logistically than the local ZESCO teams to handle the technical operational issues that may arise with the voltage regulators and the transformer substations.

18.4.7.3. Power factor correction

The average plant power factors calculated from the estimated maximum are 0.86 at Muntanga and 0.88 at the satellite plants. It is a statutory requirement from the Energy Regulatory Board ("ERB") that consumers operate at a power factor that is not less than 0.92. To comply with this requirement, power factor correction equipment will have to be installed at each site.

18.4.7.4. Muntanga solar plant

The Muntanga solar plant is not part of the FS Capex, but the economics will be reassessed at each subsequent stage of study development.

As described in Section 18.4.5, the proposed plant would comprise 9 MWp of fixed tile solar panels on a ground-mounted structure with a 2 MWh battery energy storage system to smooth the transitions from grid to solar and vice versa. It is estimated that the inclusion of the solar plant would reduce the average tariff to 10 USD c/kWh with payback on the renewable investment in seven years.

The proposed solar plant would be located close mine site and the main incoming OHL substation from Siavonga to the southwest of the Muntanga Road (see Figure 18-93) and will feed in behind the metre at the 11 kV level.

Figure 18-93: Muntanga 9MW solar plant layout

18.4.7.5. Power supply: GoviEx camp and pit dewatering

The cost for 11 kV OHLs from the main 11 kV board to the GoviEx camp and to pit dewatering holes has been included in the Capex cost.

18.4.8.Project execution plan

18.4.8.1. Implementation

GoviEx will construct the required infrastructure, which will then be transferred to ZESCO for operation and maintenance. ZESCO will ultimately contribute to a proportion of the Capex cost, which will eventually be recouped through the tariff structure, and this will be confirmed in the Connection Agreement. The project delivery model will be an Engineer, Procure and Construct Contract ("EPC"). GoviEx will procure a contractor ("EPC Contractor") to undertake delivery of the project, from design to commissioning. GoviEx will appoint an Owner's Engineer ("OE") to manage the tender and monitor construction and commissioning. The EPC Contractor, OE, and ZESCO representatives will form the "Implementation Team" as defined within the Connection Agreement.

18.4.8.2. Package breakdown

The Project comprises four components, where Component A will be constructed prior to production:

- Construction of the 132 kV line from Siavonga to Muntanga, including works within the Siavonga substation, and construction of the Muntanga 132/33/11 kV substation
- Construction of the 11 kV line from Gota-Gota 88/11 kV substation to Dibbwi, including works within the Gota-Gota substation, and construction of the Gwabi 11/04 kV substation
- Construction of the 33 kV line from Muntanga 132/33/11 kV substation to Njame, and construction of the Njame 33/11/04 kV substation.

- Construction of the 33 kV line from Muntanga 132/33/11 kV substation to Dibbwi, and construction of the Dibbwi 33/11/04 kV substation.

Package A will have three Lots:

1. Lot 1: Siavonga-Muntanga 132 kV transmission line
2. Lot 2: 132 kV line switch-bay at Siavonga 330/132 kV substation
3. Lot 3: Muntanga 2 x 30/15/15 MVA 132/33/11 kV substation.

18.4.8.3. Responsibilities

GoviEx will provide the following:

- Selection and securing of land and right of way for the OHL
- Construction permits for construction
- Funding the works and
- Importation permits.

The EPC Contractor is responsible for the engineering, development, construction, and supply periods of the project, i.e., engineering, procurement, construction, testing, commissioning, and supply of the Works and shall deliver the packages such that it is fit for purpose and free of defects to be handed over to ZESCO.

ZESCO will form part of the implementation team and ensure the construction of the infrastructure is fit for purpose and meets the specifications required for their future ownership and operation.

18.4.8.4. Construction schedule

A construction schedule has been developed for the FS. The implementation team will prepare the actual construction schedule. The key phases and dates are shown in Table 18-30.

Table 18-30: Key schedule items

Item	Duration	Responsible
EPB	Q4, 2024 - Q1 2025	GoviEx
Wayleave agreements	Q1, 2025 – Q2 2025	GoviEx
Tendering	Q2 2025	GoviEx
EPC detailed design, procurement	Q4 2025 – Q1 2026	EPC contractor
EPC construction & equipment manufacture & delivery	Q2 2026 – Q2 2027	EPC contractor
Power infrastructure commissioning	Q3 2027	Implementation team
Processing plant commissioning	Q4 2027	GoviEx

18.4.9.Risks

18.4.9.1. Grid reliability under normal operating conditions

ZESCO has not provided specific details of outage records at the connecting substations; however, it has been confirmed by Utilink from prior historical knowledge, that the availability of both substations is understood to be >96 % at the primary voltage level (e.g., 132 kV and 88 kV busbars).

Under normal operating conditions in Zambia, and with a suitable connection agreement in place providing preferential connection, unplanned outages are not expected to impact annual plant production. As part of the connection agreement, ZESCO will assign a team to the Project, which will manage this and all the technical issues. The connection agreement will establish maintenance schedules on the transmission lines.

Daily, the Project electrical team will liaise with the ZESCO team with regard to demand, consumption, maintenance shutdowns at the plant and line maintenance, and this would be a two-way interaction. The occurrence of unplanned outages would need to be monitored and tracked, and corrective actions taken should the level of outages increase.

18.4.9.2. Drought/ Load shedding

The Zambian power sector is currently experiencing a “challenging period” with drought causing low water levels in reservoirs and as a consequence, power rationing (eight-hour daily load shedding) across the country. The low water levels are associated with an "El Nino" year. As previously mentioned, a "Special Negotiated Tariff" is anticipated to ensure preferential access to power supply at all times.

Zambia experienced low water levels before being associated with El Nino; the last major period of load shedding where Zambia imported between 785-2,000GWh was from 2015 to 2017, which coincided with one of the strongest El Nino events on record. Other periods of load shedding have been experienced but it is expected that power will be maintained to critical industries such as mining under the PPA; during these times it might be possible that mines are requested to provide "load support" using their critical back-up generation capacity. It is anticipated that Zambia will, over the coming years, diversify its generation mix and strengthen SAPP interconnectors to ensure power can be imported albeit at a price.

18.4.9.3. Assumed Zambia Electricity Supply Corporation Limited tariff

Ideally for the FS, negotiations with ZESCO would have been advanced to a stage where a PPA was drafted which indicates the tariff. In its absence, Utilink and SRK proposed a tariff based on benchmarks from actual and negotiated PPA for operating mines in Zambia.

18.4.10. Opportunities

18.4.10.1. The connection agreement

The total capacity of the Muntanga 132 kV is stated as 25 MVA. The CA should be agreed for the full 25 MVA which ensures GoviEx has maximised capacity versus investment. ZESCO should reimburse 30 % of the line cost within the tariff; however, the details of this (length, and duration) are yet to be defined.

18.4.10.2. Muntanga 11kV board

The 11 kV board has been designed to an FS level. The capital cost is based on benchmark cost from a prior project in 2022 adjusted for inflation, where the MV board arrived as a pre-engineered and fabricated solution under an engineer, procure and supply contract. The cost is considered conservative and should be reduced through competitive tendering for design and construction.

18.4.10.3. Assumed Zambia Electricity Supply Corporation Limited tariff

In the absence of a draft PPA, the proposed tariff of 12.5 USD c/kWh is considered a reasonable estimate. There is the opportunity to negotiate a tariff that is less than 12.5 USD c/kWh. The ZESCO contribution to the capital cost of infrastructure would likely be through a tariff adjustment.

18.4.10.4. Independent power producer solar plant with net metering

Installing a solar plant would reduce the overall levelised cost of energy but would increase capital costs. Potentially, this could be initiated following the commencement of operations. With recent changes to legislation, there is the potential to investigate oversizing the plant and net-metering back to the ZESCO grid.

18.4.11. Recommendations and next steps

The following recommendations are made:

- Continue to discuss and agree on the CA and commence PPA negotiations with ZESCO
- Review the design basis for the 132 kV line with respect to the requirements from the satellite projects to determine if the substation design can be refined
- Development of the powerline requires action during 2025. Completion of the EPB and wayleave agreements coupled with the procurement of an Owner's Engineer to facilitate the preparation of a tender package for competitive tender for the bulk power supply infrastructure should be carried out in accordance with the FS schedule. In advance, contact with and pre-screening of potential Contractors should be carried out.

18.5. Mine Camp

18.5.1. Camp Buildings

Camp buildings consisting of accommodation units housing 82 site officials, dining and laundry rooms will be constructed at site from prefabricated materials. The accommodation units, covering an area of 1356 m^2, will be as follows -

- **Management**: Two units each of one bedroom executive will be constructed for the Management Staff. The units will accommodate four persons and each room will be self-contained, air conditioned and will be equipped with a lounge, kitchenette and one bedroom. The rooms will also have a bed, a table and chair, and a built-in cupboard.
- **Senior Level:** Approximately 19 units housing 38 persons will be constructed for the Senior level mine officials. Each room will be self-contained, air conditioned and will be equipped with a bed, a table and chair, and a built-in cupboard. Each room has an ablution facility with a shower, hand basin and WC
- **Dormitory Units:** Two units for 40 persons will be constructed. Each room will be equipped with a beds, a table and chair, and a built-in cupboard. Each unit has a communal ablution facility with showers, hand basins and toilets. The units are equipped with ceiling fans.
- **Camp kitchen and mess:** The camp kitchen and mess area with suitable stores and freezer rooms attached will be constructed from prefabricated materials. The kitchen will be fully equipped and will be run by the catering service provider. The dining room will accommodate a total of 80 people at a sitting.
- **Recreational area and gymnasium:** The recreational area and gymnasium will be constructed with suitable stores and toilet facilities. The area will have bar facilities and will be equipped with DSTV, a pool table and darts area. A football pitch for seven-a-side soccer will be provided.
- **Accommodation for others:** In alignment with the GoviEx local employment policy, the majority of unskilled and semi-skilled labour will be recruited from nearby communities and will therefore be housed within designated resettlement zones or neighbouring villages. Senior professionals and other skilled personnel will be sourced from external towns. Due to limited housing capacity at the mine camp, skilled employees who cannot be accommodated on-site will be required to secure their own housing in Chirundu, approximately 70 km away.

19. Marketing studies and contracts

This section has been authored by GoviEx personnel and aims to provide an overview of the fundamental principles of the uranium market and how the derived U_3O_8 is sold into the market; transported; and transformed for use in nuclear reactors. As such the following elements will be described:

- Understand the position and role of uranium within the nuclear fuel cycle
- Analyse U_3O_8 demand with reference to the U_3O_8 requirements of the world's reactors
- Explain the transformation of U_3O_8 into UF6 and the role of the conversion facilities that provide such a service
- Summarise the requirements for transportation of U_3O_8 from the Project to the conversion facilities
- Examine the contractual relationship between GoviEx as the Uranium producer and the conversion facilities
- Analyse the uranium market prices and pricing mechanisms.

19.1. Nuclear fuel cycle

The "nuclear fuel cycle" includes all nuclear operations ranging from the mining of uranium ore to the reprocessing of spent fuel and the ultimate radioactive waste disposal.

The fuel cycle is made up of a series of processes that manufacture reactor fuel, burn the fuel in a reactor to generate electricity, and manage the spent reactor fuel (Figure 19-1). These processes are grouped into three components, the front end, which includes all activities prior to placement of the fuel in the reactor, the service period, when the fuel is converted into energy in the reactor, and the back end, which covers all activities dealing with spent fuel from the reactor. If the spent fuel is sent to storage, the cycle is referred to as open. If it is reprocessed to recover useful components, it is known as closed. The United States employs an open fuel cycle, while France, Russia, China, and Japan reprocess their spent fuel.

The components of the cycle are organised as follows within this case study.

19.1.1. The front end

- Uranium metallurgy, conversion to uranium hexafluoride, and fabrication of fuel rods
- Uranium enrichment.

19.1.2. The service period

- Nuclear reactors

19.1.3. The back end

- Reprocessing spent fuel
- Nuclear waste.

Figure 19-1, identifies the key process steps required for both the front end and back-end activities within the nuclear fuel cycle. (European Nuclear Society, 2003).

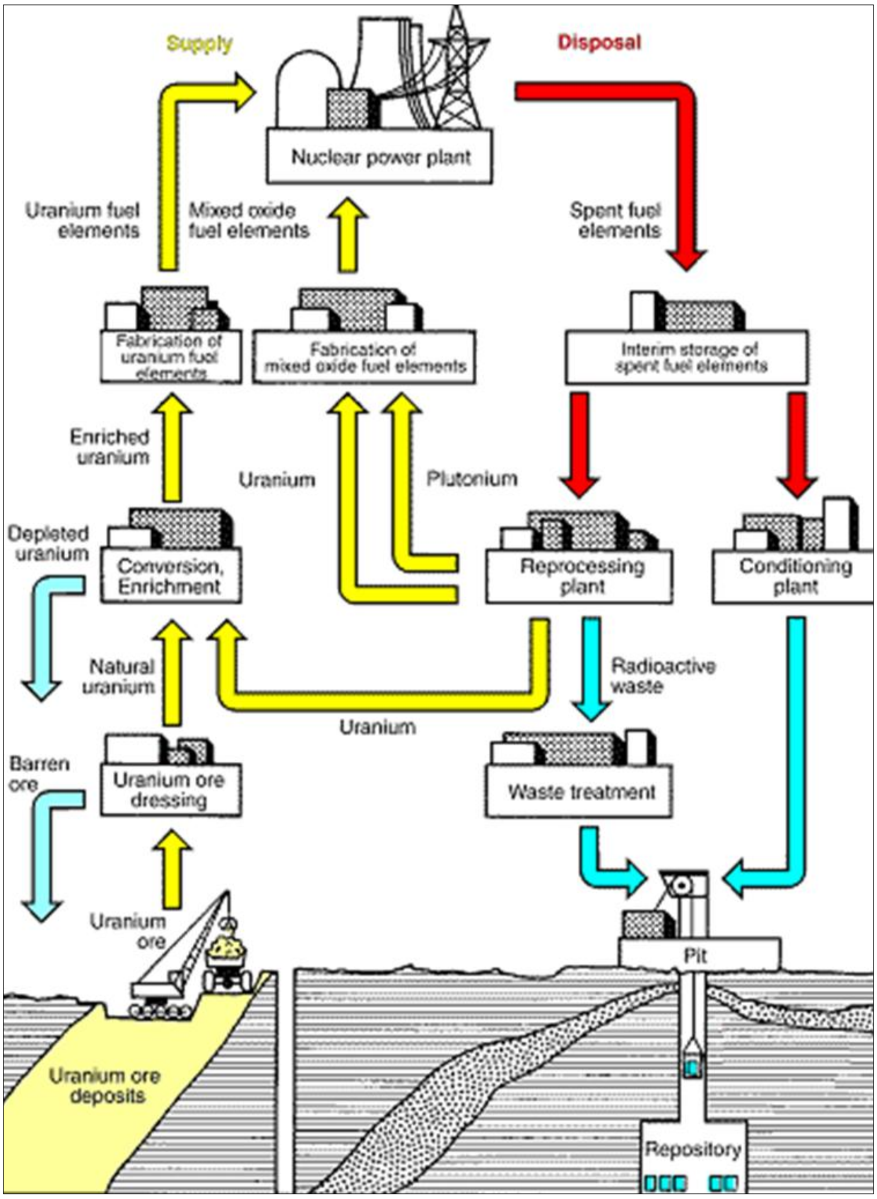


Figure 19-1: Key process steps required for front-end and back-end activities within the nuclear fuel cycle

The ' front-end' of the cycle begins with the extraction of the uranium ore by mining, then:

- It is processed to refine the material to U_3O_8. The U_3O_8 is packed in 55 US gallon drums (UN 1A2W) which are then transported from the processing plant to a UF6 Conversion facility
- Once the U_3O_8 has been converted to UF6 at the Conversion facility it is transported to an Enrichment facility
- At the Enrichment facility, the U-235 isotope is concentrated from 0.711 % up to a maximum of 5 % for Low Enriched Uranium (“LEU”) and 20 % for High Assay LEU (“HALEU”)
- Following Enrichment, the Enriched Uranium Product (“EUP”) is produced which is transported to a Fuel Fabrication facility where the fuel that powers the Reactor is manufactured
- The nuclear fuel is transported from the Fuel Fabrication facility to the utility site where it is loaded into the Reactor to generate electricity.

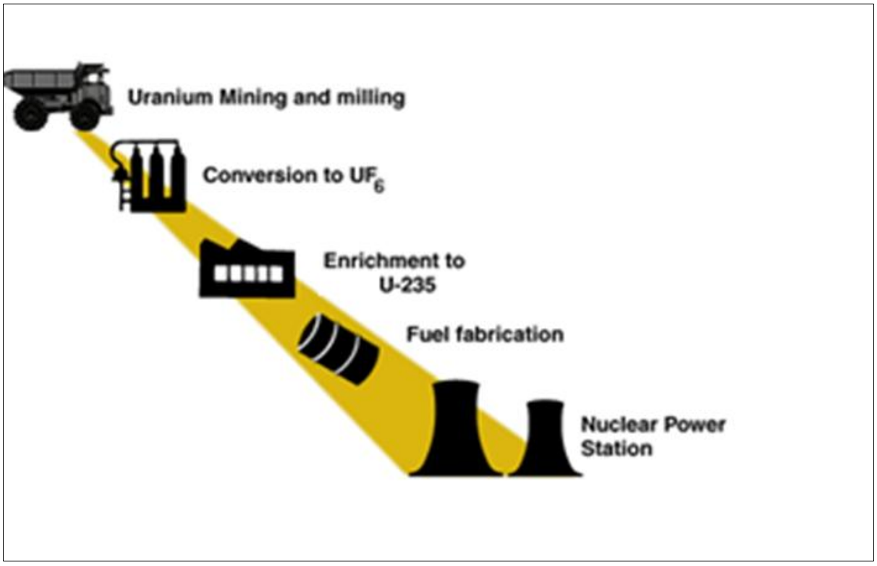


Figure 19-2: Front-end fuel cycle

After approximately three years of electricity production, nuclear fuel is removed from the Reactor and undergoes further steps including temporary storage, reprocessing, recycling and eventually disposal. This is commonly known as the back end of the nuclear fuel cycle.

19.2. Uranium market

19.2.1. Demand

According to the World Nuclear Association updated September 2023:

- The world will need a significantly increased energy supply in the future, especially cleanly generated electricity
- Electricity demand is increasing about twice as fast as overall energy use and is likely to rise by more than half by 2040
- Nuclear power provides about 10 % of the world's electricity and 25 % of low-carbon electricity
- Almost all reports on future energy supply from major organisations suggest an increasing role for nuclear power as an environmentally friendly way of producing reliable electricity on a large scale.

A key advantage of nuclear is its ability to provide reliable and economic base-load power on a near zero-carbon full life-cycle basis. For example, it is worth mentioning that in the United States alone, nuclear energy currently provides around 55 % of the country’s carbon-free electricity, and in the European Union it accounts for 53 % of the region’s carbon-free electricity.

In 2022 the world's nuclear power plants supplied 2 487 TWh of electricity through 391 GWe of operable capacity. This avoided the emission of 2.0 billion tonnes of carbon dioxide compared to the equivalent amount of coal power generation, in addition to total avoided emissions of around 78 billion tonnes since 1970. Nuclear power avoids the emission of pollutants including oxides of sulfur and nitrogen and is therefore favoured by some countries as a solution to combat air pollution.

In the future, nuclear energy could contribute substantially more given the expectation of rapidly rising electricity demand and the changes in energy consumption. The Artificial Intelligence (“AI”) and transport sectors offer great potential with the spread of high-tech AI data centres, electric vehicles, and programmes to implement higher use of passenger electric vehicles are underway in numerous countries worldwide. Apart from electricity generation, nuclear represents a credible low-carbon source of process heat for various applications, such as district heating, water desalination, oil and chemical refining, and hydrogen production.

Table 19-1: World Nuclear Association nuclear capacity scenarios for 2040, GWe (World Nuclear Association (WNA) 2023)

2023 [GWe]	Case	Forecast [GWe]	Variance [%]
	Lower	486	+29
377	Reference	686	+82
	Higher	931	+147

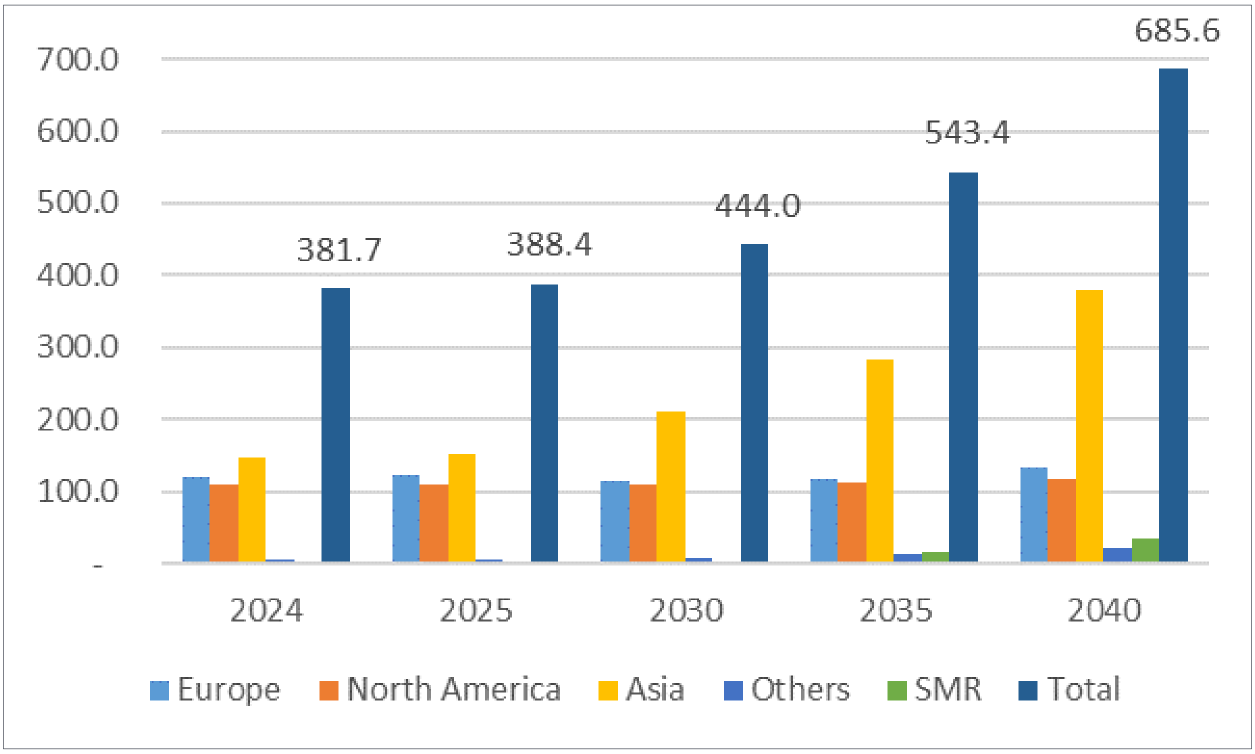


Figure 19-3: Nuclear generating capacity to 2040, GWe net – Reference Case (WNA Sept 2023)

The drivers for the World Nuclear Association scenarios embrace broader changes than climate change policy alone. The Reference Scenario in Figure 19-3 is largely a reflection of current government policies and plans announced by utilities for nuclear in the next 10 to 15 years.

In the International Energy Agency ("IEA") World Energy Outlook 2020 it is noted that achieving the pace of CO_2 emissions reductions in line with the Paris Agreement is already a huge challenge, as shown by their Sustainable Development Scenario. The IEA noted that it requires large increases in efficiency and renewable investment and an increase in nuclear power. This report identifies the even greater challenges of attempting to follow this path with much less nuclear power. It recommends several possible government actions that aim to ensure existing nuclear power plants can operate as long as they are safe, support new nuclear construction and encourage new nuclear technologies to be developed.

During the World Climate Action Summit of the 28th Conference of the Parties to the United Nations Framework Convention on Climate Change in late 2023, more than 20 countries from four continents launched the Declaration to Triple Nuclear Energy. The declaration recognises the key role of nuclear energy in achieving global net-zero greenhouse gas emissions by 2050 and keeping the 1.5° goal within reach. Core elements of the declaration include working together to advance a goal of tripling nuclear energy capacity globally by 2050 and inviting shareholders of international financial institutions to encourage the inclusion of nuclear energy in energy lending policies.

Endorsing countries include:

- United States of America ("USA")
- Armenia
- Bulgaria
- Canada
- Croatia
- Czech Republic

- Finland
- France
- Ghana
- Hungary
- Jamaica
- Japan
- Republic of Korea
- Moldova
- Mongolia
- Morocco
- Netherlands
- Poland
- Romania
- Slovakia
- Slovenia
- Sweden
- Ukraine
- United Arab Emirates, and
- United Kingdom.

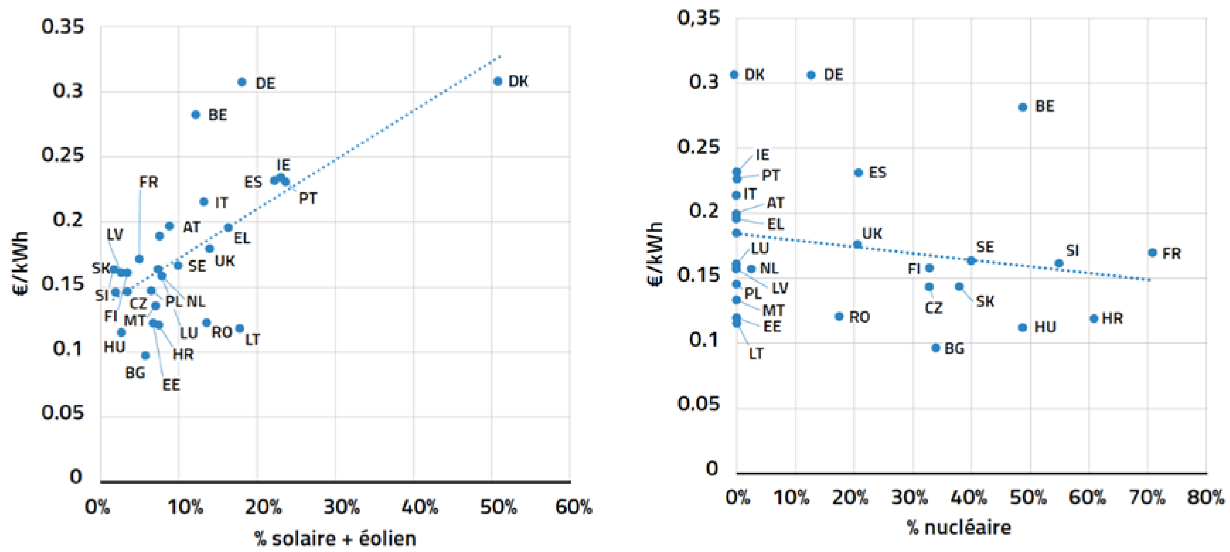
In the USA there has been a wave of bipartisan legislation in support of nuclear energy, for example, the Infrastructure, Investment and Jobs Act under which USD6 billion has been pledged to support "at risk" nuclear units in the USA. In addition, the Inflation Reduction Act has allowed significant tax credits and incentives (available until 2032) for all existing and new reactors.

Furthermore, the July 2024 passing of the ADVANCE Act will boost new Small Modular Reactors ("SMR") and advanced reactor deployment in the USA in response to growing requirements for energy, as data centres, electric vehicles, and industrial processes all search for a clean and reliable source of power. In this context, the Department of Energy ("DOE") states that nuclear will be part of the energy solution, whereby the USA has already committed to tripling the nation's nuclear capacity over the next 25 years, to help secure a clean energy future.

The inaugural Nuclear Energy Summit took place in Brussels in March 2024, with representatives from 32 countries declaring support for nuclear energy, in areas including financing, regulatory cooperation, technological innovation, and workforce training. This will support the expansion of nuclear energy to help address climate change and boost energy security.

Nuclear power plants contribute to electricity security in multiple ways. They help to keep power grids stable and can be a good complement to decarbonisation strategies since, to a certain extent, they can adjust their operations to follow demand and supply shifts. As the share of variable renewables like wind and solar photovoltaics ("PV") rises, the need for such services will increase.

In a recent report "Road to EU Climate Neutrality by 2050", authored by the European Conservatives and Reformists ("ECR") Group and Renew Europe in January 2021, it was reported under their key takeaways that with respect to both spatial requirements (area of land required) and costs of electricity, nuclear power provides further advantages over renewable energy. The cost advantage of nuclear energy increases once system costs are added and power increases further with higher penetration rates of renewable energy as highlighted by the figures below for Europe.



Price of electricity (household)

From: Prof. Samuel Furfari, Uinversite Libre de Bruxelles, 2019.

Source: Eurostat (Dec 2018)

Figure 19-4: Europe's cost of power based on the percentage of solar, wind, and nuclear energy

Table 19-2: Average power generation by area by energy source for Europe

	Average GWh/ km²		Indexed to nuclear [i.e. nuclear produces x times more electricity per km²]	
	NL	CZ	NL	CZ
Onshore wind land	13	13	534	534
Onshore wind water	14	n/a	506	n/a
Offshore wind	26	n/a	266	n/a
Solar roof	136	163	51	43
Solar land	47	65	148	108
Nuclear	6 982	6 982	1	1

Beyond the historical large-scale reactors that have been under construction since the early nuclear renaissance that began in the 1950s, where reactor sizes grew from 60 MWe to more than 1 600 MWe, focusing on economies of scale, there has been an increasing focus on the development of SMRs.

SMRs are defined as nuclear reactors generally 300 MWe equivalent or less, designed with modular technology using module factory fabrication, pursuing economies of series production and short construction times.

Today, due partly to the high capital cost of large power reactors generating electricity via the steam cycle and partly to the need to service small electricity grids under about 4 GWe, there is a move to develop smaller units. These may be built independently or as modules in a larger complex, with capacity added incrementally as required.

Economies of scale are envisaged due to the numbers produced. There are moves to develop independent small units for remote sites. Small units are seen as a much more manageable investment than big ones whose cost often rivals the capitalisation of the utilities concerned.

An additional reason for interest in SMRs is that they can more readily slot into brownfield sites in place of decommissioned coal-fired plants, the units of which are seldom very large – more than 90 % are under 500 MWe, and some are under 50 MWe. In the USA coal-fired units retired over 2010 to 2012 averaged 97MWe, and those expected to retire over 2015 to 2025 averaged 145 MWe.

Generally, modern small reactors for power generation, and especially SMRs, are expected to have greater simplicity of design, the economy of series production largely in factories, short construction times, and reduced siting costs. Most are also designed for a high level of passive or inherent safety in the event of malfunction. Also, many are designed to be placed below ground level, giving a high resistance to terrorist threats.

19.2.2.Supply

The supply of uranium is provided from two sources, the first and primary source is mined production and the second is from non-mined production, which are secondary sources such as inventory and enrichment tailings upgrading.

19.2.2.1. Primary supply

In its September 2023 Nuclear Fuel Report, the WNA reported that primary production was at approximately 73 % of annual demand, down from 94 % in 2012. In the same Report, WNA forecasted that this deficit would continue to widen unless new primary production was forthcoming (see Figure 19-5).

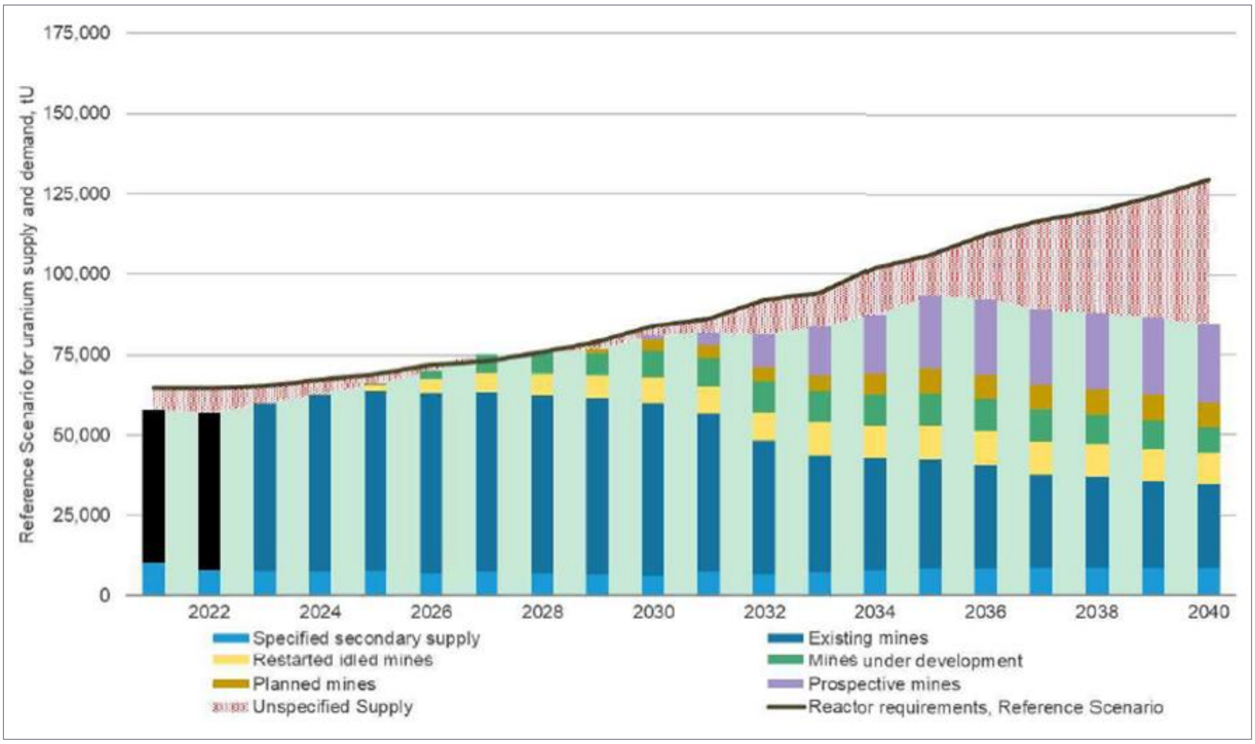


Figure 19-5: Supply/ demand deficit forecast increase without new production

Primary production has been under pressure for many years as a result of low prices and a policy of production cutbacks ("supply discipline") adopted by several major suppliers.

In December 2017, the largest global uranium producer Kazatomprom, accounting for about 40 % of annual global production, announced that it would reduce annual production by 20 %, compared to planned levels under Subsoil Use Contracts, over three years from January 2018, and in August 2021 extended the "flex down" of production by 20 % through 2022.

During 2020, the uranium mining industry was impacted by COVID-19 with Cameco closing the Cigar Lake Mine for some months, restarting in 2021. Kazatomprom halted all in-situ recovery ("ISR") wellfield development for six months. The production halts focused the major producers to purchase production shortfalls from the spot market.

At the beginning of 2021, the Ranger Mine in Australia closed due to resource depletion, and at the end of March 2021, the COMINAK Mine in Niger was finally closed. These two mines will account for the loss of 6 Mlb U_3O_8 from the primary supply.

In February 2022, Cameco announced that McArthur River would gradually re-start late in the year. Now, with the market having recovered, both Cigar Lake and McArthur are expected to produce around 18 Mlbs (each) in 2024. Furthermore, several other uranium mines, at the time of authoring, which were on standby/ care and maintenance, including Langer Heinrich in Namibia, Honeymoon in Australia, are now restarting, with first deliveries to be made this year (2024). Although the USA produced only 50 000 lbs U_3O_8 in 2023, new production is expected in the near future.

Ux Consulting Company LLC ("UxC") estimates that approximately 70 % of global production has a cost in excess of USD 30 /lb U_3O_8 and 20 % with costs over USD40 /lb, and with an increasing supply deficit higher cost production must be brought online to offset declining inventories and reserve depletion. Producers and developers will not risk capital to bring on idle capacity or construct new projects until uranium prices recover. In recent months, however, the spectre of high inflation threatens to increase the costs of mining materials and inputs and thus will inflate the cost of production in the uranium sector.

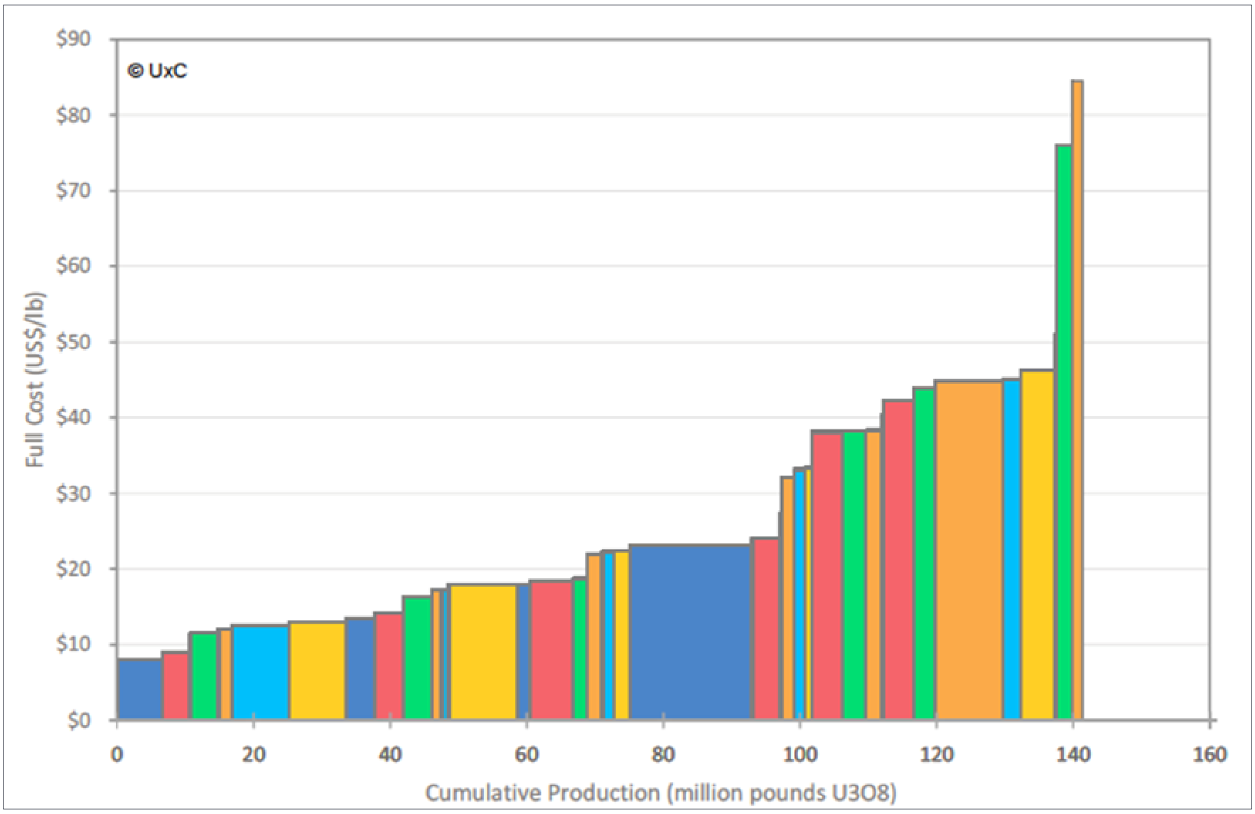


Figure 19-6: Production costs (Source: (UxC's Uranium Production Cost Study))

19.2.2.2. Secondary supply

Several sellers in the uranium market are supplied by various sources of inventory. This inventory is in the form of U_3O_8, UF_6, and EUP. Secondary sales are forecast to decline as a percentage of demand through declining inventories and reducing enrichment capacity.

Table 19-3: WNA reference scenario secondary supplies (tU equivalent)

	2021	2025e	2030e	2035e	2040e
US DOE	320	450	100	2,000	2,000
Global ERU/MOX	2 250	2 140	2 850	4 490	4 620
Global underfeeding and tail re-enrichment	7 930	5 280	3 340	2 000	2 000
Additional Russian secondaries					
Total	**10 500**	**7 870**	**6 290**	**8 490**	**8 620**
% Global reactor demand	17	11	8	8	7

Source: (WNA Market Reports 2021 and 2023)

The secondary sources are varied and comprise the following:

- Mixed oxide fuel ("MOX" and RepU – MOX is a combination of plutonium oxide recovered from spent fuel and new uranium oxide from DU (a "waste" product of the fuel enrichment), while reprocessed uranium (RepU) involves the removal of uranium and plutonium from spent fuel to fabricate new fuel. Although these two fuel sources have been used for many years, their contribution has been quite low (approximately 5 % of total uranium supply)
- United States ("US")/ United States Enrichment Corporation ("USEC") Government Stockpile Sales – Government strategic stocks that have been deemed surplus. Some of this material was sold through the

USEC/ Centrus, a public company that was previously a government organisation. In 2008, the US Department of Energy ("DOE") released its Excess Uranium Inventory Management Plan which provided details regarding the US government's plans to dispose of its excess uranium inventories. These inventories, totalling 153 million pounds, were built up over decades primarily through enrichment activities, weapons programs, and the US-Russian HEU (highly enriched uranium) programme. The uranium is in various forms, some of which are readily saleable, whereas others require substantial processing to bring to commercial reactor standards

- Global Underfeeding - With the Russian invasion of Ukraine in 2022, coupled with the ensuing war, transport barriers and international sanctions, Russian separative work unit ("SWU")/ LEU supplies are now less accessible to the West. Consequently, enrichers are having to review their underfeeding campaigns. In the current climate of geopolitical instability and uncertainty, western enrichers are now moving towards overfeeding (using more natural uranium feed to service their centrifuges) while they consider whether they should invest in expanded SWU capacity. This increase in natural uranium feed requirements/ higher tails assays at enrichment plants strengthens the case for new natural uranium production.

19.2.3.Outlook 2024 onwards

Since 2011 the key impact on primary uranium demand was excess inventories throughout the supply pipeline. Increasing nuclear energy production and primary uranium supply constraints have resulted in declining inventories. The uranium miners have reduced their inventories to just-in-time levels through supply reductions, sell down of surplus inventories, on-market purchases and in the case of Kazatomprom sale of its surplus inventory to financially fund Yellow Cake.

Utility inventories have been declining as long-term contracts have unwound, and utilities have undertaken active inventory control. This has been compounded by the uncertainty associated with geo-political factors, especially affecting the USA, including the Iran Sanctions, Russia Suspension Agreement, and Section 232/ Nuclear Fuel Working Group. During 2020 and into the start of 2021, the utilities were affected by COVID, which while it reduced nuclear energy generation by approximately 4 % in 2020, resulted in suspended mine production, a uranium price uptick and a decline of between 20 % to 30 % of annual purchases.

Since 2021, the uranium spot price has risen significantly owing to the geopolitical instability prevailing in the RSOI, including the Russian invasion of Ukraine in February 2022 and the ongoing war. Furthermore, with the purchasing activities of investment funds such as SPUT and Yellowcake, together with the 2023 military coup in Niger, uranium supply has become tighter, with the result that considerably more investment in new primary uranium production is required to support the upsurge in interest in nuclear energy as a secure and clean form of electricity. In this context, the longer-term outlook for nuclear has been boosted by multi-national policy moves to decarbonise the world's electricity generation and to bolster energy security in uncertain times. Such nations as France, Sweden and South Korea held elections in 2022, and the winning candidates have declared their intention to support significant nuclear energy build in the future.

Furthermore, the USA, UK, Japan, Netherlands, Bulgaria, Hungary, Romania, Slovakia, Czech Republic and Canada have all pivoted towards providing financial and legislative support for future nuclear growth, while Belgium announced lifetime extensions of two nuclear units. In addition, China plans to expand nuclear generation capacity threefold in the next 16 years to a level of around 180 GW installed nuclear capacity, to promote clean electricity and provide energy security, while UAE and India have re-confirmed their pursuit of nuclear energy as a vital part of their energy base.

Nuclear energy programmes are being pursued in Poland, Egypt, Turkey, and Bangladesh, while Saudi Arabia, Serbia, Indonesia, Vietnam, Philippines and Italy are considering the potential of nuclear energy as an important plank in their energy policy, which means that the outlook for world nuclear energy growth is positive.

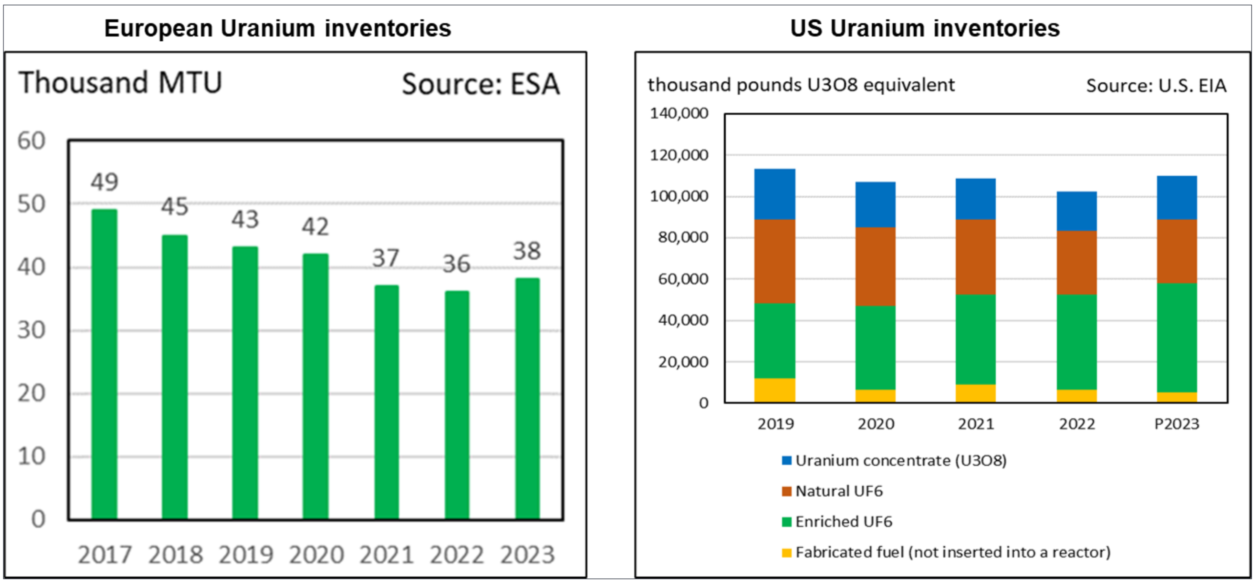


Figure 19-7: European and US uranium inventories have declined over recent years

Conversion and enrichment markets have been impacted by the rising price and increasing concerns on conversion and enrichment capacity medium to long term.

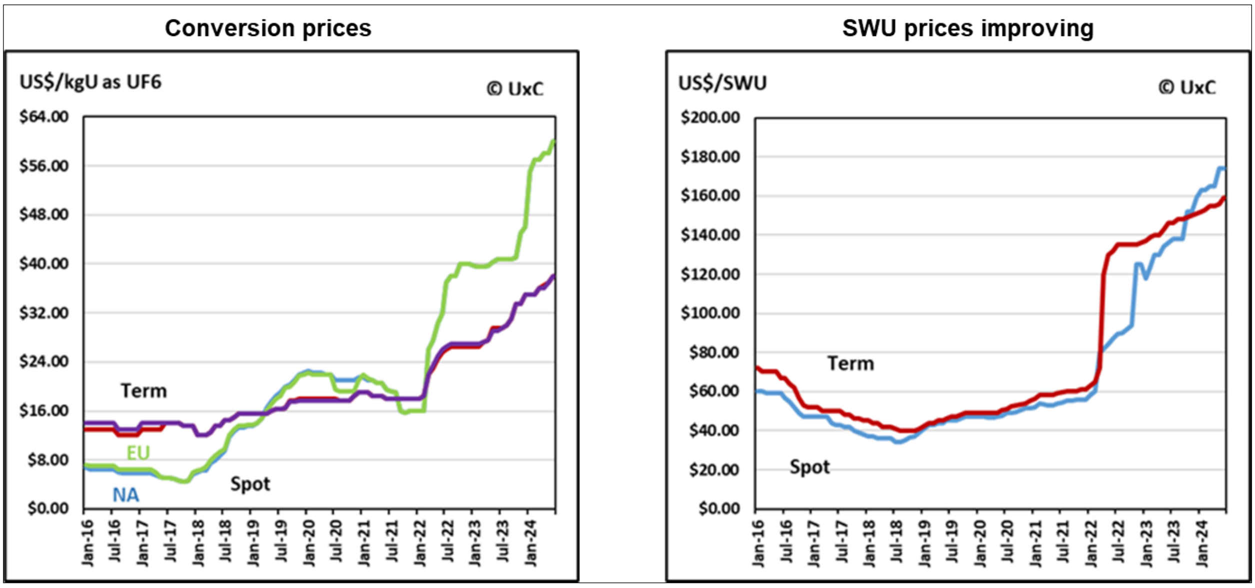


Figure 19-8: Conversion costs and SWU prices have been increasing since 2016

The increasing supply constraints, geopolitical disruptions, significant fund purchasing and declining inventories have been reflected in the improving uranium price. Based on history alone, uranium prices can make huge swings when future production levels are uncertain due to the long lead times required to bring new projects online. Since the earlier actions taken by Cameco and Kazatomprom to constrain supply, plus the events surrounding the conflict in the Ukraine, and the problems associated with shipping Kazak and Russian material to the West, the supply situation in 2024 has become very tight. On top of this, the many economic sanctions against Russia (including some “self-sanctions” by individual companies), have had serious implications for the flow of nuclear fuel deliveries to the West; and the uranium price has responded positively as a result.

While uranium demand increases, the market continues to suffer a deficit with regard to the availability of primary supply from uranium mines. In the past, this deficit had been filled by secondary supplies, but most of these inventories have now been depleted and increasingly, the enrichment services providers are switching to higher tails and overfeeding as a requirement to meet EUP demand, which puts further pressure on uranium supply.

There is a lack of new sizeable uranium production being developed since the last uranium bull market. Additionally, older uranium mines are getting closer to their end of life, with declines in production forecast from the existing mines in Canada, Kazakhstan and Niger over the next decade. This will require the development of new mines not only to replace the lost production but also to make up for the increasing demand.

However, uranium exploration budgets remain well below historical levels with no major new projects discovered in the past decade and an outlook for supply that is increasingly reliant on new mine development, which comes with associated permitting and technical and financial development risks.

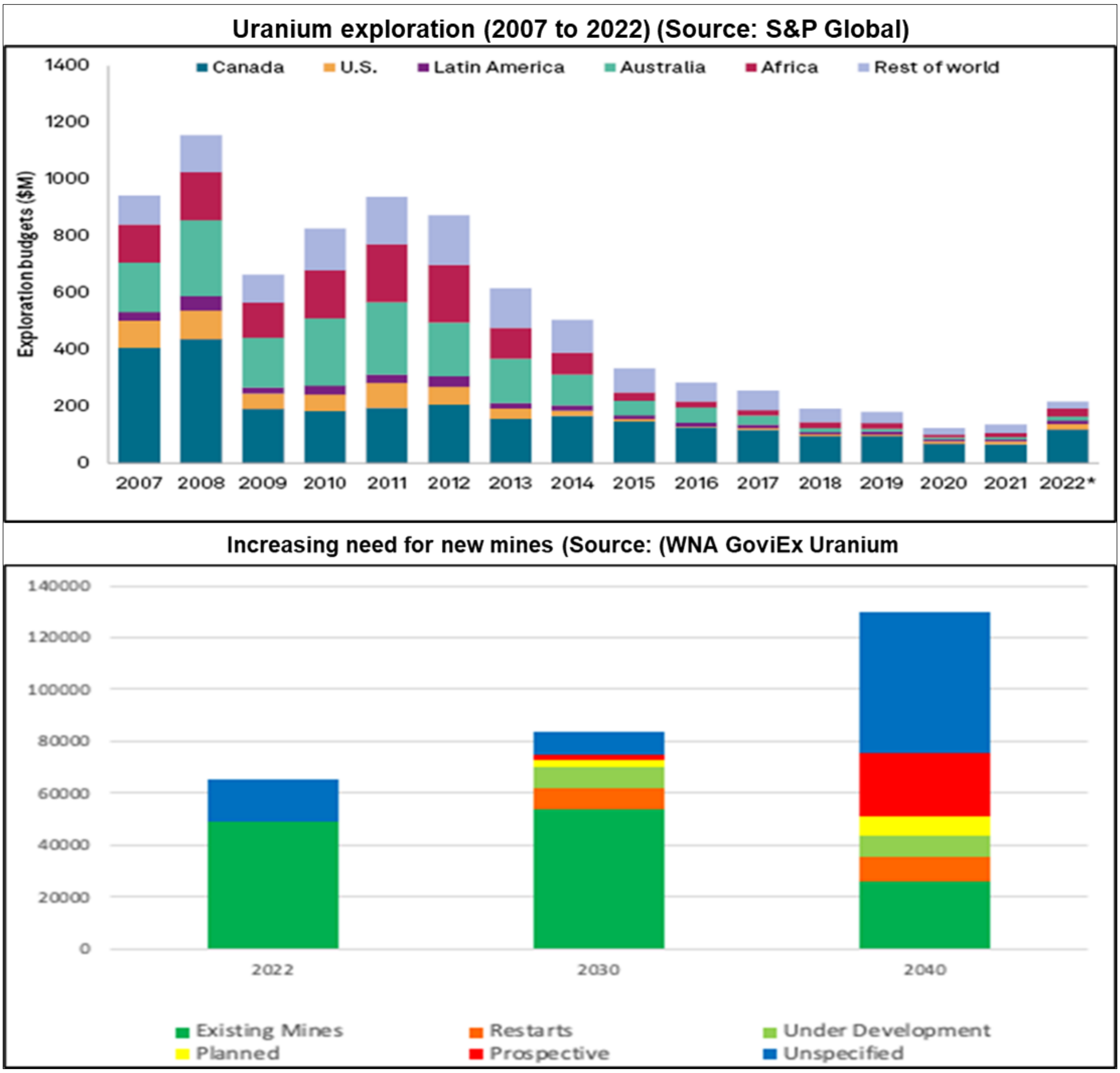


Figure 19-9: Lack of exploration and increased focus on new mines places major risk on supply

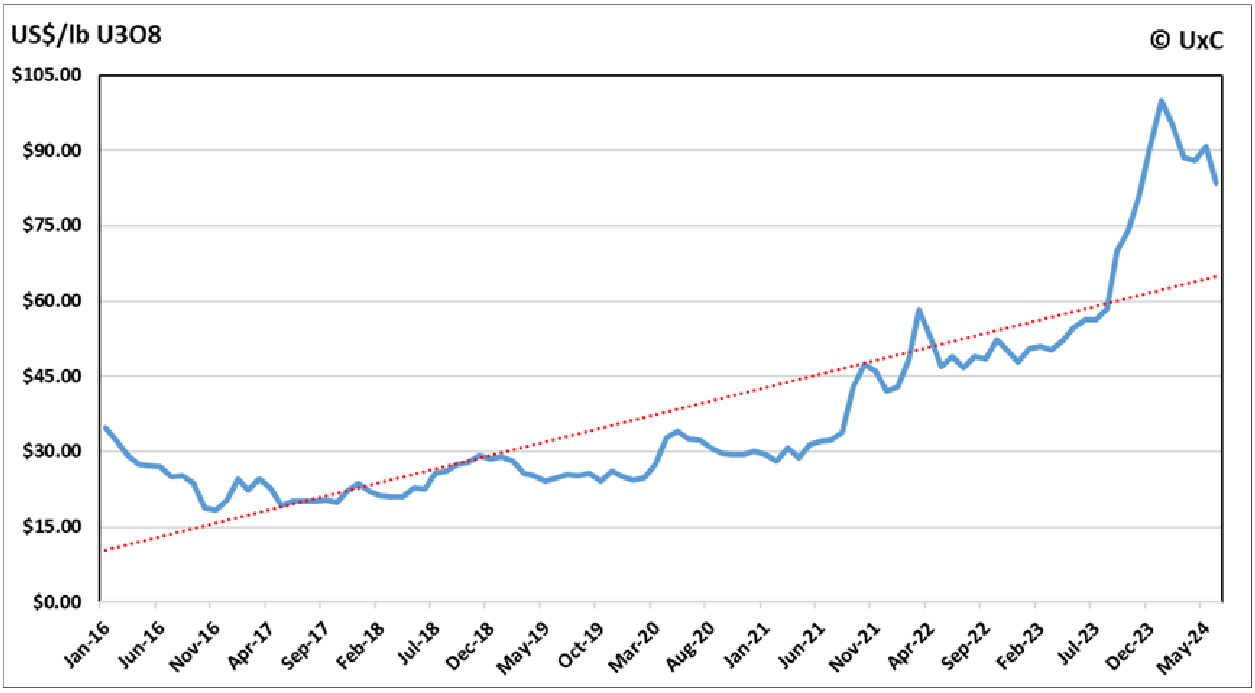


Figure 19-10: Uranium spot price quoted by UxC

Ux in its 2nd quarter 2022 Uranium Market Outlook forecast is expecting the uranium long-term contract price to remain well above the lower levels experienced in in 2020/2021.

Table 19-4: UxC forecast long-term contract prices (Q2/ 2024)

Scenario	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040
High long-term	79.82	86.30	88.36	90.30	91.68	93.78	95.36	97.26	98.62	100.95	102.94	104.50	106.94	108.77	110.65	112.52	114.46
Mid-long-term	77.83	80.18	81.36	82.81	84.86	86.30	88.30	89.81	90.60	91.54	92.62	92.82	94.81	96.25	97.47	98.33	99.57
Low long-term	76.08	77.01	75.41	73.90	73.51	74.21	74.73	76.18	77.56	78.32	78.71	79.53	79.56	80.50	81.35	82.55	83.24
Composite long-term	77.99	81.19	81.96	82.84	84.11	85.57	87.09	88.70	89.80	91.09	92.27	93.02	94.67	96.10	97.42	98.66	99.97

19.3. Conversion facilities

Globally, there are seven conversion facilities that can convert U_3O_8 into UF_6. Table 19-5 shows the capacity and estimated annual production of each of the conversion facilities.

Table 19-5: Conversion facilities

Conversion Facility	Nameplate Capacity tU as UF6
Cameco, Port Hope, Canada	12 500
Rosatom, Seversk, Russia	12 500
Orano, Comurhex, France	15 000
ConverDyn, USA (Honeywell)	7 000
CNNC China	15 000
Total	**62 000**

Source data (WNA)

- Cameco (Canada)
  - Cameco's Conversion method is split between two processing plants. U_3O_8 is delivered to Cameco's Blind River facility in Ontario, Canada where it is refined to UO_3. The UO_3 is then transported to Cameco's Port Hope facility, where it is converted to UF6
- Rosatom
  - Currently conversion in Russia is consolidated at the Siberian Group of Chemicals Enterprises at Seversk, in the Tomsk region. The site also has enrichment facilities. Rosatom supplies a full range of nuclear fuel cycle products to the global market. Conversion is exported mainly in the form of bundled products, in the form of fuel assemblies sold by TVEL and enriched uranium product ("EUP") sold by TENEX
- Orano (France)
  - Orano conducts conversion via a two-stage process at its Comurhex industrial plants at Malvesi and Tricastin sites in France. U_3O_8 is converted to UF_4 at Malvesi, and the UF4 is then transported to Tricastin for final conversion to UF_6
- ConverDyn (Honeywell Metropolis Works, USA):
  - The Honeywell Metropolis works is situated in Metropolis, Illinois, USA. The Conversion Facility is owned by Honeywell but it is operated by ConverDyn and is the only U_3O_8 to UF6 Conversion Facility in the USA. This facility has been operating since 1958. ConverDyn's ownership structure is 50 % Honeywell and 50 % General Atomics based in Denver, Colorado, USA. ConverDyn manage all aspects of the Conversion process including U_3O_8 deliveries, sampling and storage
- CNNC
  - CNNC has its conversion facilities at Lanzhou and Hengyang sites, which not only aim to meet UF_6 requirements for domestic usage but also for the supply of Chinese reactors built abroad.

19.3.1. Contracts with the conversion facilities

Deliveries to a conversion facility can be made by physical delivery or by Book Transfer. To maximise the sales opportunities relating to the U_3O_8 mined from the Project, it will be essential for GoviEx to have the ability to physically deliver its uranium ore concentrates containing U_3O_8 ("UOC") to the conversion facilities of ConverDyn, Cameco and Comurhex.

As such, GoviEx will need to enter a weighing, sampling, analysis and storage contract, for a U_3O_8 holding account to be established in the name of GoviEx (U_3O_8 holding account) with each of the aforementioned conversion facilities.

The conversion facility is responsible for the management of its customer's U_3O_8 holding accounts, which includes the administration process for crediting and debiting U_3O_8 transactions. Typically, U_3O_8 holding account statements are issued by the conversion facility quarterly.

In general, the conversion facilities do not offer their customers the ability to physically withdraw U_3O_8 and UF_6 from their U_3O_8 holding accounts; it can only be book transferred between customers who have U_3O_8 holding accounts at the same conversion facility.

19.3.2. Weighing, sampling, analysis and storage of U_3O_8

Prior to any physical delivery of U_3O_8, a producer will be required to enter into a contract with a conversion facility for the weighing, sampling, analysis and storage of U_3O_8. Depending on the conversion facility, the terms and conditions within such contract will include, but not be limited to:

- The producer's obligation to provide a U_3O_8 delivery schedule each year
- The conditions of delivery, for example, packaging, loading, marking, labelling, emergency response, shipping documentation
- The conversion facilities' weighing and sampling process
- The specification of the UOC will be accepted at the conversion facility for conversion without surcharges. The specification for each conversion facility may vary slightly but will broadly be in line with ASTM specification C967-13 Standard Specification for Uranium Ore Concentrates (typically Australian, Canadian and Kazak UOC measure around 98 % to 99 % U_3O_8, content and have few impurities).

19.4. Physical delivery of triuranium octoxide

It is the responsibility of the party physically delivering the UOC (usually the producer) to pay for the transportation to the conversion facility.

A percentage of the total UOC delivery quantity will be credited to the U_3O_8 holding account on the date of delivery. The UOC is then weighed and analysed by the conversion facility to confirm its acceptance. This analysis can take up to one hundred days. Once completed, the balance of U_3O_8 will be credited to the U_3O_8 holding account. A fee may be charged for this service, particularly if there are any surcharges imposed by the conversion facility relating to impurity concentration levels. These charges vary depending on the terms and conditions negotiated in the contract.

The title to the U_3O_8 will remain with the U_3O_8 owner (usually the producer) until it is sold. Risk of loss and damage will transfer to the conversion facility upon physical delivery of the U_3O_8 at the conversion facility.

19.5. Book transfer delivery

When a customer agrees to purchase U_3O_8 from a producer (or other supplier), the customer will expect to receive delivery by means of book transfer. This transaction will appear as a credit in the customer's U_3O_8 holding account and a debit in the producers account at the conversion facility.

The title passes from the seller to the buyer on the date the U_3O_8 is book transferred. Risk of loss and damage remains with the conversion facility.

It is standard practice for a conversion facility to impose a charge for book transferring U_3O_8. The only exception is if the buyer is a conversion customer of the conversion facility.

Normally the seller will provide a book transfer notice document around ten working days prior to the required delivery date to instruct the conversion facility to affect the book transfer from one U_3O_8 holding account to another.

19.6. Transport to market

It is proposed that deliveries will be made by road (in an escorted convoy) from the operating facilities westwards through Zambia, via Livingstone and the Caprivi Strip (a major highway), across the border into Namibia, and shipped out of the Port of Walvis Bay using a container shipping line. The road transport route from the Muntanga Mine to the Port of Walvis Bay is approximately 2 000km in length.

A Zambian trucking company familiar with the regulations for radioactive transport will be used for road transportation through Zambia into Namibia to the Port of Walvis Bay.

The planned transport route has been used previously for uranium shipments out of Malawi, via Zambia on the way to Walvis Bay. Walvis Bay itself has been used for many decades to ship Namibian and Malawian uranium to western converters. When GoviEx's shipments commence, it will be prudent to ensure the route is well managed to maintain the ability to transport the U_3O_8 to conversion facilities to support sales to customers.

The Namibian transport corridor from Zambia through Namibia to Walvis Bay is well established and has been transporting commodities safely for many years.

19.6.1. Sea freight from Walvis Bay to Europe and the United States of America

The sea freight of the ISOs from Walvis Bay to Europe or the USA will be carried out by a scheduled liner service. The shipping of radioactive material by sea must be carried out in accordance with the rules and regulations of the International Maritime Dangerous Goods Code 2012 edition or the latest version thereof.

There are several shipping lines currently used for sea freight of U_3O_8 from Walvis Bay. These companies operate routes into European ports including Marseille Fos, and Le Havre in France. From Marseille or Le Havre, the ISO can be discharged for onward delivery to the Comurhex (Orano) Conversion Facility. Alternatively, the ISO containers could be shipped to North America for delivery to the Conversion Facilities of ConverDyn and Cameco. Alternatively, the containers could be shipped from Walvis Bay to the People's Republic of China for use in the growing Chinese nuclear energy programme.

Quotations were provided that covered transportation costs from the Project near Lake Kariba in Zambia to Comurhex in France, Cameco in Canada and ConverDyn in the USA. The costs were quoted at an average USD58 200 per 20 ft ISO container. There is assumed to be 15 tU in U_3O_8 (approximately 40 000lbs U_3O_8) in each 20ft ISO container. This results in an average transport cost of USD1.45 per pound U_3O_8 from Muntanga to final conversion destination.

19.6.2. Radiation protection

It is a requirement of the Transport Safety Regulation TD-R-1, 2005 Edition that a radiation protection programme is put in place for all transport of radioactive material. The final company chosen for uranium transportation will have to follow the regulations set out by the World Nuclear Transport Institute (“WNTI”) and publish documents that detail best practices in the nuclear transport industry. Transportation companies will have to use these WNTI-published documents in conjunction with their procedures to ensure that all regulatory requirements are met or exceeded with regard to radiation protection.

19.7. Spot and term markets

When selling a commodity dependent upon strategy and available inventory, a supplier may look to sell into the spot market or the long-term market. Spot sales are those where terms and conditions are agreed for a delivery in less than three months (spot). A long-term sale (term) is one where the terms and conditions are agreed upon for delivery in greater than 24 months and a mid-term (mid) sale is between three months and two years.

19.7.1. Market publications

In the Uranium market, two trade publications are commonly used as price reference points in U_3O_8 sales contracts: Ux Weekly published by Ux Consulting Company LLC (UxC), and the Nuclear Market Review published by TradeTech LLC. Both issue a weekly U_3O_8 Spot price and a Term month-end price (Figure 19-11).

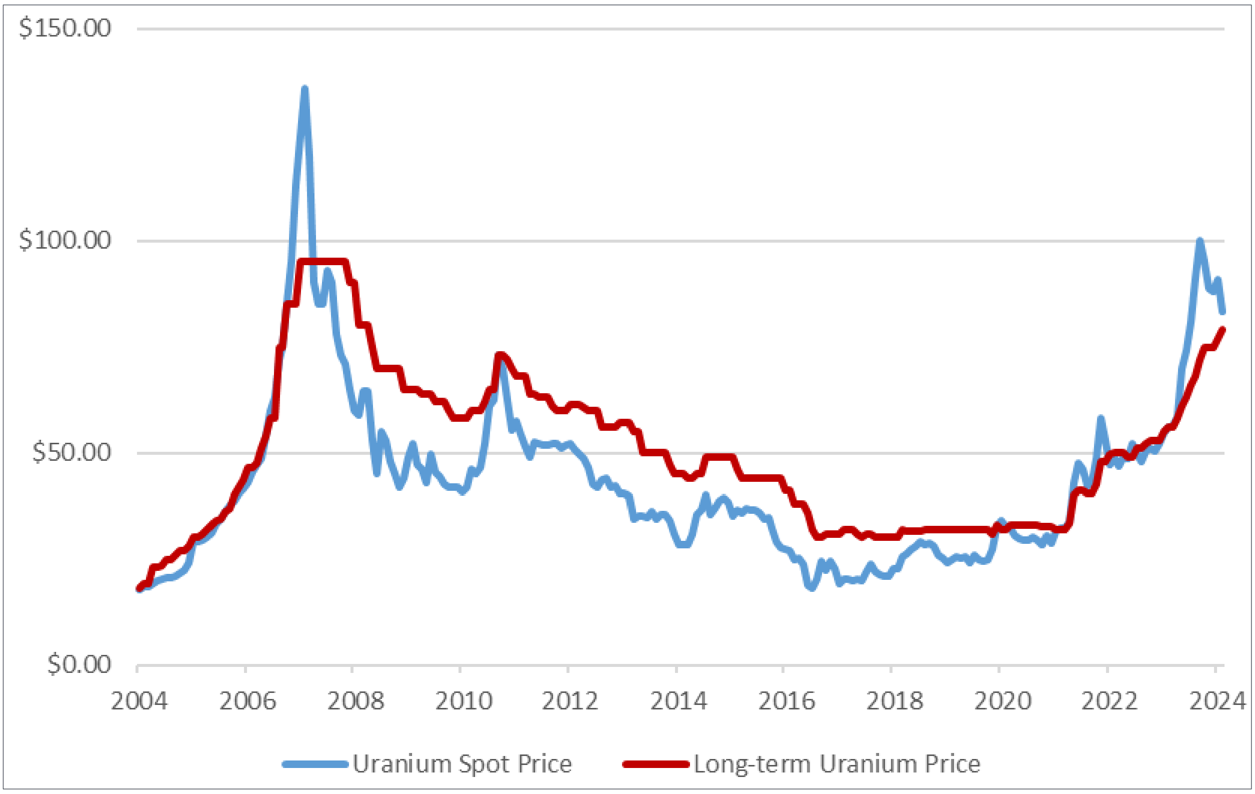


Figure 19-11: Spot versus term price (2004 – 2024)

19.8. Pricing mechanisms

There are a variety of pricing mechanisms that can be used when negotiating and concluding a contract for the sale of U_3O_8, in general, there are three mechanisms that are favoured by the industry.

1. Fixed price
2. Market-related price
3. Hybrid price.

19.8.1. Fixed price

Fixed pricing can be divided into two categories:

1. Fixed price which is not subject to escalation
2. Based on the price escalated.

A fixed price contract is where the buyer and the seller agree to a specific price, which is not escalated by inflation indices. This type of pricing has been typically used for Spot or Mid sales, however more recently, buyers are now requesting fixed prices for longer-term contracts.

Fixed pricing can be advantageous to the buyer and seller as both can easily forecast and measure cash flows, budgets and material inventory prices.

A base price escalated mechanism is traditionally used for longer-term contracts. The base price is agreed upon and fixed in a contract between the buyer and seller and is escalated in line with an agreed escalation factor, the starting date of which is also agreed. A commonly used escalation factor is the US Gross Domestic Product Implicit Price Deflator ("GPDIPD"). The GPDIPD is one measure for the US annual inflation rate and is typically used in the nuclear industry.

The base price escalated approach has an element of unpredictability, but since the commonly used escalation factor or GPDIPD is used, it is unlikely that a huge variance will be seen year on year.

19.8.2. Market-related price

Market-related pricing is usually based on the spot or term price (as published by UxC and TradeTech or an average of both) near or at the time the delivery of U_3O_8 takes place. For term contracts, it is not unusual for a seller to offer a buyer an agreed percentage discount against the term price.

Market-related pricing ensures that the purchase price will be more in line with the market conditions at the time of the purchase, but overall, it gives a greater level of uncertainty for the buyer and the seller. This uncertainty can be mitigated somewhat using price floors and ceilings, whereby the floor price protects the seller, and the ceiling price protects the buyer. The level at which the floor and ceiling prices are fixed will depend on the market conditions at the time of contract negotiations.

19.8.3. Hybrid price

In recent years buyers have looked at ways to optimise pricing mechanisms, especially for term contracts. As a result, hybrid prices have been agreed which is a combination of both the fixed price and market price mechanisms. The percentage split between the two varies and is negotiated between buyer and seller. For example, a term sales contract could have 60 % of the annual quantity delivered using a fixed price mechanism and 40 % delivered using a market-related price mechanism.

19.9. Yellow cake sales and marketing

GoviEx is predominantly an exploration and mine development company and will contract out the logistics of its uranium deliveries to specialist nuclear transport companies. However, the marketing and sales of its U_3O_8 will be performed in-house.

The GoviEx marketing team will undertake a number of activities on behalf of the Project, with an overall approach of structuring GoviEx's sales and delivery to maximise return on uranium sales through a blend of spot and term contracts, based on appropriate pricing structures.

The marketing team will develop a marketing and sales strategy for presentation to GoviEx. The intent of the marketing and sales strategy will be to establish:

- When and how much U_3O_8 will be available for sale
- The terms and conditions for such sale
- The market conditions, to leverage contract size to obtain prices favourable with respect to the market
- The appropriate length of supply contracts to balance customers' security of supply requirements with optimal timing of GoviEx production
- A list of potential customers ranging from utility end users to traders and intermediaries.

The marketing and sales strategy will be continually reviewed to anticipate and accommodate any change in market dynamics. The marketing team will work closely with GoviEx to understand the U_3O_8 production forecasts for future sales activities based on planning information received from GoviEx. Based on market demand for U_3O_8 requirements, the marketing team will approach potential customers and negotiate U_3O_8 sales on behalf of GoviEx.

Where volume, duration or price mechanism options exist within the portfolio of U_3O_8 supply contracts, the marketing team will analyse market conditions and provide a summary of such analysis to operate the contracts optimally to meet GoviEx needs.

The marketing team will provide GoviEx with a summary of all negotiations and provide an analysis and evaluation of the proposed sales. GoviEx will review to provide approval for the marketing team to proceed. Based on the approval by GoviEx, the marketing team will enter further negotiations with the customers to agree on the final terms and conditions of supply through to contract drafting, conclusion and signature.

Prior to the signature of any supply contract, the marketing team shall arrange for each contract to undergo a full legal review by a recognised lawyer specialising in the applicable international law pertaining to each supply contract as negotiated by the marketing team and the individual customers. Where applicable, the marketing team will obtain Euratom Supply Agency ("ESA") concurrence or notification as appropriate for all U_3O_8 supply contracts.

The marketing team on behalf of GoviEx will draft and negotiate a weighing, sampling, analysis and storage contract with each of the conversion facilities, and the marketing team will manage the U_3O_8 deliveries for receipt into U_3O_8 holding accounts at the conversion facilities to ensure that contractual obligations relating to the sales of U_3O_8 are met from both a quantity and lead-time perspective. In addition to the aforementioned, the marketing team will take responsibility for reconciling inventory movements in and out of the U_3O_8 holding accounts and verify the account balances against account statements issued by the operators of the conversion facilities .

19.10.Price used for discounted cash flow analysis and valuation

Based on UxC's Q2/2024 long-term contract, the forecast is outdated and is now too low based on current data points and references, the latest set used by existing producers are closer to reality on fundamentals, and lastly the referenced market reports are to some degree contradictory but remain good indicators of congruence on main factors impacting the market.

The valuation QP, Alan Clegg, has been following closely developments in the uranium sector and been in contact with North American analysts as well as uranium asset and fund managers for deeper analysis. The latest UxC and Tradetech forecasts which are significantly up, Tradetech in particular.

The QP's outlook on the uranium price is informed by the following factors and fundamentals:

- 2024 consumption closed at approximately 185Mlb, with supply at approximately 50Mlbs, leaving a deficit of 35Mlbs
- Underfeeding by utilities is dropping significantly
- Inventory re-stocking from secondary supply active but incremental supplies are small at 2 – 3 Mlbs per annum
- Required big projects are not coming on line short term and only from around 2029 onwards
- Reactor life extensions, shutdown delays and mothballed restarts are impacting at increasing rate
- Technology industry intervention in the nuclear generation industry, both direct and indirect
- 67 new reactors are in the build phase, 50 % of which will join the grid in the next three years
- If triple capacity is realised as stated by US and others then demand will increase strongly beyond 2030.
- On a macro scale, the market is going to be in imbalance for may years to come. It may reach close to balance by 2029 to 2031, but only if major projects come on line, such as NexGen Arrow, Denison Phoenix, and Kazatomprom's full ramp with a new acid plant.
- However, not long after in the mid-2030's production declines accelerate again, in particular for Rossing, Langer Heinrich and Husab in Namibia, 50 % of Kazatomprom's projects and Cameco's Cigar Lake Phase 1. So, if demand is to be met a massive supply response is needed.
- Add Russian export limitations and rising polarisation, with East staying east, potentially the entire Global South doing so, and the West or Global North staying west. Most Namibian uranium is purchased by China.
- Out to 2030, only 50 % of estimated utilities requirements are covered. Suppliers are sitting and not signing term contracts, especially if no mining right/permit has been issued. Contract flexing has all but disappeared.
- Utilities only keep 2 to 3 years' inventory at absolute minimum.

With deficit consensus among most institutional analysts at 60 Mlb to 70 Mlb by 2035 and 130 Mlb in 2040, and total consolidated deficit in the period from 2028/9 to 2040 of more than 700 Mlb, the average predicted long-term price for 2025 to 2028 is around USD90 /lb and higher than USD100 /lb by the mid-2030s among same institutions.

Long term price consensus among the QP's sources now stands at USD92 /lb with the spot floor at USD75 to USD80, but they see no return under any circumstances below USD70, and expect greater than USD100 by Jan 2026 with a median price of USD96/lb. Looking specifically at the forward term market, the 3yr price is sitting at USD94 and the 5yr price at USD10, which is expected to create a further increase in the floor in the spot market at least to USD80/lb. Similarly, this has created an indicated ceiling range with buyers of USD120/lb to USD130/lb in next 10 years which means that buyers expect the price is to go higher.

The QP's view is that for Muntanga a base price of USD90/lb should be applied to the valuation, with sensitivity analysis at USD80/lb and USD70/lb to the down side and USD100/lb and USD110/lb to the upside as this will cover the widest range of current potential scenarios.

20. Environmental studies, permitting, and social or community impact

This section presents the current status of environmental studies being done as part of the project including the environmental social impact assessment ("ESIA") process being followed. The section summarises GoviEx's:

- Approach to governance and their environmental and social management systems
- Assesses the environmental and social risks to the project.

20.1. Current status of environmental studies

An environmental impact assessment ("EIA") was prepared for the Chirundu (Njame and Gwabi) sites in 2008. This was based on baseline data collected between March 2007 and February 2008 (AFR, 2008). Similarly, an environmental impact study was prepared for the Project in 2009 by African Mining Consultants ("AMC") as part of the Denison Feasibility Study (MDM, 2009).

As of December 2024, AMC is in in the final stages of a full ESIA process that builds on the earlier studies but includes a comprehensive update of the baseline studies and assessment of the impacts based on the new project design. GoviEx is committed to developing the Project to International Finance Corporation ("IFC") standards and the ESIA process has been scoped to achieve this. The Project will result in the resettlement of a number of villages and accordingly AMC are developing a resettlement action plan ("RAP"). The potential impacts described in the following sections are drawn from the AMC work and based on SRK's knowledge of the project design. The details associated with the social and environmental impacts from the Project will not be repeated in this document but a summary is included to flag the potential risks and opportunities associated with the project development.

The potential environmental impacts of the Project are being systematically assessed using the source-pathway receptor framework. An environmental management plan ("EMP") will form part of the AMC deliverable. AMC plans to finalise the ESIA in quarter ("Q") 1 2025 and submit the report for regulatory comment and approval towards the end of Q1. The regulatory consultation process for the ESIA and RAP is expected to take approximately 6 to 12 months.

The ESIA report will describe what are effectively two phases in the project life:

1. Phase one focuses on the central Muntanga and Dibbwi East area and includes the development of the Dibbwi satellite deposit.
2. Phase two describes the development of the two additional satellite deposits, Njame and Gwabi.

20.2. Legislative and permitting context

The main laws and legal requirements applicable to the Muntanga Project are covered in Chapter 4 and are not repeated here. Of relevance to the environmental and social aspects is the approval of the ESIA and RAP and the application for, and granting of, various permits required before construction can commence. The key permits required will be granted by the Zambia Environmental Management Agency (ZEMA).

Other licences granted by ZEMA that will need to be applied for include, but are not limited to:

- Air pollution monitoring permits
- Water effluent and discharge licences and
- A waste management licence.

As part of the active exploration programme, GoviEx already holds a licence for the management of hazardous waste. The existing licence was granted on August 9, 2022 and is valid for three years. As part of the wider permitting discussions with ZEMA, GoviEx will have to apply for an extension or renewal of this licence and ensure the conditions match the operating phase of the project.

20.3. Social and environmental context

This section provides a summary of the social and environmental context with a focus on how the setting has influenced the project design and sets the context for why specific design measures have been considered. It highlights aspects that influence risks to the projects.

20.3.1.Social baseline

The social baseline information reported in this section is drawn from the information collected by AMC as part of the project ESIA baseline data collection and RAP preparation. A mixed approach was used to collect the baseline socio-economic data. Secondary data was collected using litreature reviews and primary data were collected using observations, focus group discussions, GIS field map and semi-structured questionnaires.

20.3.1.1. Stakeholder participation

Stakeholder engagement was an overarching feature of the socio-economic impact assessment process. Stakeholders were engaged before commencing the social census to explain the process being followed and to enable them to provide the input necessary to refine approaches to the study. Equally, stakeholders were engaged immediately before the property valuation exercise commenced which gave them an opportunity to ask questions and raise any concerns. To ensure effective participation in the process, separate meetings were held with women and youth groups to ensure that their voices were heard.

20.3.1.2. Population demographics

The Zambia Statistics Agency (2022) shows that the 2022 population of Siavonga District was 66 030 comprising 32 701 males and 33 329 females while in Chirundu District it was 78 780 comprising 38 215 males and 40 565 females. Between the years 2010 and the year 2022, the population of Siavonga District increased by 54 % while the population of Chirundu District increased by 66.4 %. The total population likely to be affected by the project is 2 386 comprising 1 210 males and 1 176 females. The median age is 16 years while the average age is 22 years. Of the ~2 400 affected people, 958 will be physically relocated as a result of the Muntanga and Dibbwi East project infrastructure. The phasing of the resettlement programme is described in more detail in Section 20.3.2 below.

Tonga is the most dominant ethnic group comprising 76 % of the population followed by the Goba ethnic group comprising 22 % of the population. Bemba, Lozi and Nyanja-speaking ethnic groups make up 2 % of the population within the affected project area. Protestants make up 81 % of religious groupings within the project area. Within the protestant grouping, Methodists, Pentecostals, Jehovah's Witnesses and Seventh-day Adventists are the most prevalent.

None of the communities in the project area are considered indigenous communities or groups. This important distinction is explained in the ESIA baseline study which draws from a book by Elizabeth Colson. Essentially, the influence of the country's colonial period and the relocation of the communities from their original settlements (a result of the construction of Kariba Dam) means their systems of governance and traditional culture is significantly different when compared to their historical traditions.

20.3.1.3. Economic and livelihood activities

The economy of Southern Province is primarily driven by agriculture, tourism and mining (Zambia Statistics Agency, 2014). The economy of Siavonga District is driven by tourism, agriculture and fishing activities in the Kariba dam while the economy of Chirundu District is mainly centred around agriculture, trade and activities happening at the Chirundu Border Post.

The economy of the project area is largely driven by subsistence agriculture. The major crops grown are sorghum, millet and maize. Except for Gwabi, where there are commercial farmers, the proposed mining areas are used primarily for subsistence agriculture, with 95 % of households in Muntanga, Dibbwi, Dibbwi East, Njame and Gwabi depending on rain-fed subsistence farming for their livelihood.

Households get their income mostly from the sale of crops and livestock and an average household in the project area earns approximately ZMW1 910 (USD 73) per year. 71 % of the income earned by households is spent on food. In the recent past dependency on charcoal burning has been on the increase as people look for alternative income sources away from rain-fed agriculture which has been hit by poor rainfall patterns and droughts.

20.3.1.4. Settlement patterns and houses

The settlement patterns in the project area are generally a mixture of dispersed and nucleated settlement patterns. In Dibbwi and Gwabi, the settlements are dispersed while in Muntanga, Dibbwi East and Njame, settlements are nucleated around facilities such as water points, and access roads.

The total affected project area has 365 main houses and 190 auxiliary houses. 82.2 % of the houses have one to two rooms while only 17.8 % have more than two rooms. 70 % of the houses are made of raw bricks, 75 % have mud floors and 69.2 % have iron roofs.

20.3.1.5. Land use

Land in the project area is used for agriculture, residence, commerce and natural vegetation cover. The majority is used for agriculture. Interviews conducted in the project area showed that 82 % of households use their land primarily for agriculture and 18 % for other uses such as housing and commerce.

20.3.1.6. Access to social and economic services

Based on the baseline review completed by AMC in 2023 the affected households in the project area access health services from Muntanga Rural Health Post, Manchanvwa Inland Rural Health Post, Sikoongo Rural Health Post and Chibende Rural Health Post. The most prevalent diseases in the project area are respiratory tract infections, malaria and diarrhoea. The human immunodeficiency virus ("HIV") prevalence rate in the project area is much lower than the provincial prevalence rate of 12.4 %. However, the prevalence rate in Gwabi is higher than the prevalence rate in these other areas.

Members of the community living in the project area access secondary education from Sikoongo, Chaanga, Gotagota, Chirundu, Kapulurila, and Mandenga Secondary Schools. Primary education is accessed from Sikoongo, Muntanga, Hachibozu, Manchanvwa Inland, Malabanyika, Chiwena, Dambilo, Mandenga, Kamutengo, Mangwalala, Hachibbuba, Dibbwe,and Chilomba Primary Schools and the recently built Mutuba Community School.

Of the population currently in school, 86 % attend primary school, 11.8 % attend secondary school and 1.6 % attend tertiary education. Njame has the highest overall school attendance and Dibbwi East has the lowest. Overall, there are more males than females attending school in the project area. 29.7 % of the eligible adult population have completed primary education, 3.8 % have completed secondary education and only 1.6 % have completed tertiary education.

The litreacy rate for the population aged five years and older within the project area is approximately 38 % while the litreacy rate for the population aged 15 to 24 years is 58 %. For both the population aged five years and older and the population aged between 15 to 24 years, the percentage of males who are litreate is higher than the females.

There is no market located in Muntanga, Dibbwi, Dibbwi East, and Gwabi. There are only isolated stalls within people's yards which sell limited merchandise. The nearest market for Muntanga, Dibbwi and Dibbwi East is Chaanga Market. GoviEx recently completed the construction of a market shelter for the Njame community about 300m east of the high-tension electricity pylons on the D500 road. The nearest market for Gwabi is the Chirundu Market located right within Chirundu Central Business District ("CBD"). Approximately 59 % of households in the project area sell their agricultural products such as maize and sorghum to mobile traders who camp at strategic places within villages, 12 % of the households sell their agricultural merchandise within nearby local markets, 5.1 % sell at local markets within the Chirundu and Siavonga Districts.

Common sources of water in the project area are communal boreholes, shallow wells, rivers and streams. The main source of water in Muntanga, Dibbwi, Dibbwi East and Njame are communal boreholes. More than 80 % of the people in these areas get water from boreholes. Approximately 68 % of the people living in Gwabi get their water from the Kafue River. Approximately 49.7 % of households in the project area use pit latrines, 0.4 % use flushable toilets and 49.9 % practice open defecation.

20.3.1.7. Employment levels

The project areas in both Siavonga and Chirundu Districts offer very few employment opportunities due to the lack of economic activity in the area. Data collected in the project area show that only 4.2 % of the available labour force is employed, and of these 19 % are women and 81 % are men. 90 % of the available labour force in the project area does not have employable skills. Out of the 10 % classified as having employable skills, only 25 % went through formal training.

20.3.1.8. Gender

Gender inequality in education, access to economic activities and decision-making presents a challenge in the project area. There are more men who have completed grade 7, grade 9 and grade 12. There are more men who participate in paid economic activities than women. 91 % of household chores such as fetching water and taking care of children are performed by women.

20.3.1.9. Vulnerable groups

3.5 % of the population within the defined buffer zones live with some form of disability. 3.1 % of the population were reported to suffer from some form of chronic illness and 36 % are female-headed households. Women and youths are vulnerable due to their exclusion from participation in community decision-making.

20.3.1.10. Affected households

There are a total of 491 affected households living within the defined buffer zones in each project area. Njame has the highest number of affected households and Dibbwi has the least number. A total of 692 fields measuring a total of 6 372.356ha are affected. Of the total hectarage, Dibbwi has the lowest hectarage of affected fields while Muntanga and Dibbwi East have the highest hectarage.

20.3.1.11. Perceptions about the project

92 % of the affected households want the project to be implemented. Other stakeholders such as government departments and traditional leaders want the project to be implemented but insist this must be accompanied by comprehensive mitigation measures. Tour operators and lodge owners downstream of Gwabi on the Kafue River have expressed concern over the potential of the project disrupting their business activities if the Kafue River is impacted by project activities. They expressed concern over project activities driving away wildlife which is critical to the tourism industry. This potential impact is related specifically to the Gwabi deposit and is not a concern for the first phase of the project focused on Muntanga and Dibbwi.

20.3.1.12. Potential impacts and mitigation measures

Potential negative impacts include loss of land and immovable property; loss of livelihoods; community detachment from existing social groups; closure of access routes due to mine infrastructure; community exposure to increased levels of radiation; accidental spills or leaks in the process; loss of business by tour operators and lodge owners; and pressure on existing social and economic services.

The potential positive impacts include the creation of local job opportunities; local economic development of the project area; improved access to water and sanitation for affected members of the community; an increase in revenue for both local central governments and improved household income. Benefits will include better housing for project affected persons through the provision of improved replacement houses made of more permanent materials including concrete block and mortar and iron roofing.

Proposed measures to mitigate the negative impacts and enhance the positive impacts include:

- Adequate compensation for the affected households
- Training to improve access to job opportunities
- The creation of a livelihood restoration programme, local employment targeting unskilled labour
- Ensuring that resettlement sites have good access to natural resources
- Establishing a safety buffer between active mine operations and the settlements
- Implementing procurement policies which encourage locals to participate in the supply of various goods and services
- Implementing pollution control programmes necessary to enhance occupational health and safety and public health and safety.

20.3.1.13. Cultural heritage and archaeology

Some known archaeological and palaeontological sites in the region include Ing'ombe Ilede and the Chirundu Fossil Forest. An early archaeological study at Muntanga and Dibbwi was conducted in October 2006. Systematic transects plus examinations of old exploration pits and ant hills revealed no existence of archaeological sites or artefacts, although a large area of precipitated iron was sampled at UTM 35 659667E 8192636N or 16º 20' 61'' S 28° 29' 68'' E. Two sacred cultural sites were identified: Hapepe and Malende, located 3km north of Matuba Community School and 5km to 6km north past Changa Village on the Machinga Stream. These sites are used for worship during the suspected occurrence of droughts, but they will not be affected by the Project activities.

A similar study was conducted near Njame and Gwabi during October 2007. There is a known abandoned Manyuchi village near Njame, but of a recent age and not considered to have any archaeological significance. Three burial grounds were identified near Njame; however, these are not ancient and fall outside of heritage legislation. According to information gathered during interviews, no chief has been buried at any of the sites.

The more recent survey conducted in 2023 revealed previously overlooked sites that show a long history of human activity in the area. A total of 11 new archaeological sites and 26 individual graves from seven graveyards (burial sites) were recorded in the proposed project area.

A further two sacred or ritual sites for rain making purposes exist in the western part of the project area. The heritage sites documented in the area are of local significance. This contrasts with Ing’ombe Ilede and Chirundu Fossil Forest sites present in Chirundu district that rank as National Monuments.

The Project may have negative impacts to archaeological heritage resulting from site clearance and the construction of linear infrastructure. Mitigation measures aim to reduce the negative impacts that may threaten the integrity of heritage resources. In cases where the destruction of the archaeological resources is inevitable, rescue and salvage operations will be carried out in line with the Cultural Heritage Management Plan.

20.3.2.Resettlement

The Project will require the resettlement of local communities at all five project sites. AMC are in the process of finalising the detailed resettlement plan. Full community baseline and household surveys have been completed and the asset inventory cut-off date, 19th November 2024, has been communicated to the people affected by Phase 1 of the Project, which involves those impacted by mining at Muntanga, Dibbwi East and Dibbwi.

Phase 2 will impact the communities and farmers at Njame and Gwabi. These groups have been included in all communications to date and have been comprehensively surveyed. However, due to the project schedule, these areas will only be disturbed five to seven years into the project life and therefore will only be resettled later in the project process.

The focus of this Feasibility Study is the Central Area of Muntanga and Dibbwi East. The project will impact 157 household heads in Muntanga and 32 household heads in Dibbwi East. Of the affected household heads, 140 are male and 49 are female. The total number of people including household heads affected in Muntanga are 784 comprising 412 males and 372 females. The total number of people including household heads affected in Dibbwi East are 174 comprising 94 males and 80 females.

These households will need to be relocated to pave the way for the project and consequently will lose their current immovable assets such as dwelling houses, fields, economic trees and other structures. The RAP being prepared for the project clearly lays out the compensation process being followed to ensure compensation for any loss of assets, land and livelihoods.

In addition, the project footprint in Muntanga and Dibbwi East has 5 graveyards and approximately 50 individual graves. The closest graves to the open pit and Waste Rock Dump (WRD) footprint are approximately 9 graves. These graves are a minimum of 100 m from the boundary of either the open pit or WRD in Muntanga. There are no graves within 100 m radius from the boundary of either the open pit or WRD in Dibbwi East.

At the time of preparing this FS document, discussions with the project affected people are still in progress and the exact number of graves to be relocated and the relocation sites is still being negotiated.

20.3.3.Environmental baseline

20.3.3.1. Climate

The climate of the Project is described as tropical wet and dry with distinct wet and dry seasons. The wet-hot season is from November to March, with the highest rainfall occurring in February. The mean annual rainfall is recorded as 720mm. The dry-cool season is from April until October. There is a notable variation in the temporal and regional distribution of rainfall. During the dry-cool season maximum temperatures range from 23°C to 40°C and minimum temperatures range from 6°C to 28°C. During the wet season the maximum temperatures range from 22°C to 46°C and the minimum temperatures range from 20°C to 38°C. The highest maximum temperature that has been recorded at the site was 46°C and the lowest minimum temperature that has been recorded is 6°C

Climate change and greenhouse gas accounting are discussed further in Section20.8and Section 20.9. There are two main features of the climate that are relevant to the project design:

1. The relatively short and limited rainfall season. With annual precipitation less than 800 mm per year, the area is water stressed with few perennial surface water sources. In addition, the rainfall tends to be in intense tropical storms resulting in rapid flash run-off. This provides challenges in managing erosion during construction and for rehabilitation efforts.
2. The long dry season that can result in excessive dust generation and competition for limited water resources, particularly surface water.

As part of the project design, SRK have undertaken a climate change risk assessment, the findings of which have been used to inform engineering designs.

20.3.3.2. Soil

The iron- and aluminium-rich soils within the Project area are extremely weathered, nutrient deficient and have poor water retention capacity. Gleysols, sandy loams and sandy soils are the predominant soil types, and lateritic horizons are common. Regional land classification based on the USDA standards indicates medium to low potential for sustainable use of these soils. Despite the relatively poor quality soils, communities use specific areas for cultivation of cotton, maize, sorghum and a variety of green vegetables.

20.3.3.3. Land Use

Existing land cover in the project area is a combination of Mopane or Miombo woodland, bare rock outcrops, small agricultural fields and degraded grassland and scattered villages. Land use associated with the villages is mainly cotton, sorghum, millet, and maize farming, with minimal animal husbandry in the form of pig, goat or cattle herding, and poultry. Fertile flood plain zones adjacent to watercourses is generally cultivated to produce vegetables or fruit during the wet season. There is evidence of timber cutting activities and charcoal production, particularly in the Muntanga area. Exploration activities are the only major form of industrial land use in Muntanga. Housing is typically built using traditional methods and materials, with earth floors, mud brick walls and grass thatch or iron sheets roofs, with access to these resources a key consideration in the location of settlements.

20.3.3.4. Air quality

An air quality baseline has been developed as part of the ESIA process. Based on the results obtained, all the 24 hours average for TSP and PM10 results obtained are below the ZEMA guideline limit of 70 µg for PM10 and 120 µg/m^3 for TSP. The 98th percentile for all the sampled points did not exceed the ZEMA guideline limits. PM10 was within the guideline limits provided by World Health Organization (“WHO”).

Variation in air quality does occur as a result of bushfires, charcoal burning and shifting cultivation practices during the dry season which generate smoke and dust. Temperature inversions trap smoke near ground level in the cooler months. Field observations indicate that few vehicles travel along the Muntanga road through the license area and therefore exhaust emissions are localised and disperse rapidly. Dust levels increase around exposed surfaces and roads.

Carbon monoxide, carbon dioxide, oxygen and sulfur dioxide were sampled at five locations using an Altair 5X. The air quality baseline includes an extensive review of available weather data focussing on the last five years.

A dust dispersion model has been developed for the project and demonstrates that with mitigation, dust emissions from the proposed mining operations will be highly localised in extent and decrease with distance away from the mine. Key description of the impacts are as follows:

- Predicted 99th percentile 24-hour average for TSP and PM10 concentrations at discrete sensitive receptors are expected to be compliant against the 24-hour Zambian ambient air quality standard of 120 µg/m^3 and 70 µg/m^3 respectively
- Predicted period average PM10 concentrations at discrete sensitive receptors are expected to be below the 24 hours average WHO Guideline of 45 µg/m^3.

20.3.3.5. Noise and vibration

A noise and vibration baseline survey was conducted across the project areas. Current noise levels in the Project area are very low owing to the remote location of the site and the absence of active and industrial activity. Baseline noise is typical of the rural setting and linked with domestic sources.

Similarly, the vibration baseline recorded very low levels of vibration associated with domestic activities. However, the baseline did pick up other more substantial seismicity. The survey team attribute these readings to the location relative to the African Rift Valley and fluctuating rock stress due to changing loads from Lake Kariba related to changing water levels.

Although blasting in the open pits may be audible, the limited frequency and short duration means this is unlikely to be a significant source of nuisance to communities that will have been resettled some distance from the operations. A complaints register will be set up to assess the impact of noise on the surrounding community.

20.3.3.6. Traffic

AMC have carried out a traffic survey to establish the existing baseline. Current traffic volumes are minimal. Increases in traffic have been modelled based on estimated trip generation as a result of the construction and operational activities.

The Project will significantly increase the volume of traffic in the site vicinity. The main impacts are associated dust generation and safety to local communities especially along the access road from the D500. The road national network, while in need of improved maintenance, has excess carrying capacity and the additional project traffic will not significantly impact road capacity.

20.3.3.7. Radiation

Radiation surveys were conducted as part of the overall baseline data collection. This again built on previous baseline data collected in 2008. The results showed significant variation across the sites with one anomalous reading attributed to high levels of potassium with mineralised uranium rocks. The majority of the sites assessed had a calculated annual dose rate of less than 1mSv/year from terrestrial radiation.

Based on a conservative occupancy time of 2 000 hours for all areas of the site, the survey indicated that there are areas where workers potentially can incur a radiation dose above 1 mSv/a, based on the maximum potential exposure calculated, but well below the 20 mSv/a limit for workers in the industry. As per International Standards Organisation, areas with less than 1 mSv/a are classified as an ‘Uncontrolled’; 1 mSv/a to 5 mSv/a is classified as ‘Supervised’ and 5 mSv/a to 20 mSv/a is ‘Controlled’. In reference to the abovementioned limits any worker that will incur a dose of 1 mSv/a or greater will be classified as an Occupational Exposed Person (“OEP”) and will form part of the Dose and Health Register.

There is potential to impact the environment through the accidental release of radioactive materials into soils, surface water, groundwater or air that may produce ionising radiation that may be harmful to health, such as dust, water, transport spills. These impacts will be managed through the implementation of the ESMP and specifically the Radiation Management Plan.

20.3.3.8. Hydrology

Watercourses in the Project area are generally seasonal, only flowing in the rainy season and usually through flash flooding in channels. Existing surface waters are characterised by circum-neutral pH, low to moderate total dissolved solids (“TDS”) (150 mg/L to 200 mg/L) and high faecal coliform counts.

A policy of zero discharge is planned for all mine and process contact water. Process water in the plant is recycled with a small volume discharged to a ‘bleed water pond’ for evaporation. Rainwater runoff from mine areas will be designated as contact water and contained. Suspended solids from otherwise clean water will be settled prior to discharge to local water courses or used as process makeup water.

The project will employ ex-situ pit dewatering to minimise the volume of contact water to be treated and managed. Residual and meteoric waters from the open pits will be pumped to settlement ponds and PCDs. This will allow for settlement of solids and depending on the water quality, excess water will be discharged to the environment. Surface water run-off from the overburden dumps will be collected in toe drains and directed to sedimentation ponds for settlement of suspended solids. Storm water cut-off drains will be constructed around the perimeter of the ROM pad, spent ore dump and plant areas to collect run-off and divert drainage to sedimentation ponds. Non-contact water will discharge to the diversion system. Within the process plant, all spills and wash water from maintenance activities will be returned to the process circuit. Oil traps will be installed in the workshop drainage system to treat all effluent prior to release. Sludge from the oil traps will be treated as a hazardous substance and disposed of at an approved site.

20.3.3.9. Hydrogeology

Groundwater in the Project area is used extensively for domestic and potable water. Groundwater is circum-neutral and fresh (non-saline) with elevated TDS (typically 300 mg/L to 600 mg/L). Groundwater quality historically analysed near Muntanga and Dibbwi showed generally good water quality with most parameters within WHO guidelines with the exception of iron and manganese. Certain parameters occasionally exceeded WHO guidelines including arsenic, aluminium, manganese, total chromium and selenium. Radiological analyses results showed radium, uranium and thorium concentrations in drinking water that were compliant with guideline limits. There were some WHO guideline exceedances of gross alpha concentrations in the groundwater samples.

Pit dewatering will lower the water table in the vicinity of the open pits. This has been modelled and the impacts carried through in the ESIA report. The wells in the neighbouring villages will be monitored to assess impacts on water levels and quality. Surplus water from the dewatering programme will be released to environment after treatment for removal of solids.

20.3.3.10. Surface water quality results from 2023-24 test programme

A series of surface water samples were collected as part of the environmental baseline. The results were evaluated to establish compliance with the KC guidelines limits. The following results are highlighted:

- The lower reaches of the Lusithu and Kasungu streams have a greater sediment load, which is reflected in TSS levels ranging between 164 mg/L to 266 mg/L. This is likely after storm events as these streams mainly flow only after storm events
- The Lusithu stream was recorded as having low salt and hence TDS concentration levels, with marginally elevated Fe concentrations. There is no discernible change in water quality along the stream
- The Kasungu Stream was found to have elevated levels of metals, Al, Fe, Pb and Mn but with neutral pH and salt concentrations within the catchment limits. This result would need verification, and the river should be resampled and form part of regular monitoring
- The Machinga stream has high concentrations of NO_3-N (6.53 mg/L) and Na (25.7 mg/L), which are slightly above the KC guideline limits of 6.0mg/L and 20mg/L, respectively.

20.3.3.11. Groundwater quality results

The 2023-24 borehole sampling results were evaluated to establish compliance with the ZDW and ZED standard guidelines limits. The following results are highlighted:

- Fe, Mn, SO_4 and electrical conductivity ("EC") exceed the recommended ZED guideline limits across all deposits
- The SO_4 level across all deposits except at Muntanga is relatively elevated by comparison with the other salt ion concentrations
- Muntanga has overall better water quality in comparison to the other deposits, with an EC of less than 347 µS/cm. Boreholes at Muntanga, Dibbwi, and Njame have lower concentration levels, which gradually increase sequentially from Muntanga to Dibbwi, Dibbwi East, and Njame, with a steep increase at Gwabi, especially at the GWDTH 1657 borehole. The artesian water at the GWDTH 1657 borehole is likely deeper older water.
- The water quality at Gwabi is poor, with elevated levels of SO_4, Na, Ca, Mn, Fe and gross alpha and beta readings. The HTH boreholes drilled into the orebody returned poorer quality than nearby community wells. The poor water quality may be related to its low elevation, local agricultural activity and the influence of the orebody geology
- Most trace elements are within the ZED and ZDW guideline limits except for Mn and Fe. Fe-rich goethite is known to host a significant portion of uranium mineralisation in the project area. The P and Mn concentrations are high because the uranium mineralisation in Karoo deposits is often associated with phosphorus and manganese mineralisation (Cairncross, 2004)
- According to the WHO guidelines, the ingestion of Mn-Fe water does not pose a significant threat to consumer health. However, elevated phosphorus levels in groundwater can potentially lead to adverse health effects, particularly kidney diseases
- There are some discrepancies between the two sample sets (December 2023 and May 2024), and careful sampling and laboratory analysis are required as part of the next round of sampling.

20.3.3.12. Flora and fauna

The Project area does not fall within, or abut, any protected areas or Key Biodiversity Areas ("KBAs"). Figure 20-1 show the potential mine areas in relation to designated and formally protected areas.

The natural vegetation in the Project area is savannah grassland with riparian forest along stretches of stream and rivers. Three main ecosystems were found; namely Miombo/Mopane woodland and shrub savanna with intermittent semi dambos at all sites except for Njame. Generally, findings show that the proposed sites are largely dominated by Savanna woodland and shrubs, comprising mainly of miombo biome with typical species of Isoberlinia, Julbernardia and Brachystegia type and Mopane shrubs.

Historical baseline surveys near Muntanga and Dibbwi included an assessment of uranium bioavailability in vegetation was conducted on root, stem and leaf samples from various tree species. The highest uranium concentration occurred in the species Burkea africana (4.11 mg/kg) which may only have been related to concentrations in the soil.

There is a diverse and abundant number of insects in the Project area, including dragonfly, wasp, bees, caterpillars, crickets, grasshoppers, termites, mosquitoes, ants, butterflies, and moths. Mammal species are relatively low in abundance and diversity, although field surveys near Muntanga and Dibbwi indicate that mammals such as bushbuck, klipspringer, duiker, bush pigs, aardvarks, jackal, hares, baboon, vervet monkeys, and greater kudu are present in the Project area. These mammals are classified as “Least Concern” on the IUCN Red List. Elephants are rarely seen, but are active in the area. The most recent baseline has identified an elephant migratory corridor that crosses between the Muntanga and Dibbwi pits. This will require additional mitigation measures and is discussed further in the ESIA.

The Kafue River provides a habitat to a small number of crocodiles, although there has been a decline in numbers owing to poaching and river sedimentation from anthropogenic activities. Smaller reptiles, such as terrapin, turtles, snakes and lizards, are also present. No threatened amphibian species have been identified in the mining lease area. Important habitats within the mine area for amphibian diversity include moist grasslands, ravines and temporary pools in Miombo woodland.

To a large extent Zambia’s avifauna is that of the Central African Plateau. Bird distribution is closely associated with the various habitats on the site. In some areas the miombo woodland noted above is replaced by other woodland types, dry forest or thicket. Along watercourses there may be riparian forest, or in wetter areas in the north, wet evergreen forest. Most of these habitats have a distinctive set of birds. The avifauna of Zambia includes a total of 779 species, of which one is endemic, one has been introduced by humans, and four are rare or accidental. Eleven species are globally threatened.

The Avifauna survey covered 63 survey points across both Natural and Modified habitats over two seasons (wet and dry) at all the five proposed Project satellite sites. A total of 155 bird species, with one critically endangered and two vulnerable species in the IUCN Red List of threatened species were recorded during the surveys.

Of the bird species recorded, 34 % were migratory and 66 % resident. Based on the critical habitat assessment and screening conducted, the proposed project sites (Muntanga, Dibbwi East, Dibbwi, Gwabi and Njame) did not meet any of the criterion and thresholds to qualify as a critical habitat for bird life. A precautionary approach is however recommended, in light of the diversity of species documented, particularly during the processes of the proposed project that include site clearing of habitats. Paleoarctic migratory birds have been identified in the Mutulanganga Local Forest, a sensitive habitat located 20 km east of Muntanga.

Site clearance and the removal of vegetative cover during the Project pre-construction phase will affect indigenous woodland areas (including rejuvenating miombo woodland and riparian forest) and part of the farming land for the local community. Despite a certain loss of terrestrial and aquatic flora and fauna, site clearance is expected to have little impact on species number or diversity in the project area. GoviEx will only conduct clearance where necessary and will implement a revegetation programme as part of its environmental management plans. In order to conserve resources, the local community will be given the opportunity to use non-commercial timber free of charge for construction materials and firewood. In addition, the mine will aim to stockpile vegetation and topsoil for use in progressive rehabilitation.

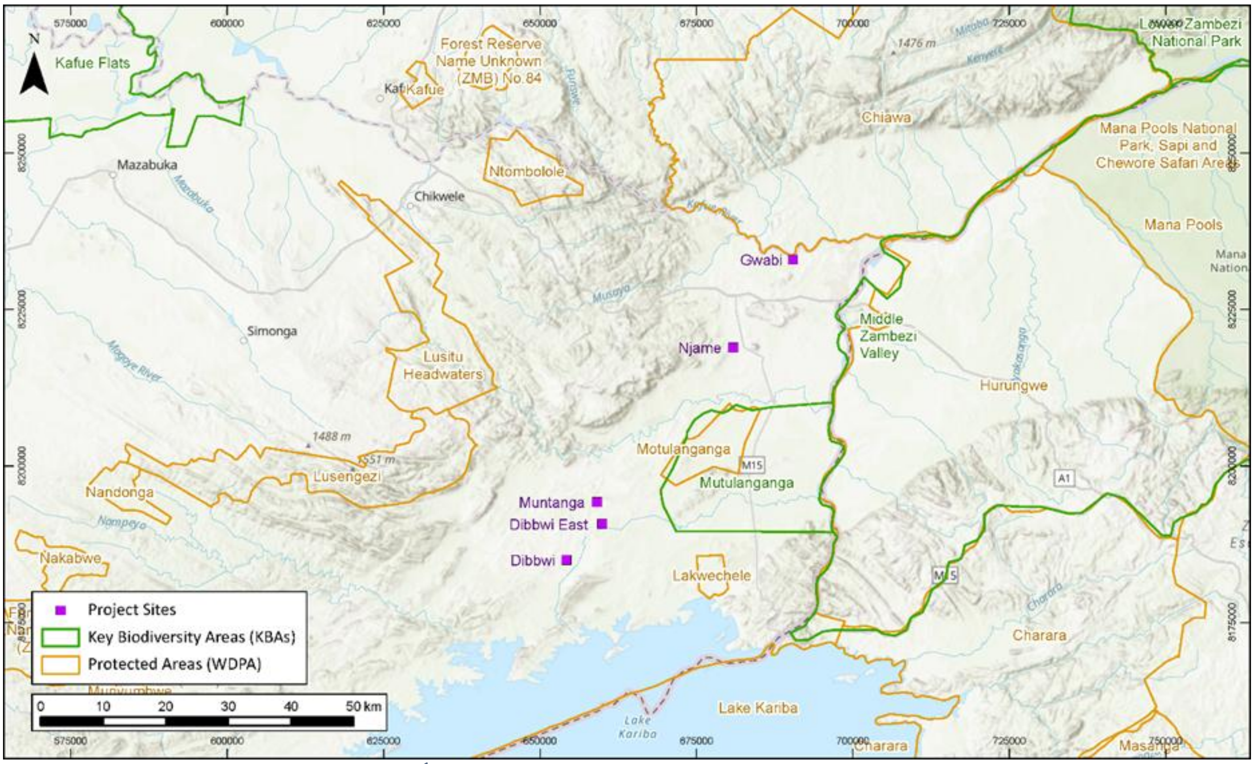


Figure 20-1: Protected areas and KBAs[1] in relation to the project sites

20.4. Approach to social and environmental management systems

20.4.1. Corporate approach

GoviEx is committed to the application of policies, strategies and practices that treat people and the environment with respect while pursuing the underlying business objective of creating value. GoviEx's commitment to sustainable development is captured in a series of Corporate Policies, Charters and Mandates. These set the framework from which more specific ESG policies, strategies, and management system have been developed. These documents and commitments are available on the GoviEx corporate website.

GoviEx's ESG focused policies include:

- Environmental policy
- Socio-economic development policy
- Stakeholder engagement policy
- Human rights policy
- Health and Safety policy
- Radiation Safety policy
- Diversity and Inclusion policy
- Child Labour policy and
- Anti-Slavery policy.

20.4.2. Environmental and social management system

A management system framework was prepared by GoviEx in 2021 to support the development of its internal governance structures for the management of environmental, social, health and safety matters and facilitate the achievement of its stated corporate values and responsibilities.

[1] Key Biodiversity Areas Partnership (2024) Key Biodiversity Areas factsheet: Mutulanganga. Extracted from the World Database of Key Biodiversity Areas. Developed by the Key Biodiversity Areas Partnership: BirdLife International, IUCN, American Bird Conservancy, Amphibian Survival Alliance, Conservation International, Critical Ecosystem Partnership Fund, Global Environment Facility, Re:wild, NatureServe, Rainforest Trust, Royal Society for the Protection of Birds, World Wildlife Fund and Wildlife Conservation Society. Downloaded from https://keybiodiversityareas.org/ on 2 Dec 2024.

The framework describes the expected structure and content of an environmental and social management system ("ESMS") and an occupational health and safety management system ("OHSMS") that meet the requirements of the following standards:

- International Standards Organisation ("ISO") 14001 Standard (ISO 14001:2015) and 45001 Standard (45001:2018)
- International Finance Corporation ("IFC") Performance Standard 1 and
- Towards Sustainable Mining ("TSM").

GoviEx is developing the required management systems steadily over time, in parallel with project development timelines and preparation of the supporting management plans as identified in the ESIA and ESMP. The company already has in place stakeholder engagement plans, communication plans and a grievance mechanism. These will be reviewed, modified and adapted to account for issues that are anticipated as the resettlement programme is implemented. All engagements with project stakeholders are recorded and captured in a commercial stakeholder database, Simply Stakeholders[2]. This includes community interactions and any issues and complaints. These are recorded and treated in a similar manner to those received via the more formal grievance mechanism.

The company is using the ONYEN ESG reporting software to record and track its ESG performance at a corporate and site level. This brings transparency to their environmental and social reporting and helping the company align with the IFC Performance Standards and report against the Sustainability Accounting Standards Board ("SASB") and Global Reporting Initiative ("GRI") reporting requirements.

The impacts identified in the ESIA report will be managed through the implementation of appropriate management measures captured in the ESIA report and the ESMP. GoviEx recognises the management measures will need to reach and benefit all levels of society so any societal inequalities are not exacerbated, community dependency on the project is minimised and support is given to social transitioning at closure.

The robustness of the supporting management plans, along with implementation, assurance and continual improvement functions of the ESMS, are fundamental to enabling the successful implementation of management measures by GoviEx, its contractors and sub-contractors. A key part of the ESMS is the ongoing monitoring to confirm whether the impacts identified in the ESIA are significant and evaluate the effectiveness of control measures in place. The system also allows the company to evaluate the need for additional measures to drive continuous improvement.

20.5. Environmental and social management plans

The following environmental and social management plans will be implemented for the Project and reviewed on an annual basis:

- Resettlement Action Plan – An initial RAP was developed in March 2009 targeting the communities affected by relocation. This process did result in raised expectations as well a degree of scepticism given the original project stalled. AMC are in the process of finalising and communicating a new RAP programme. This is described in more detail in section 20.3.2
- Occupational Health and Safety Plan – This plan describes the measures to manage all aspects of employee health and safety during mining activities
- Environmental Monitoring Plan – This plan describes measures to monitor air, soil, surface water, groundwater, vegetation that may be affected by the Project
- Radiation Management Plan – This plan consists of a series of plans required by the Zambian government to control and manage all aspects of radiation associated with a uranium project. These plans include Radioactive Waste Management, Storage and Transport, Accidental Spills Management, Community and Worker Training, Hazard and Safety Assessments
- Water Management Plan – This plan identifies all aspects related to water management for the Project. It focuses on identifying methods of conservation, re-use and re-cycling to minimise consumption of water resources by the Project
- Handling and Storage Plan – This plan deals with management actions for handling and storage of all materials onsite as well as spills management activities
- Waste Management Plan – The plan describes the types of waste that will be generated onsite and the management of these wastes. Waste management principles involved in the plan are minimisation, re-use or recycle.

[2] https://simplystakeholders.com/company/

- Emergency Response Plan – The plan identifies the original structure of the ERP which will be updated by GoviEx during construction. The plan will identify all emergencies that may occur onsite and identify measures for their management
- Conservation and Vegetation Plan – This plan focuses on the development of conservation areas and a sustainable programme with the local communities to manage their local resources
- Progressive Revegetation and Rehabilitation Plan – The plan will identify and schedule all areas that are likely to require revegetation through the mining operations. The plan will monitor the progress on these activities
- Sustainable Development Plan – The plan is designed to identify social development projects that can be integrated into a schedule of activities where GoviEx can target assistance
- A Mine Decommissioning and Closure Plan - This describes the activities required to plan for and cost an appropriate closure programme. The plan will be revised every three years over the life of the mine with increasing detail and confidence in the costing as the mine come closer to the point of closure. The closure plan will align with the current ICMM Integrated Mine Closure: Good Practice Guide.

20.6. Geochemistry

20.6.1. Overview

A geochemical characterisation study was undertaken to quantify the acid generating and metal leaching ("ARDML") potential of materials likely to be exposed during the mining process. The processes of acid generation and metal leaching can operate independently, although the development of acidic conditions enhances the leachability of some metals. ARDML may occur naturally in sulfide-bearing rock strata, but is commonly accelerated by mining activity, which increases the likelihood of exposure of sulfide minerals to oxygen (air) and water, effectively accelerating natural weathering processes.

20.6.2. Previous and Concurrent Studies

In 2009 African Mining Consultants Ltd ("AMC") (AMC, 2009) coordinated an Environmental Impact Assessment ("EIA"). Acid base accounting ("ABA") was undertaken on ten waste rock and ten ore samples from the Muntanga and Dibbwi deposits. The results showed that eight of the ten waste rock samples were likely to be net acid generating and one had uncertain acid generating potential however total sulfur was reported below 0.3 % in all samples and the report concluded that long-term sustained acid generation was unlikely.

In 2017, a project was undertaken to assess the geochemical and bioreactivity of future mining operations at Muntanga (van Aardt, 2017). Six waste rock samples from Dibbwi and six from Muntanga were analysed using XRF alongside low grade ore and surficial samples. A subset of samples also underwent X-ray diffraction ("XRD") and a 10:1 leaching test followed by ICP-MS analysis. Bulk geochemistry of the materials was reported to comprise aluminium, silicon, phosphorous, sulfur, potassium, calcium, chromium, magnesium, nickel, copper, arsenic, zinc, selenium and molybdenum; trace amounts of vanadium, rubidium, strontium, lead, titanium, thorium and uranium were also reported. Parameters reported in the short-term leachates included aluminium, silicon, potassium and low concentrations of vanadium, lead and uranium. One Dibbwi sample was an exception to this and showed high uranium mobility.

In 2024, an updated EIA including geochemistry baseline was undertaken by AMC alongside the FS. The study involved ABA testing of 38 samples comprising overburden, hanging wall, footwall and ore materials. The results of this study are presented in Geostratum (2024a). Sampling from this study was poorly constrained and QAQC was not undertaken so the results are at best semi-quantitative.

20.6.3. Geology

Project geology is described in Section 7.

20.6.4. Hydrogeochemistry

Water quality monitoring has been periodically undertaken historically across the Project site since 2008, and further water quality sampling was undertaken by AMC in 2023/24 for the EIA and opportunistically by SRK during the hydrogeological assessment for the FS. Full baseline water quality data is reported in Geostratum 2024b and the results of the water quality sampling undertaken in 2023/24 are discussed in Section18.3.5.

20.6.5.Sample acquisition

20.6.5.1. Waste rock

SRK used the exploration drill core database provided by GoviEx to select samples for the geochemical characterisation study. Leapfrog® Geo 4.0 was used to visualise drillcore data in 3D and enable evaluation of core materials based on lithology, location and depth. The USD 70 /lb U_3O_8 pit shells were used to select drill core intercepting the pits and where assay data was available a cut-off grade of 85 ppm uranium was applied, and a range of sulfur assay values were targeted. As the project has since developed, radiometric ore sorting has been introduced so that ore material with a uranium content of approximately 40 ppm and above will report to the HLF for processing.

The actual selected samples were limited by drill core availability and the condition of the drill core for sampling. Older drill core was not selected due to the extent of weathering. This resulted in some of the samples having to be selected without any existing assay data or from drill core located just outside of the pit shells. Sample collection was undertaken by the GoviEx onsite Exploration Manager. A total of 39 samples were selected and four laboratory split duplicate samples were generated for QAQC purposes.

20.6.5.2. Spent ore

Samples representative of the spent ore materials were generated by Mintek Laboratories from the 6 m columns initiated during the metallurgical testwork:

- Composite 1 (Dibbwi East Oxidised)
- Composite 2 (Dibbwi East Reduced)
- Composite 3 (Muntanga + Njame)
- Composite 4 (Gwabe Oxidised)
- Composite 5 (Dibbwi Main Oxidised)

20.6.5.3. Ore sorter rejects

Samples representative of the ore sorter reject materials were not available for geochemical characterisation.

20.6.5.4. Gypsum waste stream

During the metallurgical testwork, Mintek Laboratories generated two samples representative of the gypsum waste stream which will report to the gypsum pond from the processing plant:

- pH 2 gypsum solids
- pH 4 gypsum solids.

The first gypsum was generated at pH 4 (IX eluate), and the second sample was at pH 2 (nano filtration) after acid neutralisation. The pH 2 sample is likely to be more representative of what will occur at the process plant.

20.6.6.Testwork Methods

The static geochemical test methods used for the Muntanga FS are designed to address the bulk geochemical characteristics of the samples. Geochemical testing on the waste samples aims to assess the potential of these materials to generate acid or release metals and other solutes in contact waters. “Static testing” is a general term describing those analytical methods applied to characterise acid generation and metal leaching characteristics of material at the time of testing. Static tests are distinguished from “kinetic testing” and do not account for reaction rates/temporal changes that may occur in the material as chemical weathering proceeds. Static tests provide a balance of bulk acid generating and acid consuming reactions at an end point and may be used to determine the potential magnitude of leaching metals/solutes from a given sample material.

20.6.6.1. Waste rock

Geochemical characterisation of the waste rock samples was undertaken by Waterlab (Pty) Ltd, South Africa. Each of the 39 waste rock samples (plus 4 duplicates) were analysed according to the following methods:

- ABA, including paste pH, carbon and sulfur speciation, acid generating potential (“AP”) and modified neutralisation potential (“NP”)
- Net acid generation (“NAG”) testing.
- Multi-element analysis following acid digestion.

20.6.6.1.1. Acid base accounting

ABA methods provide an industry-recognised screening-level assessment of the potential for acid generation or acid neutralisation from a material. The approach is based on the balance and comparison between the potential to generate acid quantified by analysis of sulfide content and the potential to neutralize acid either from back titration method or by measuring carbonate content. It does not specifically consider mineralogy, kinetics or other influencing factors controlling sulfide oxidation but can be considered as characterising the 'total potential reservoir of acidity or alkalinity in a given material'.

ABA was carried out in accordance with EPA 600 Modified Sobek Method (Sobek et al., 1978). The test method reports paste pH, total sulfur by combustion and infrared (IR) spectroscopic detection (LECO), and NP by reacting a sample with hydrochloric acid ("HCl") and titrating with sodium hydroxide (NaOH). Organic and inorganic carbon were determined by the LECO method, sulfate was determined by dissolution with dilute HCl, addition of BaCl and spectrophotometric analysis of BaSO4; sulfide was determined by difference. Waterlab reported total sulfur by Leco with a reporting detection limit (RDL) of <0.01 %. Further sulfur speciation was sub-contracted to UIS Analytical, South Africa, who reported total sulfur, sulfide and sulfate to an RDL of <0.006 %.

AP is expressed as kg $CaCO_3$ equivalents per tonne and calculated as:

$$AP\ (kg\ CaCO_3\ eq/t) = \text{sulfide sulfur (as pyrite) (\%)} \times 31.25\ (kg/t)$$

NP was determined through titration as described above and expressed as kg $CaCO_3$ equivalents per tonne. A comparative NP was calculated from the total inorganic carbon ("TIC") content, which is assumed to be derived from carbonate, and converted to kg $CaCO_3$ equivalents per tonne:

$$NP_{TIC}\ (kg\ CaCO_3\ eq/t) = TIC\ (\%) \times \left[\frac{100.09\ g/mol\ CaCO_3}{12\ g/mol\ C}\right]$$

From the values of AP and NP it is possible to determine the difference of these factors expressed as Net Neutralisation Potential ("NNP") and the ratio as Neutralisation Potential Ratio ("NPR") of each sample as follows:

- NNP (kg $CaCO_3$ equivalents per tonne) = NP – AP
- NPR = NP / AP/

The NNP and NPR allow classification of the samples as PAG or non-potentially acid generating (Non-PAG). The results of ABA testwork can be used to determine the potential for acid generation from each lithology using the criteria outlined in Table 20-1

Table 20-1: Interpretation of ABA data

Classification	NNP [kg $CaCO_3$ eq/t]	NPR
Potentially acid generating (PAG)	NNP < -20	NPR < 1
Non-potentially acid generating (Non-PAG)	NNP > +20	NPR > 3
Uncertain acid generating characteristics (Uncertain)	-20 ≤ NNP ≤+20	1 ≤ NPR ≤3

20.6.6.1.2. Net acid generation testing

NAG testwork was undertaken on all samples to determine the maximum net potential for acid generation. The static NAG test differs from the ABA test in that it provides a direct empirical estimate of the overall sample reactivity, including any acid generated by semi-soluble sulfate minerals (i.e., jarosite) as well as other potentially acid-forming sulfate minerals. The NAG test was conducted in accordance with the Prediction Manual for Drainage Chemistry from Sulfidic Geological Materials MEND Report 1.20.1 (2009). The method involves intensive oxidation of the sample using hydrogen peroxide (H_2O_2), which accelerates the dissolution of sulfide minerals and has the net result that acid production and neutralisation can be measured directly. Leachate is then titrated with sodium hydroxide in two stages (pH 4.5 and to pH 7) to determine the NAG value, calculated as follows:

$$NAG = (VInit / Vx)\ (49 \times VNaOH \times M) / W$$

- Where:
  - NAG = net acid generation (kg H_2SO_4 eq/t)
  - VInit = volume of initial hydrogen peroxide solution (mL)
  - Vx = volume used to determine NAG by titration (mL)

- VNaOH = volume of NaOH used in titration (mL)
- M = concentration of NaOH used in titration (moles/L), and
- W = weight of sample reacted (g)

The guidelines used for assessing the acid generation potential based on NAG testwork results are summarised in Table 20-2. Samples with a NAG pH greater than pH 4.5 and a NAG value less than 1kg H_2SO_4 eq/t are typically classed as non-acid forming materials. A NAG value greater than 1 kg H_2SO_4 eq/t indicates the sample will generate some acidity in excess of available alkalinity and is potentially acid forming. However, by convention, any NAG value below 10kg H_2SO_4 eq/t of material has a limited potential for acid generation and the results are considered inconclusive because a blank hydrogen peroxide solution (the reagent in the NAG test) can generate a NAG artefact value up to 10 kg H_2SO_4 eq/t.

Table 20-2: Acid generation criteria for NAG results

Acid Generation Capacity		Final NAG pH	Static NAG [kg H_2SO_4 eq/t]
Potentially acid generating (PAG)	Higher capacity	< 4.5	>10
	Lower capacity	< 4.5	1 < NAG ≤10
Non-potentially acid generating (Non-PAG)		≥ 4.5	0

20.6.6.1.3. Multi-element analysis

A multi-element analysis was carried out to provide an estimation of the availability of metals for leaching. The analysis involves a strong multi-acid digestion followed by ICP-MS analysis for a suite of metal(loid)s. The results were assessed using the Geochemical Abundance Index (“GAI”) (INAP, 2002) to compare the concentration of an element in a given sample to the average crustal abundance (“ACA”) of that element. The GAI values are particularly useful in determining the relative enrichment of elements based on lithology and may be used to identify elements enriched above average crustal concentrations. Elements enriched relative to average crustal concentrations may provide an indication of potentially higher concentrations in contact waters. However, the release of an element into water is a function of many factors, such as solubility, pH and oxidation conditions, and not just on the solid concentration or enrichment factor. GAI values were calculated as follows:

$$GAI = \log_2 [C/(1.5*S)]$$

Where C in mg/kg (or ppm) is the concentration of an element as determined from the multi element analysis and S is the ACA of the element of interest (Mason, 1966). Materials are then assigned a GAI value between zero and six based on the degree of enrichment (Table 20-3). According to the INAP (2002) protocol, a GAI value greater than three indicates significant enrichment.

Table 20-3: GAI value interpretation

GAI value	Interpretation
0	< 3 times average crustal concentrations
1	3 to 6 times average crustal concentrations
2	6 to 12 times average crustal concentrations
3	12 to 24 times average crustal concentrations
4	24 to 48 times average crustal concentrations
5	48 to 96 times average crustal concentrations
6	>96 times average crustal concentrations

20.6.6.1.4. Leaching environmental assessment framework methods

Testing was undertaken in accordance with the Leaching Environmental Assessment Framework (“LEAF”) Methods and Guidance produced by the United States, Environmental Protection Agency (US, EPA). A sub-set of ten waste rock samples were selected to represent the different lithologies across the five deposits and submitted for variable pH leach tests and leach testing at various liquid to solid ratios. The following methods were undertaken:

- EPA Method 1313: Liquid-Solid Partitioning as a Function of Eluate pH Using a Parallel Batch Extraction Procedure (US EPA, 2017a).
- EPA Method 1316: Liquid-Solid Partitioning as a Function of Liquid-to-Solid (L/S) Ratio Using a Parallel Batch Extraction Procedure (US EPA, 2017b).

The EPA 1313 method is designed to evaluate the partitioning of constituents between liquid and solid phases at or near equilibrium conditions over a wide range of pH values. The method consists of a series of parallel batch extractions of solid material at various target pH values, achieved with an aliquot of either dilute acid or base. Testing is undertaken on a pulverized sample at a fixed L/S ratio of 10 and produces a liquid-solid portioning curve of constituents as a function of pH. Each sample underwent leaching at pH 2, 3, 4, 5, 6, 7 and 8.

The EPA 1316 method is a series of batch dilution tests intended to provide eluate solutions as a function of the L/S ratio. This method consists of five parallel batch extractions in reagent water over a range of L/S ratios. The method provides liquid-solid partitioning at the natural pH of a solid material as a function of L/S ratio. Each sample underwent leaching at L/S ratios of 10:1, 2:1, 1:1 and 0.5:1.

20.6.6.1.5. Spent ore

The 6m columns at Mintek Laboratories were discharged in 1m intervals. A representative sample from each 1m section was dried, crushed to minus 1.7mm then analysed for U_3O_8 by XRF and multi-element analysis by fusing solid samples with sodium peroxide and sodium carbonate, dissolving with hydrochloric acid and water followed by ICP-OES analyses. A further representative sample from each of the 1m sections was recombined with the other sections from the same column to generate a single sample per column. This single sample was assayed according to the following four size classes:

1. +1.7 mm
2. +3.35 mm
3. +1.18 mm
4. -1.18 mm

The four size classes were analysed for U_3O_8 by XRF and multi-element analysis as above. The four different size classes were subsequently reconstituted into a single sample representing each column. The reconstituted solid samples underwent ABA testing including total sulfur by combustion. Sulfide and sulfate sulfur was determined by dissolving samples in trichloroethylene to extract elemental sulfur, boiling the residue in sodium carbonate, filtering and reacting the filtrate with barium sulfate to determine sulfate sulfur by gravimetry then determining sulfide sulfur by combustion analysis of the filtered solids.

NP was determined by contacting each sample with 0.1M hydrochloric acid to pH 2-2.5 and neutralising excess acid with sodium hydroxide. The reconstituted samples from each of the columns also underwent a toxicity characteristic leaching procedure ("TCLP") according to method EPA 1311. The test involves preparation of two extraction solutions (pH 4.93 (fluid #1) and pH 2.88 (fluid # 2)). The solid sample is first tested with distilled water and heated, if the pH is < 5 extraction solution #1 is used, if the pH is >5 extraction solution #2 is used. The sample is contacted at a liquid to solid ratio of 1:20 over 18 hours and the subsequent leachate is analysed by ICP-MS.

20.6.6.1.6. Gypsum waste

The two gypsum solid samples underwent the following analyses:

- Total sulfur, sulfide sulfur, sulfate sulfur analyses and determination of NP as described above
- Multi element analysis as described above
- TCLP in accordance with method EPA 1311 described above
- Shake flask extraction where the gypsum residues were contacted with water at a 3:1 liquid to solid ratio for 24 hours and the filtrate was analysed by ICP.

20.6.7. Quality assurance/ quality control

20.6.7.1. Waste rock

In addition to the Waterlab inhouse QAQC checks, SRK reviewed the data to confirm the quality of laboratory results received, including comparison of four split duplicate samples. Overall, the data was considered acceptable.

Modified NP results were issued as two datasets by the laboratory:

- Including negative values for Modified NP
- With negative values for Modified NP replaced by zero.

Negative NP values occur when acidity is stored within the sample and the sample releases this acidity as it is dissolved during the test. The negative NP values were reviewed alongside paste pH and carbon speciation data to determine the accuracy of these results. In some instances, the negative NP results appeared to over-estimate stored acidity within the samples. Waterlab confirmed that this can occur due to imprecise titrations where very small volumes are required to achieve the desired pH. Hence, the laboratory re-issued results with negative NP values replaced by zero. The overall conclusions regarding the potential for acid generation of the samples are not altered whether the zero or negative values are used in the interpretation; therefore, zero values are reported as default.

In the variable pH test, boron and phosphorous concentrations were typically >5x RDL in the majority of blank samples. These results were rechecked with the laboratory and are potentially attributed to the reagents used to adjust pH although reagents were analytical grade; these are not key parameters of concern for the Project. An acceptable ion balance was reported for the majority of parameters. Some exceptions were noted for the Muntanga Sandstone sample in the variable L/S test; these were investigated with Waterlab and potentially occurred due to the difficulty in recovering solutions at low L/S ratios or potential loss of alkalinity during the filtration process. The results suggest that some particulate matter was still present in the leachates during analyses and therefore the leachate results may be more representative of totals rather than dissolved concentrations. Consequently, this sample typically reports elevated trace element concentrations in the L/S test at neutral pH when compared with the variable pH results. All pH 6 and pH 7 results were verified through repeat analyses at Waterlab and produce an acceptable ion balance, so these are considered more representative of true leaching values.

20.6.7.2. Spent ore, gypsum waste and bleed stream data quality assurance/ quality control

Mintek Laboratories undertake inhouse QAQC; depending on the availability of materials, a minimum of 20 % of samples are analysed in duplicate or more. In cases where triplicate (or more) analyses are undertaken, variation for replicates are assessed with a Horwitz Function tool which assesses relative standard deviation. If available at least one certified or in-house reference standard is analysed for ever batch of ten samples and blanks are analysed for every data set.

Duplicate gypsum samples were analysed as part of the ABA, multi element analysis and shake flask extraction test (for pH, EC, fluoride and phosphate); all results f were reported within the acceptable criteria.

20.6.8.Waste rock geochemical testwork results

20.6.8.1. Acid generating potential

20.6.8.1.1. Dibbwi, Dibbwi East and Muntanga

Figure 20-2 shows the relationship between paste pH and total sulfur content in the Dibbwi, Dibbwi East and Muntanga samples. Paste pH provides an indication of the availability of readily soluble sulfate salts where low pH values (<pH 5) may indicate the presence of acidic reaction products generated by sulfide oxidation. Paste pH was greater than pH 5.5 in all samples from these deposits except for one Dibbwi East sandstone sample (24-2329) which reported a paste pH of 5.4. Paste pH for all other samples ranged from pH 5.9 to 7.5. Consequently, reactive acidity is unlikely from these materials during operations.

Total sulfur content was low in all of the Dibbwi, Dibbwi East and Muntanga samples. Highest total sulfur was reported at 0.84 wt.% in one Dibbwi East sandstone sample (24-2327) but concentrations in all other samples ranged from 0.004 to 0.37 wt.%. Figure 20-3 shows that only some of the sulfur is present as sulfide sulfur. The siltstones and mudstones generally show a better correlation between sulfide and total sulfur but the sandstones comprise both sulfide and sulfate with no obvious trends across the different deposits. AP was calculated from sulfide sulfur content and ranged from 0.003 to 25 kg CaCO3/t.

Modified NP was low in the Muntanga, Dibbwi and Dibbwi East samples ranging from 0kg to 44kg $CaCO_3$/t and NP calculated from TIC ranged from 0.25 kg to 56 kg $CaCO_3$/t. Figure 20-4 presents a comparison of modified NP and NP_{TIC} (zero modified NP values are set to 0.001 kg $CaCO_3$/t for graphing purposes). NP_{TIC} results were either similar to or consistently higher than modified NP determined by titration which could indicate the presence of non-reactive carbonates or potential inaccuracies with the titration method.

The AP versus modified NP and NP_{TIC} is shown in Figure 20-5. Due to the low modified NP results; the Muntanga sandstones and four Dibbwi East sandstones are classified as PAG according to this method. However, according to the NP_{TIC}/AP just the one Dibbwi East sandstone sample with the higher sulfur content is classified as PAG; a further two Dibbwi East sandstones and one Muntanga sandstone reported uncertain acid generating potential. Figure 20-6 compares sulfide content with NPR derived from modified NP and NP_{TIC}. As described above several sandstones are classified as PAG material according to the modified NP but the majority of these PAG samples have low sulfide content (less than 0.02 %, except for one sample at 0.2 %). The reported classifications are due to low buffering and low sulfide concentrations close to the RDL of <0.006%. In reality this is an overly conservative classification; a sulfide sulfur threshold of 0.01 % is also shown on Figure 20-5 and Figure 20-6; long-term sustained acid generation from samples with less than 0.01% sulfide would be unlikely..

Static NAG testing was used as an additional measure of ARD potential. NAG pH is compared with total NAG values in Figure 20-7; these results support the NPTIC/AP results and conclusion that acid generating is not expected at such low sulfide concentrations. All samples classified as non-PAG with NAG pH values ranging from pH 5.0 to pH 6.7 in all samples from Dibbwi, Dibbwi East and Muntanga. Total NAG values ranged from 1.4 kg to 8.2 kg H_2SO_4 eq/t.

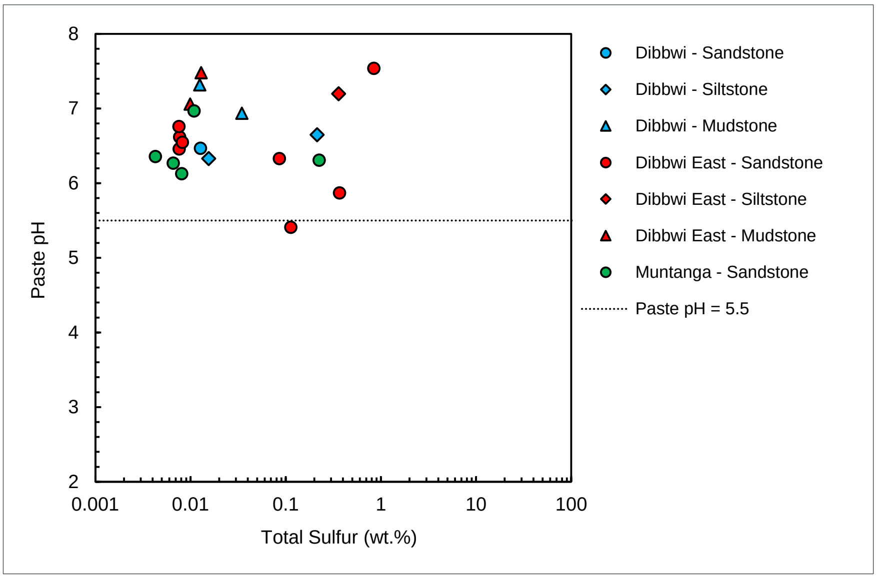


Figure 20-2: Total sulfur vs. paste pH for Dibbwi, Dibbwi East and Muntanga Samples

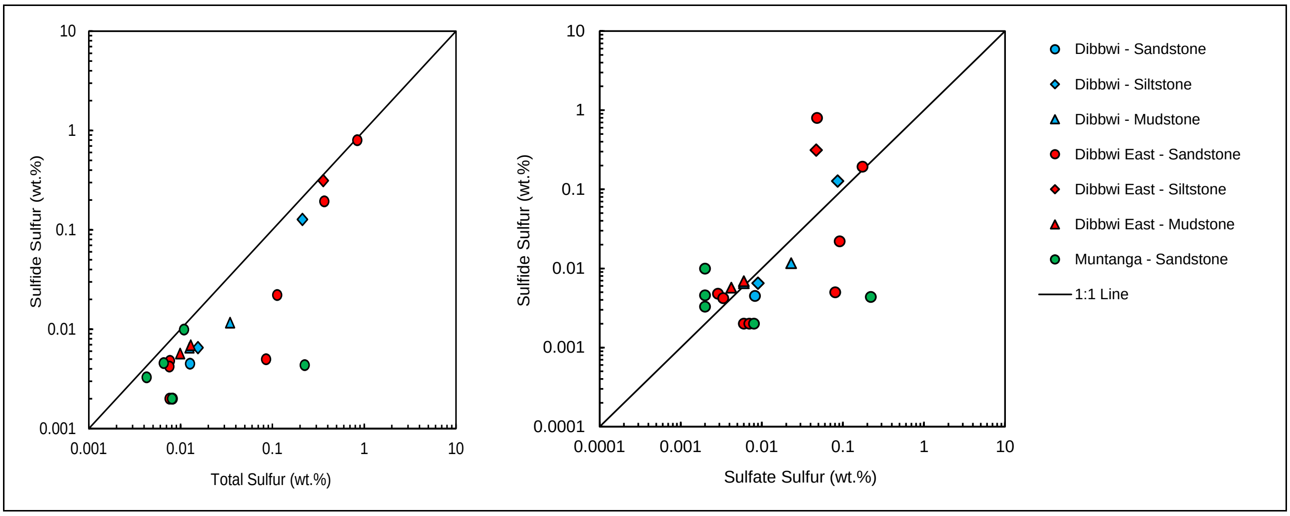


Figure 20-3: Total sulfur vs sulfide sulfur (left) and sulfate sulfur (right) for Dibbwi, Dibbwi East and Muntanga samples

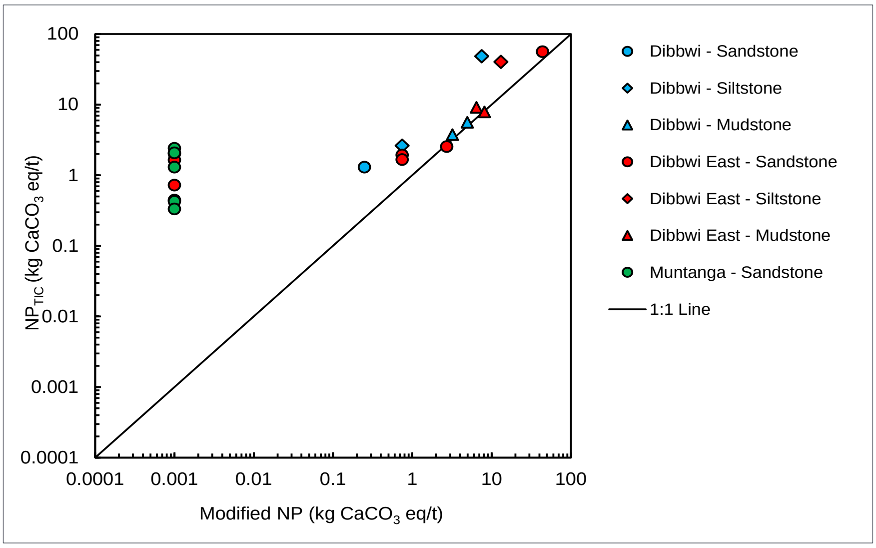


Figure 20-4: Modified NP vs. NPTIC for Dibbwi, Dibbwi East and Muntanga samples

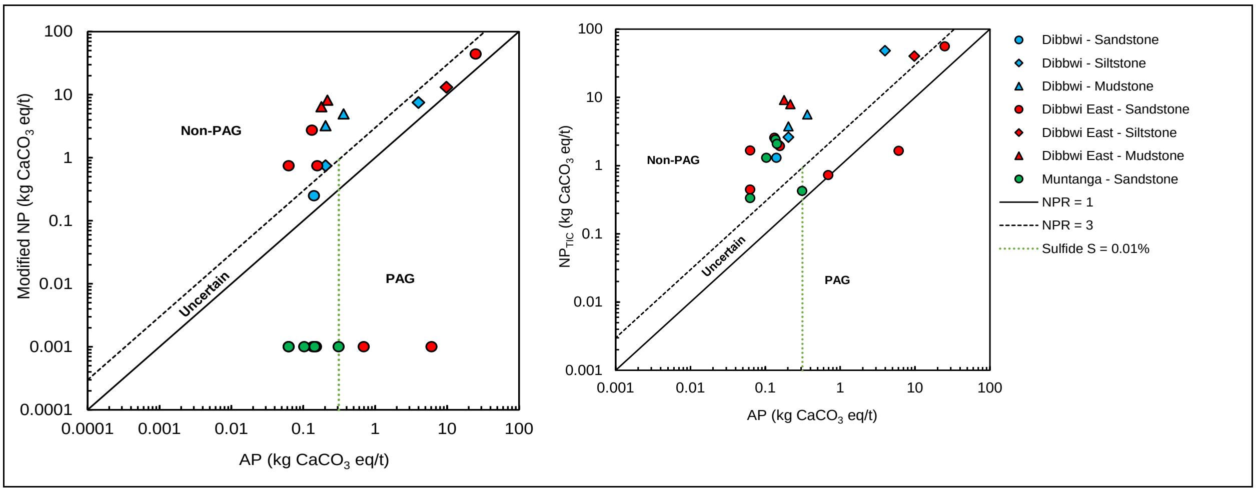


Figure 20-5: AP vs. modified NP (left) and NPTIC (right) for Dibbwi, Dibbwi East and Muntanga samples

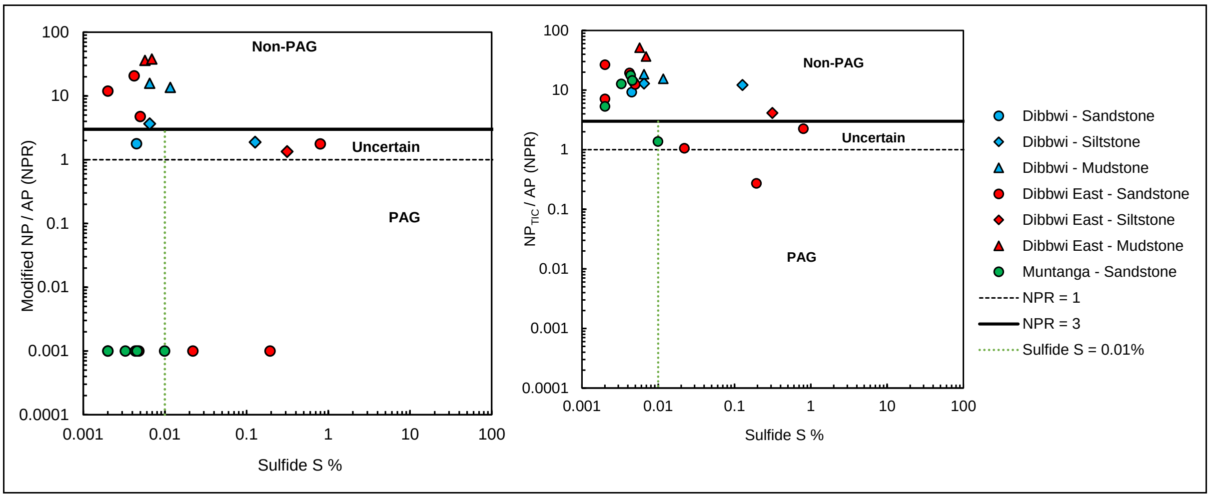


Figure 20-6: Sulfide s vs. NPR by modified np (left) and NPTIC (right) for Dibbwi, Dibbwi East and Muntanga samples

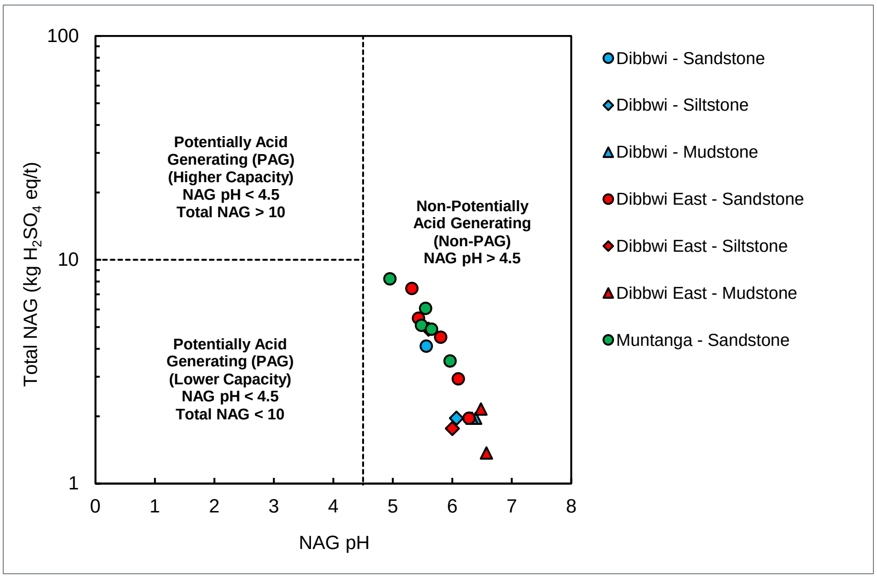


Figure 20-7: NAG pH vs. Total NAG for Dibbwi, Dibbwi East and Muntanga samples

20.6.8.2. Metal leaching potential

Multi-element analysis results for key parameters are presented in Table 20-4 and compared with average crustal abundance (“ACA”) (Mason, 1966) using the GAI. Some parameters reported detection limits exceeding three times ACA including silver, gold, beryllium, mercury, molybdenum, palladium, selenium and tellurium. Assessment of potential enrichment of these parameters is limited to concentrations measured about the RDL.

20.6.8.2.1. Dibbwi, Dibbwi East and Muntanga

Uranium was elevated in the majority of waste rock samples; Dibbwi showed the lowest enrichment with just one sample reporting concentrations greater than three times ACA. At Dibbwi East and Muntanga uranium concentrations averaged 48 ppm and 39 ppm respectively relative to an average crustal abundance of 1.8 ppm (Mason, 1966). Other literature sources quote naturally occurring uranium in the Earth’s crust at concentrations 1 ppm to 3 ppm (Cumberland, 2016). A box and whisker plot in Figure 20-8 shows the minimum, average and maximum uranium concentrations reported in the geochemistry characterisation data in relation to exploration drillcore analysis data for samples intercepting the pits (excluding uranium concentrations >85 ppm.

The plot shows that the maximum uranium concentration reported for Dibbwi East (320 ppm) is significantly higher than the exploration drillcore analysis data suggesting that this result is an outlier. As discussed in Section 20.6.5, the introduction of ore sorting will target ore concentrations >40 ppm so a uranium concentration as high as 320 ppm is not expected within waste rock.

On the contrary, it is possible that the uranium concentrations reported in the geochemical characterisation for Dibbwi potentially underestimate average waste rock uranium concentrations based on the comparison with the exploration drillcore analysis data. The maximum Dibbwi uranium concentration of 12 ppm reported in the geochemical characterisation study is below the average concentration for this deposit from the exploration drillcore database (28 ppm). The bottom image of Figure 20-8 shows the Dibbwi East outlier removed and demonstrates that the geochemical characterisation samples otherwise show good representivity when compared with the exploration drillcore dataset. Overall uranium concentrations will still be elevated in the waste relative to average crustal abundance.

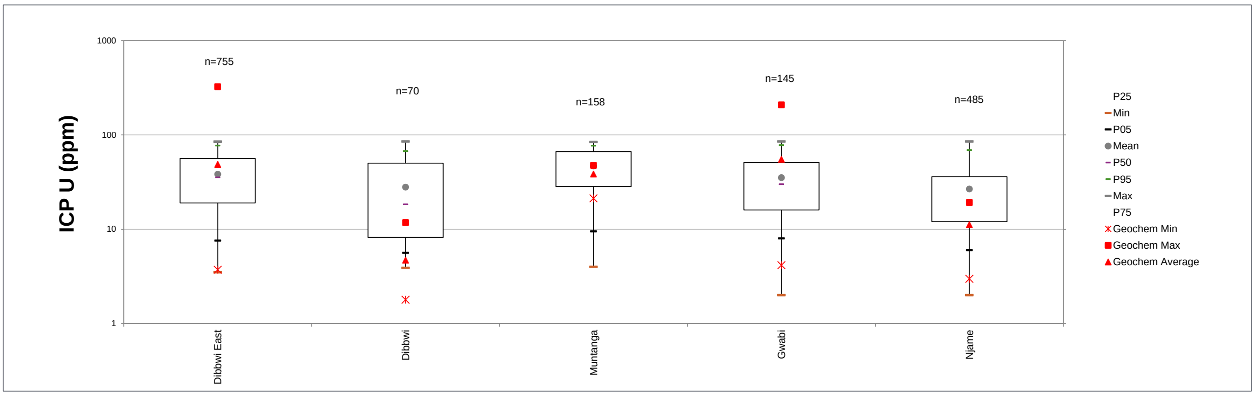
Other parameters which were elevated by more than three times ACA were boron (67 % of samples), molybdenum (43 % of samples, excluding those at RDL) and selenium (48 % of samples, excluding those at RDL). Less common exceedances were also noted for silver (four samples), gold (one sample), bismuth (two samples), lithium (one sample), manganese (two samples), antimony (one sample), tantalum (seven samples) and tellurium (five samples).

Table 20-4: Summary of Multi-Element Analysis Results

Hole ID	From	To	Lithology	Deposit`	Ag	Al	As	Au	B	Be	Bi	Cd	Co	Cr	Cu	Fe	Hg	Mg	Mn	Mo	Ni	Pb	Sb	Se	Si	Sn	Ta	Te	U	W	Zn
	m	m			ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
Average Crustal Abundance (Mason, 1966)>>					0.07	81300	1.8	0.004	10	2.8	0.2	0.2	25	100	55	50000	0.08	20900	950	1.5	75	13	0.2	0.05	277200	2	2	0.01	1.8	1.5	70
DBD1537	8.90	24.07	Sandstone	Dibbwi	0.63	7200	2.0	0.40	10	10	0.40	0.40	10	63	4.0	35000	0.40	400	140	10	10	19	0.40	0.40	200000	0.76	0.40	0.40	1.8	2.0	54
DBD1537	24.07	29.83	Mudstone	Dibbwi	0.40	13000	2.3	0.40	10	10	0.42	0.40	10	30	4.0	56000	0.40	800	100	10	10	24	0.45	0.40	130000	3.8	0.40	0.60	2.4	3.6	40
DBD1537	29.83	34.18	Siltstone	Dibbwi	2.3	14000	5.1	0.40	98	10	1.1	0.40	11	61	4.0	30000	0.40	400	530	10	28	30	0.97	1.3	210000	5.2	3.0	0.40	3.7	3.7	100
DBD1539	5.00	10.00	Mudstone	Dibbwi	0.40	17000	3.7	0.40	10	10	0.46	0.40	10	59	4.0	32000	0.40	800	180	10	16	28	0.42	1.4	240000	3.7	0.40	0.40	3.9	3.3	76
DBD1539	14.90	17.90	Siltstone	Dibbwi	0.40	23000	2.2	0.40	270	10	0.40	0.40	10	67	4.0	30000	0.40	1600	170	20	15	27	0.40	0.40	270000	3.2	0.40	0.40	12	4.0	70
DVD1546	32.00	40.00	Sandstone	Dibbwi East	0.55	11000	2.0	0.44	210	10	0.40	0.40	10	100	15	12000	0.40	400	690	10	15	23	0.40	0.40	330000	0.40	50	0.40	12	0.88	10
DVD1389	24.50	29.50	Sandstone	Dibbwi East	0.40	18000	4.1	0.40	110	10	0.40	0.47	22	96	4.0	40000	0.40	800	12000	14	13	21	0.40	0.40	320000	0.40	20	0.40	48	1.7	65
DVD1394	25.00	31.00	Sandstone	Dibbwi East	0.40	25000	4.8	0.40	61	10	0.40	0.40	10	81	4.0	24000	0.40	800	1700	10	39	24	0.40	0.40	310000	0.40	12	0.40	320	1.2	29
DVD1409	13.42	22.50	Mudstone	Dibbwi East	0.40	18000	4.5	0.40	53	10	0.42	0.40	10	69	4.0	34000	0.40	400	490	10	40	25	0.40	10	200000	3.8	0.40	0.48	3.7	3.3	77
DVD1413	21.58	30.00	Sandstone	Dibbwi East	0.40	36000	3.8	0.40	76	10	0.40	0.40	10	81	6.1	41000	0.40	2400	740	26	16	24	0.40	2.3	270000	0.40	35	0.40	18	3.5	74
DVD1420	81.31	90.00	Siltstone	Dibbwi East	2.5	40000	2.4	0.40	40	10	0.95	0.40	10	81	7.2	34000	0.40	5200	1200	13	24	20	0.40	0.40	250000	0.40	9.8	0.57	5.5	2.2	38
DVD1430	28.50	32.50	Sandstone	Dibbwi East	0.40	27000	3.1	0.40	70	10	0.40	0.40	10	82	4.0	32000	0.40	1600	3400	10	41	27	0.40	1.7	290000	0.40	25	0.40	40	2.5	59
DVD1456	40.00	51.00	Sandstone	Dibbwi East	0.40	11000	2.8	0.40	10	10	0.40	0.40	10	82	4.0	17000	0.40	400	60	29	10	22	0.40	0.40	290000	0.83	0.40	0.40	13	2.0	10
DVD1486	131.00	138.00	Sandstone	Dibbwi East	0.40	24000	2.9	0.40	38	10	0.40	0.40	10	97	10	16000	0.40	2800	1300	11	10	24	0.40	0.40	320000	0.87	0.40	0.40	42	1.9	10
DVD1512	77.56	81.14	Mudstone	Dibbwi East	0.40	49000	2.0	0.40	70	10	0.48	0.40	10	70	4.0	49000	0.40	3200	200	10	21	28	0.40	14	210000	0.42	12	0.40	5.4	2.3	120
DVD1523	67.00	76.00	Sandstone	Dibbwi East	0.40	15000	3.8	0.40	10	10	0.40	0.40	10	84	4.0	23000	0.40	800	200	10	10	23	0.40	3.5	320000	0.85	0.40	0.40	25	1.4	10
MTD1533	7.09	10.00	Sandstone	Muntanga	0.40	13000	4.1	0.40	910	10	0.40	0.40	10	97	4.0	9200	0.40	400	450	10	15	29	0.40	0.40	310000	0.86	0.40	0.40	48	1.5	31
MTD1536	20.00	24.00	Sandstone	Muntanga	0.40	15000	3.6	0.40	37	10	0.40	0.40	10	110	4.0	10000	0.40	400	360	25	17	16	0.40	1.1	310000	0.57	0.40	0.43	45	1.6	10
MTD1541	28.50	32.56	Sandstone	Muntanga	0.40	14000	0.60	0.40	25	10	0.40	0.40	10	80	4.0	14000	0.40	400	30	22	15	18	0.40	1.8	320000	1.0	0.40	0.40	39	2.5	21
MTD1551	2.00	7.00	Sandstone	Muntanga	0.40	12000	3.2	0.40	10	10	0.40	0.40	10	50	4.0	15000	0.40	400	210	19	10	23	0.40	0.75	200000	0.81	0.40	0.94	40	4.3	10
MTD1545	0.22	5.00	Gritty Sandstone	Muntanga	0.40	12000	3.1	0.40	510	10	0.40	0.40	10	92	4.6	7600	0.40	400	150	10	16	24	0.40	0.40	290000	0.70	0.40	0.40	21	2.7	10

NOTES:
Results reported to 2 significant figures except where closure to RDL
Blue italics indicate results less than the reported detection limit (RDL)
Ppm = parts per million

Indicates <3 times average crustal concentrations
Indicates between 3 and 6 times average crustal concentrations
Indicates between 6 and 12 times average crustal concentrations
Indicates >12 times average crustal concentrations
Indicates result at RDL but RDL exceeds 3 times average crustal abundance



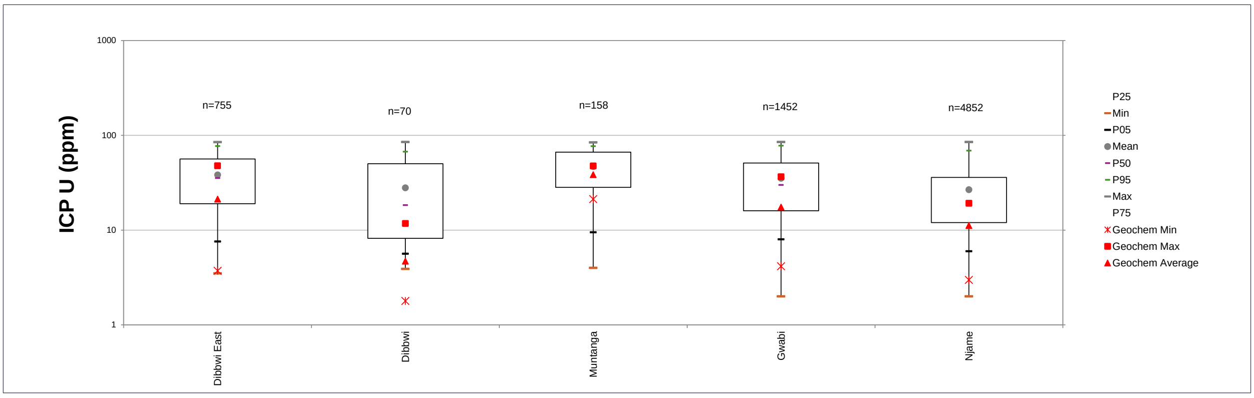


Figure 20-8: Box and whisker plot for in-pit uranium concentrations (<85 ppm) from Exploration Drillcore in relation to solid phase uranium reported in the Geochemical Characterisation (top – all data, bottom – with potential outliers removed)

20.6.8.3. Waste rock LEAF method results

The LEAF tests are a set of non-regulatory tests that provide more flexibility by evaluating leaching under a wider range of environmental conditions as a series of short-term static leach tests. A sub-set of ten waste rock samples were selected to represent the different lithologies across the five deposits and are shown in Figure 20-9 in relation to acid generating potential determined by NPTIC versus AP.

Two main aspects are assessed:

1. Leachability as a function of pH – for many elements, leachability is controlled by the solubility of host mineral phases or the nature of interactions with mineral surfaces (i.e., surface adsorption/desorption). Solubility and sorption are both strongly influenced by solution pH, and collecting data as a function of pH gives valuable insight to the potential mass of leachable constituent, likely concentrations at different solution pH levels, and predicted leaching behaviour
2. Leachability as a function of contact ratio – different constituents behave differently in leaching tests; highly soluble constituents tend to partition into the liquid phase, so concentrations in the initial solutions (e.g., at a low L/S ratio) are quite high but then drop quickly beyond a L/S ratio of 1. For less-soluble constituents, the majority of constituent mass remains in the solid phase, and concentration is more affected by pH while showing less dependence on L/S ratio.

This laboratory generated leachate results are not directly comparable to water quality standards or other water quality data but the similarities to groundwater baseline conditions are included in the discussion for contextual purposes only.

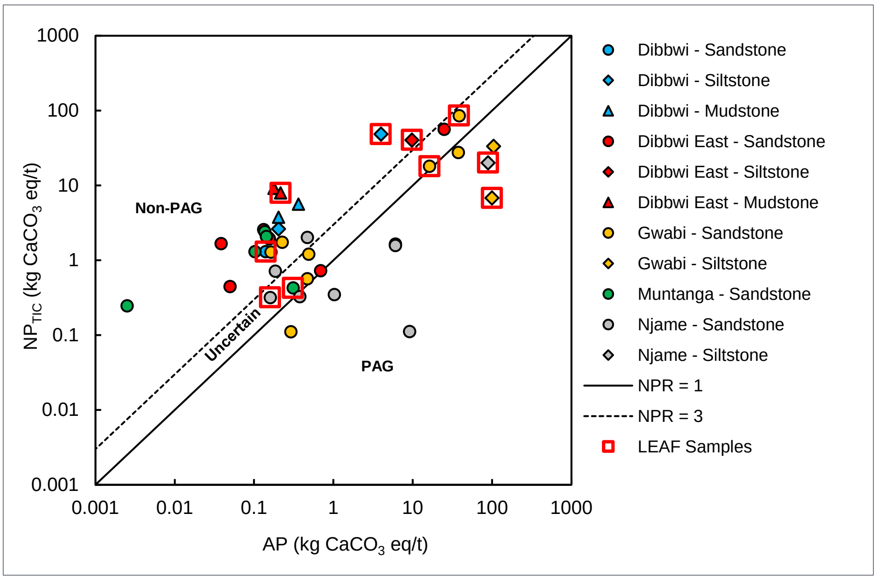


Figure 20-9: LEAF method selected samples shown on AP vs. NPTIC plot

20.6.8.3.1. Variable pH

Key parameter results for the EPA 1313 method are shown in Figure 20-10 to Figure 20-21 and summarised below:

- Sulfate concentrations were highest in the Gwabi sandstone (250 mg/L to 320 mg/L) (classified as having uncertain acid generating potential) and the Njame siltstone (150 mg/L to 250 mg/L) (classified as PAG). The concentrations are within the range reported in the groundwater baseline (2.4 mg/L to 2 100 mg/L). Most trends were generally stable with some samples showing an increase in concentrations as pH increased. A few potentially erroneous results are off-trend at pH 4
- Uranium concentrations were highest at acidic pH and showed a decrease around neutral pH. The Gwabi siltstone showed some further uranium mobility as pH increased around pH 8. The Gwabi sandstone sample reported highest concentrations of uranium (<0.57 mg/L to 7.1 mg/L). For comparison, groundwater baseline results for uranium are generally at or close to detection limit across all deposits (<0.005 mg/L). Thorium concentrations were at or close to detection limit in all samples from pH 3 and above.
- Several parameters showed higher concentrations at low pH which decreased with increasing pH including calcium, cobalt, nickel, lead, iron, manganese and zinc. Mineral solubility is often greatest at acidic pH and sorption onto mineral surfaces could be playing a role in the trends seen. Cationic solution species typically sorb most strongly at alkaline pH. Concentrations reported for these parameters at near-neutral pH were generally similar to or lower than those typically reported in the groundwater baseline; manganese concentrations were slightly higher.
- Selenium concentrations did not show any clear trends based upon changes in pH. Concentrations were highest in the Dibbwi mudstone (0.17 mg/L to 0.28 mg/L) which is consistent with the elevated concentrations in the solid phase. These selenium concentrations are higher than groundwater baseline which are generally reported at or close to RDL (<0.003 mg/L to <0.01 mg/L).
- Arsenic was elevated in some of the solid samples from Gwabi and Njame but was reported at RDL at near neutral pH which is consistent with groundwater baseline data, with the exception of one anomalous spike in the Dibbwi East siltstone at pH 5
- Boron was also reported in some of the solid samples across all deposits. Concentrations in the leachates were around 0.1 mg/L to 0.2 mg/L at circum-neutral pH although elevated boron was also noted in the method blanks. Baseline groundwater concentrations of boron are usually an order of magnitude lower than this but similar concentrations are reported in the vicinity of Gwabi.

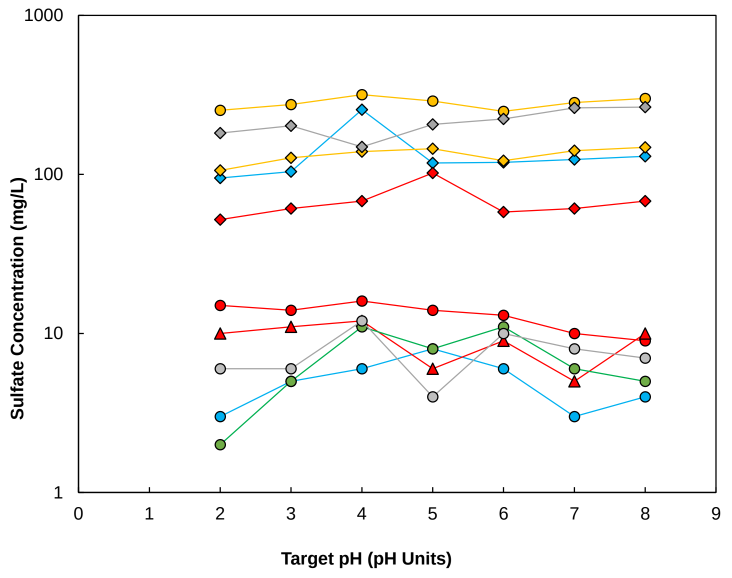


Figure 20-10: LEAF 1313 Test Results pH vs Sulfate Concentration

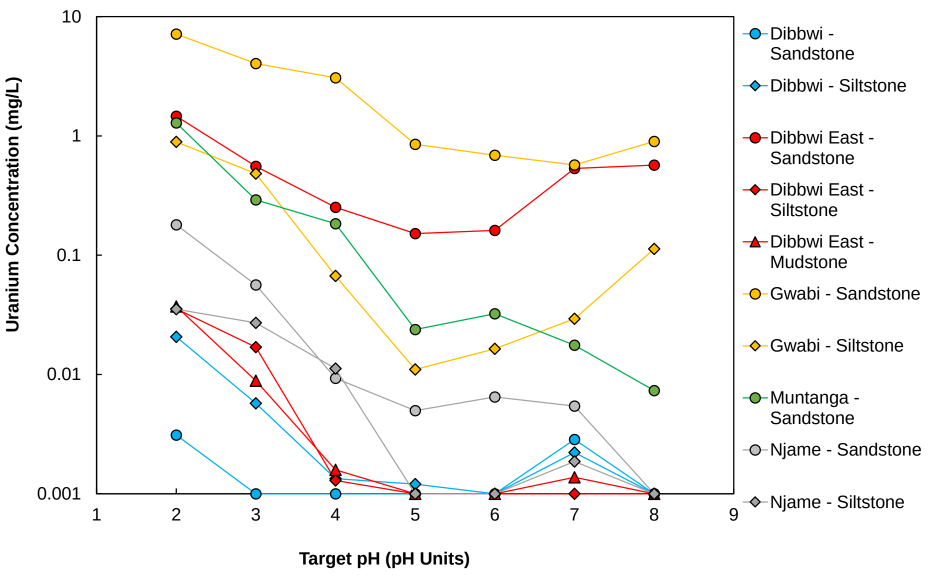


Figure 20-11: LEAF 1313 Test Results pH vs Uranium Concentration

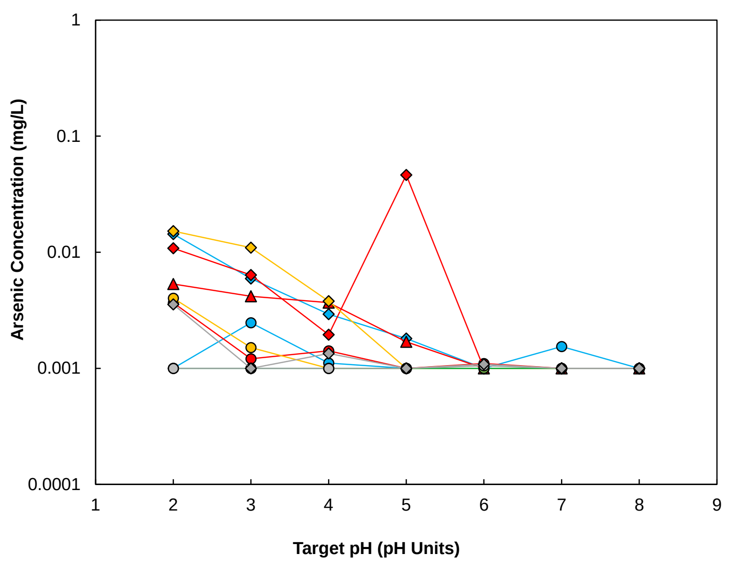


Figure 20-12: LEAF 1313 Test Results pH vs Arsenic Concentration

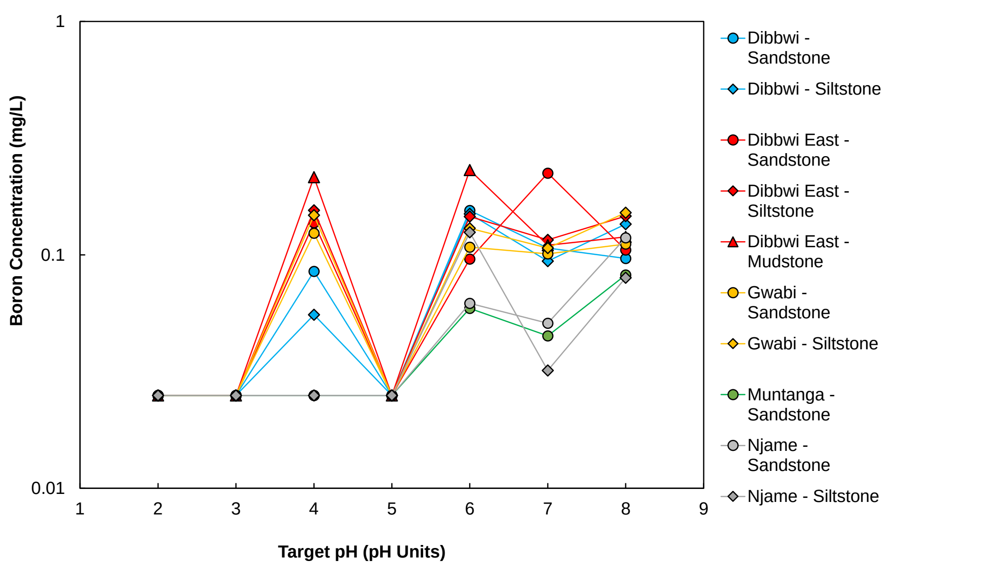


Figure 20-13: LEAF 1313 Test Results pH vs Boron Concentration

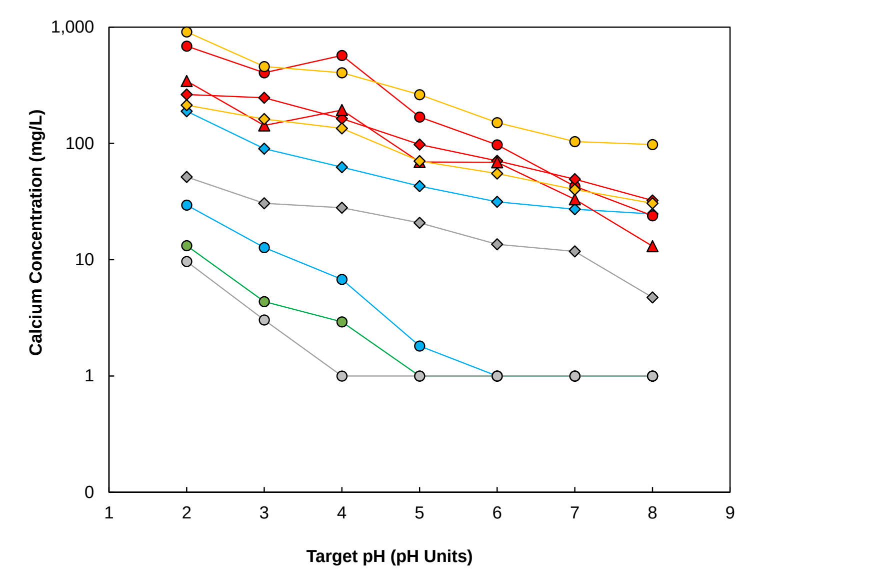


Figure 20-14: LEAF 1313 Test Results pH vs Calcium Concentration

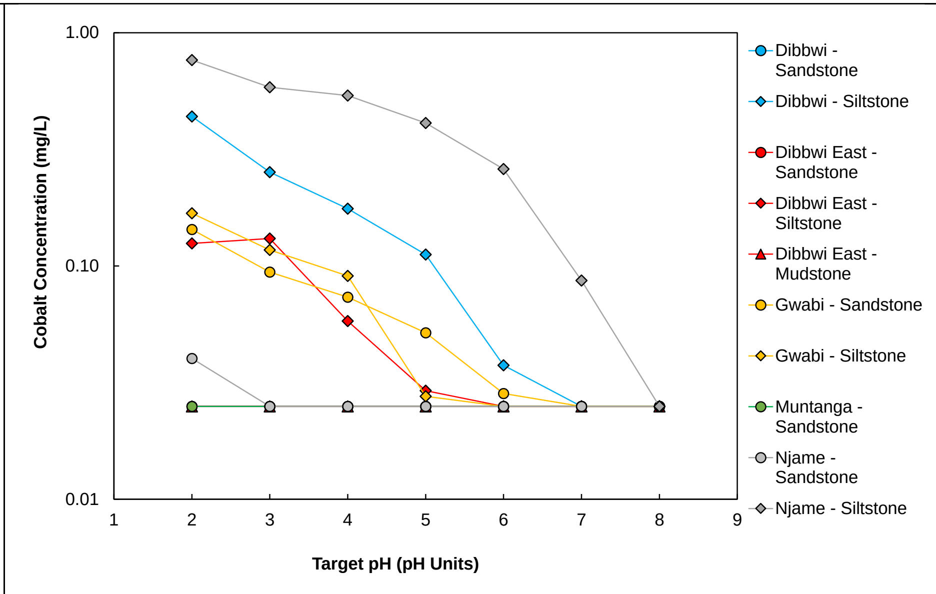


Figure 20-15: LEAF 1313 Test Results pH vs Cobalt Concentration

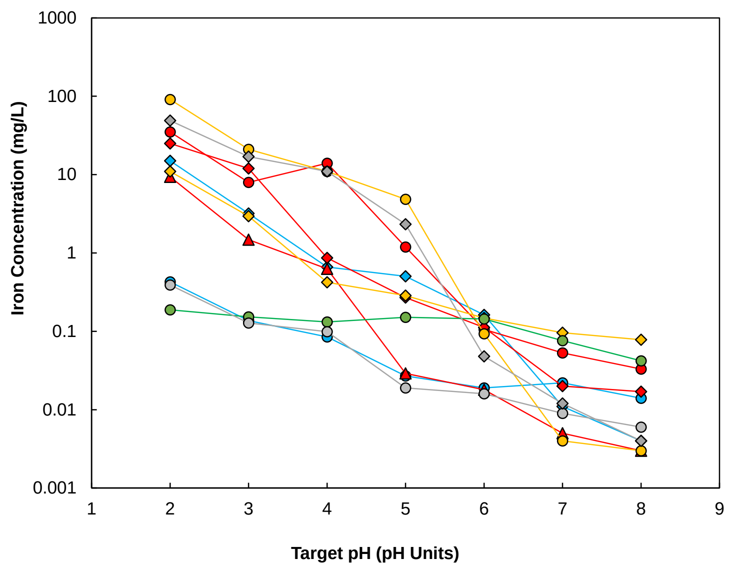


Figure 20-16: LEAF 1313 Test Results pH vs Iron Concentration

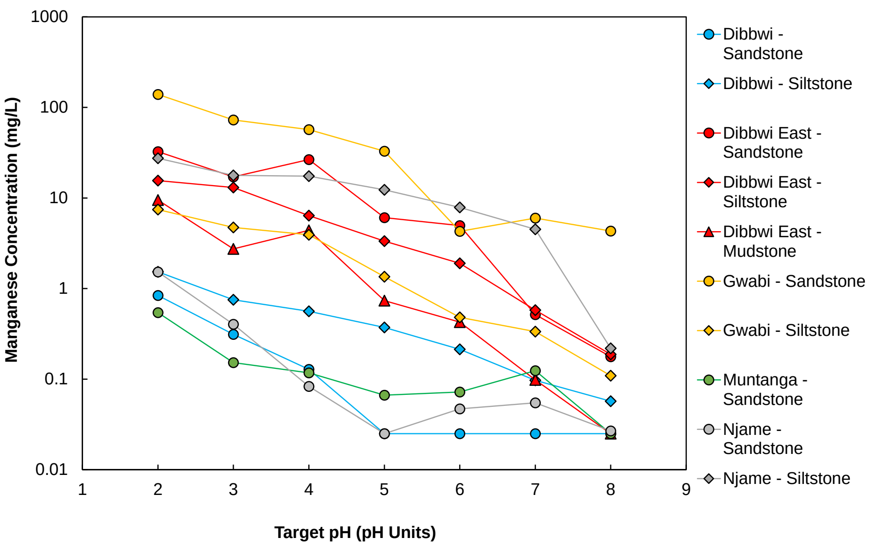


Figure 20-17: LEAF 1313 Test Results pH vs Manganese Concentration

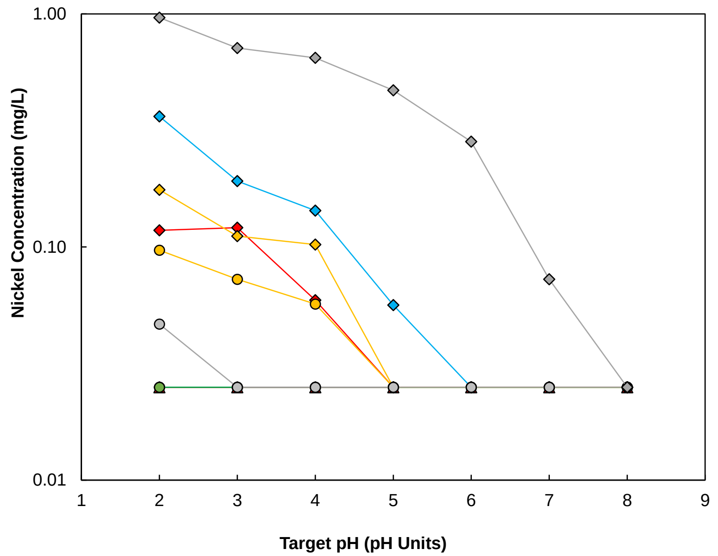


Figure 20-18: LEAF 1313 Test Results pH vs Nickel Concentration

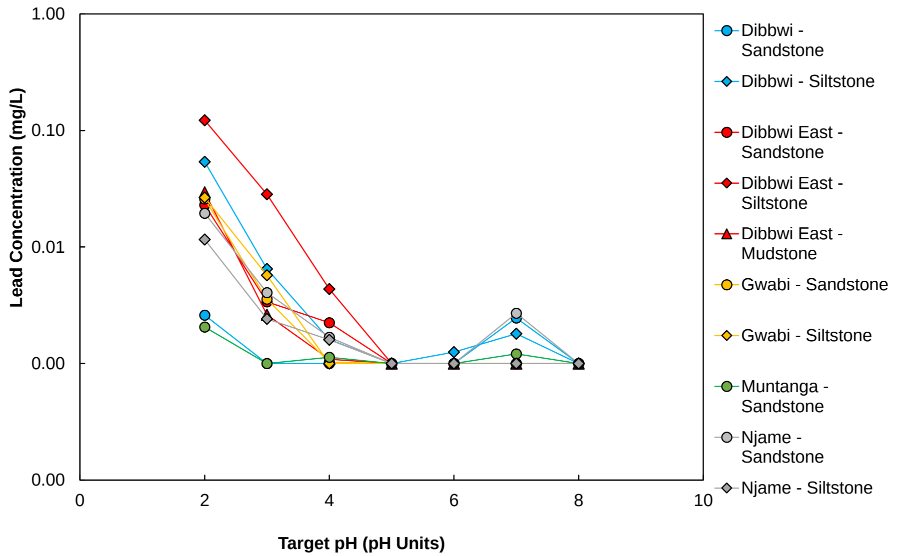


Figure 20-19: LEAF 1313 Test Results pH vs Lead Concentration

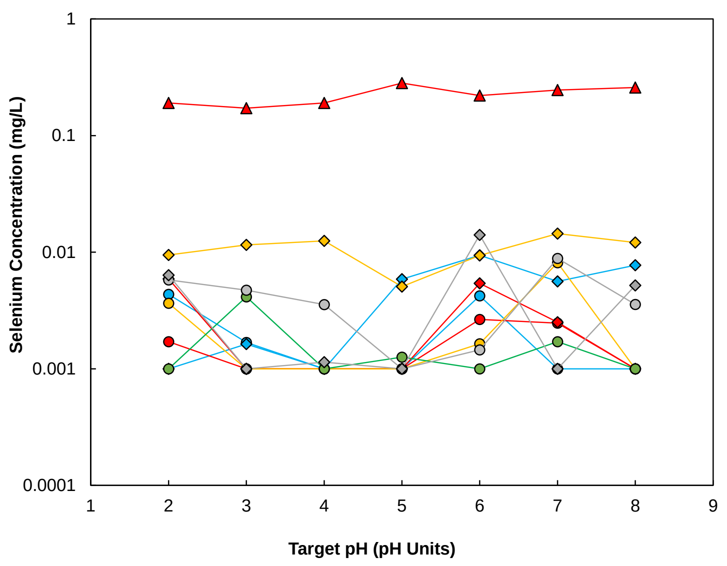


Figure 20-20: LEAF 1313 Test Results Selenium Concentration vs pH

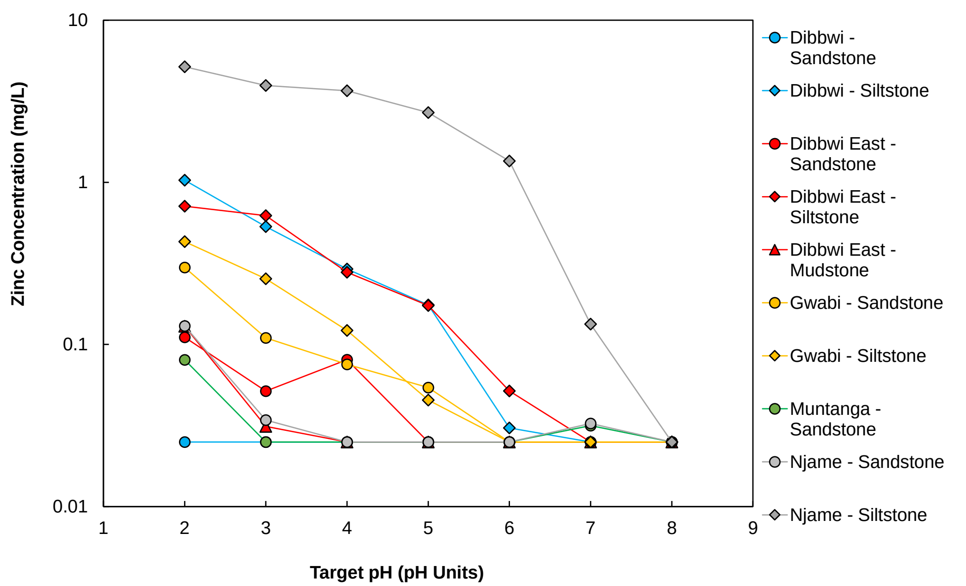


Figure 20-21: LEAF 1313 Test Results Zinc Concentration vs pH

20.6.8.3.2. Variable liquid to solid ratios

The results of the EPA 1316 test are presented in Figure 20-22 to Figure 20-33.

As per the EPA 1313 test, the laboratory generated leachates are not directly comparable to water quality standards or other water quality data but the concentrations are discussed in the relation to the groundwater baseline data for contextual purposes only. Key points are summarised below:

- Leachate pH is not typically affected by the variable L/S ratios, remaining relatively constant throughout the duration of the test as the L/S ratios increase. The Njame siltstone sample is the only sample to report an acidic pH, ranging from 3.8 to 4.6. The pH of the Njame sandstone ranged between pH 6.6 and 6.9, all other samples reported pH between pH 7.3 and 8.8
- Sulfate concentrations typically decreased with increasing L/S ratio and were overall highest in the Gwabi and Njame samples. This trend is typical of flushing of readily soluble salts. At the 10:1 L/S ratio, the sulfate concentrations were within a similar range or lower than those reported in the groundwater baseline
- Uranium concentrations also generally decrease with increasing L/S ratio. Concentrations within the Gwabi sandstone, Dibbwi East sandstone and Muntanga sandstone leachates are typically an order of magnitude or more higher than the other samples. The elevated uranium concentrations reported in the Muntanga sandstone sample are not consistent with the uranium concentrations reported around neutral pH in the variable pH test (Section **Error! Reference source not found.**) the pH 6 and pH 7 results w ere verified by repeat analyses and are considered more reliable in this instance. Thorium concentrations are consistently at detection limit
- There are not significant differences in arsenic concentrations at all stages of the test and highest concentrations are similar to groundwater baseline concentrations (<0.003 to <0.01mg/L)
- Boron concentrations are primarily at detection limit for the 2:1 and 10:1 L/S ratios despite being generally elevated in the solid phase across deposits
- Calcium follows a typical trend of decreasing concentrations with increasing L/S ratio. Highest concentrations are typically reported within the Gwabi sandstone but at the 10:1 ratio they are similar to or lower than groundwater baseline concentrations
- Cobalt concentrations are at RDL in all samples at the 2:1 and 10:1 L/S ratios except for the Njame siltstone. This sample did not show any elevated cobalt in the solid phase. At the 10:1 L/S ratio the concentration is 0.59 mg/L whereas groundwater baseline data is usually at <RDL
- Iron concentrations decrease with increasing L/S ratios in all samples except for the Dibbwi East samples and the Njame sandstone. Possibly due to low redox allowing ferrous release or if ferric iron could be contamination by particulate material. At 10:1 L/S ratios most samples report concentrations <10 mg/L which is consistent with groundwater baseline data but the Dibbwi East sandstone reports a maximum concentration of 23 mg/L
- Manganese concentrations show a general decrease with increasing L/S ratios except for in a Dibbwi East sandstone which increases. Njame siltstone reports highest concentrations (19 mg/L at the 10:1 L/S ratio) which is typically higher than concentrations seen in groundwater baseline data.
- Nickel concentrations are mostly at <RDL which is consistent with groundwater baseline data. The exception is the Dibbwi Siltstone which ranges from 9.1 to 0.7mg/L
- Lead is consistently reported at RDL
- Selenium concentrations typically decrease with increasing L/S ratio. The Dibbwi mudstone with elevated selenium in the solid phase and which reported elevated selenium in the pH variable test also reported highest concentration in the L/S test (0.3 to 2.5mg/L). Other samples were less than 0.02 mg/L in the 10:1 leachates
- Zinc concentrations showed a decreasing trend with increasing L/S ratios except for one Dibbwi East sandstone. The Njame siltstone reported highest concentrations at 3.7 mg/ in the 10:1 L/S ratio which is generally higher than groundwater baseline data.

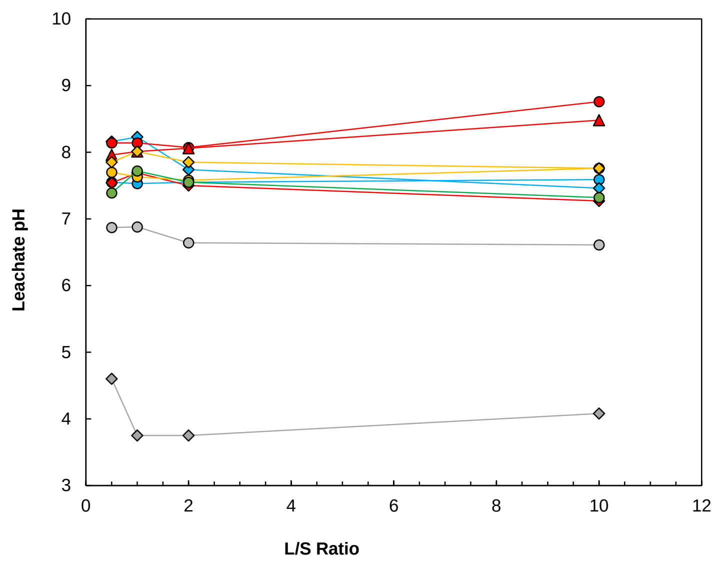


Figure 20-22: LEAF 1316 Test Results L/S Ratios vs Leachate pH

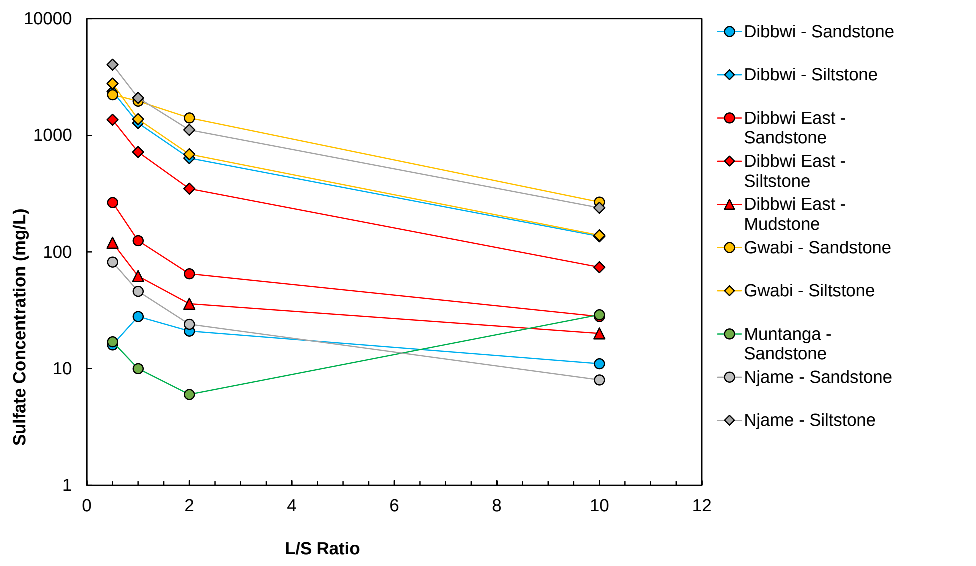


Figure 20-23: LEAF 1316 Test Results L/S Ratio vs Sulfate Concentration

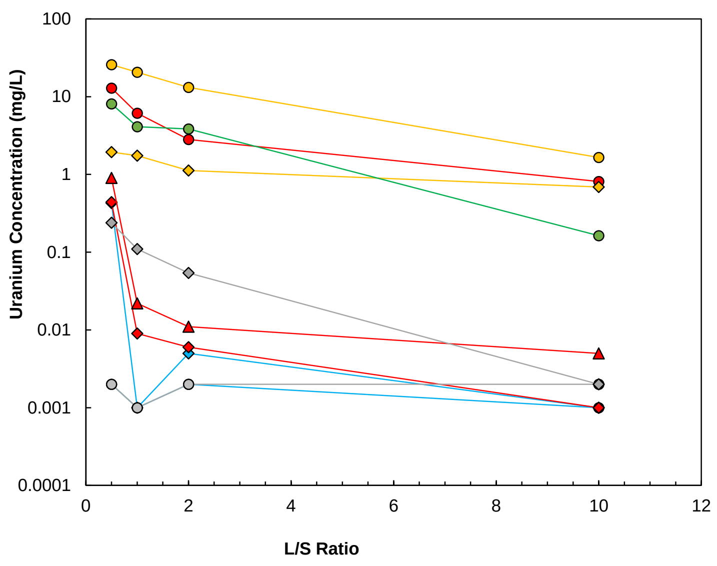


Figure 20-24: LEAF 1316 Test Results L/S Ratio vs Uranium Concentration

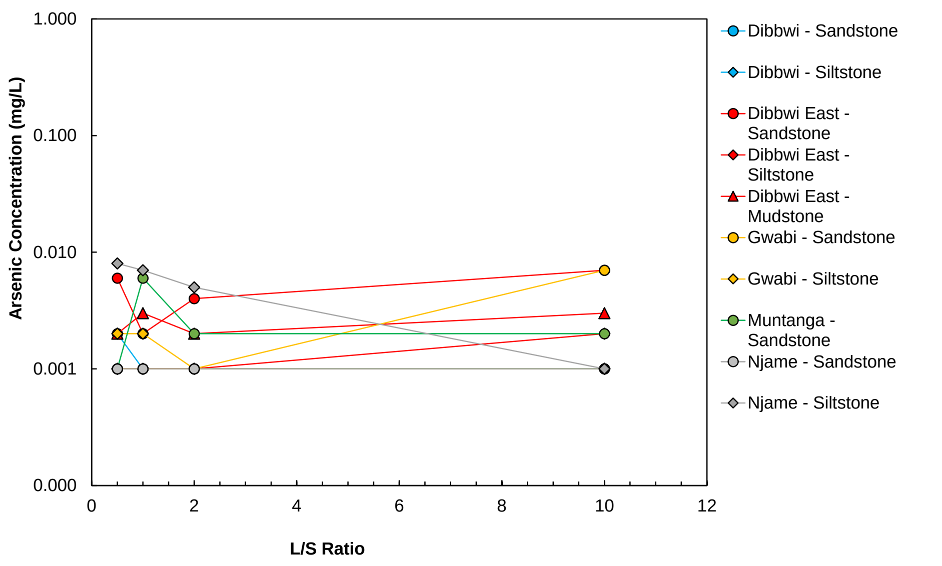


Figure 20-25: LEAF 1316 Test Results L/S Ratio vs Arsenic Concentration

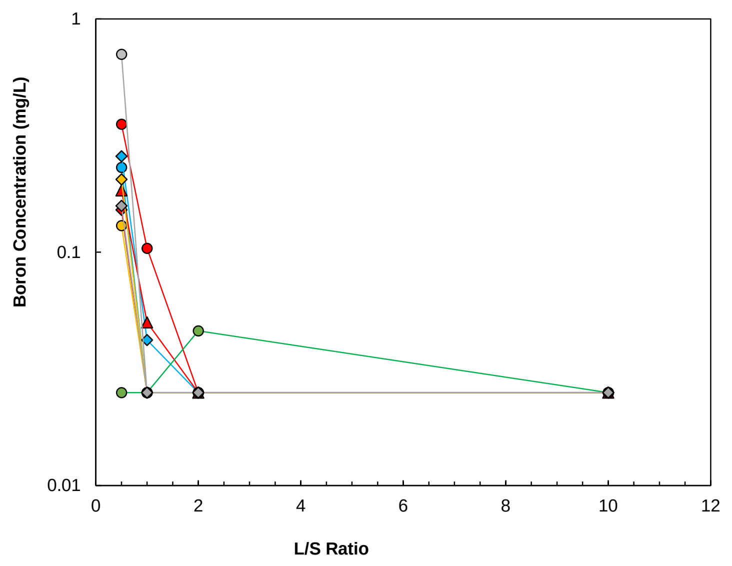


Figure 20-26: LEAF 1316 Test Results L/S Ratio vs Boron Concentration

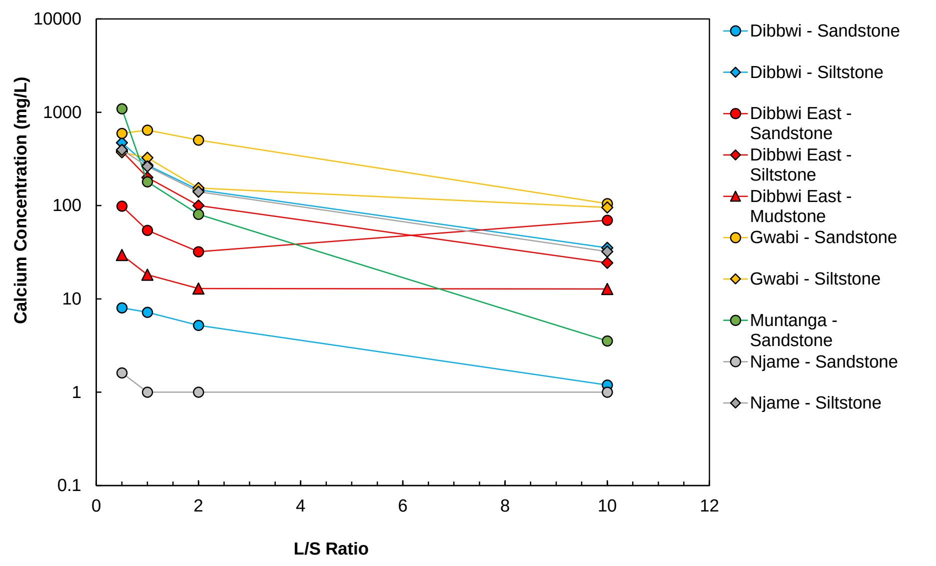


Figure 20-27: LEAF 1316 Test Results L/S Ratio vs Calcium Concentration

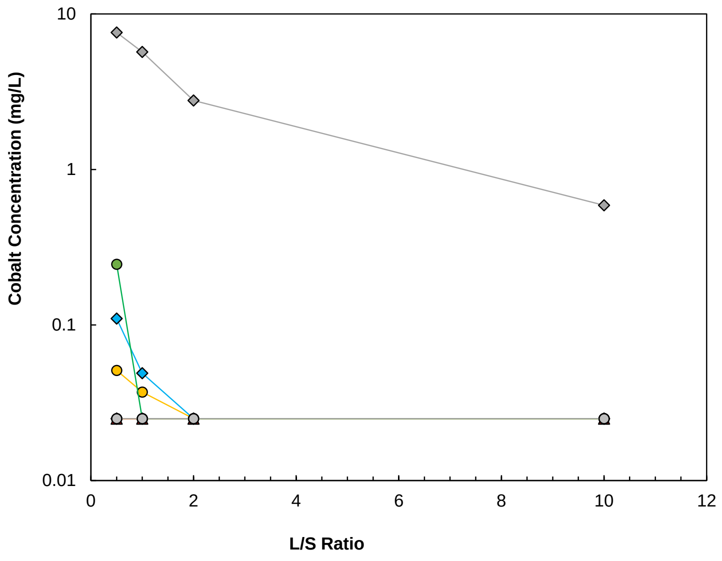


Figure 20-28: LEAF 1316 Test Results Cobalt Concentration vs L/S Ratio

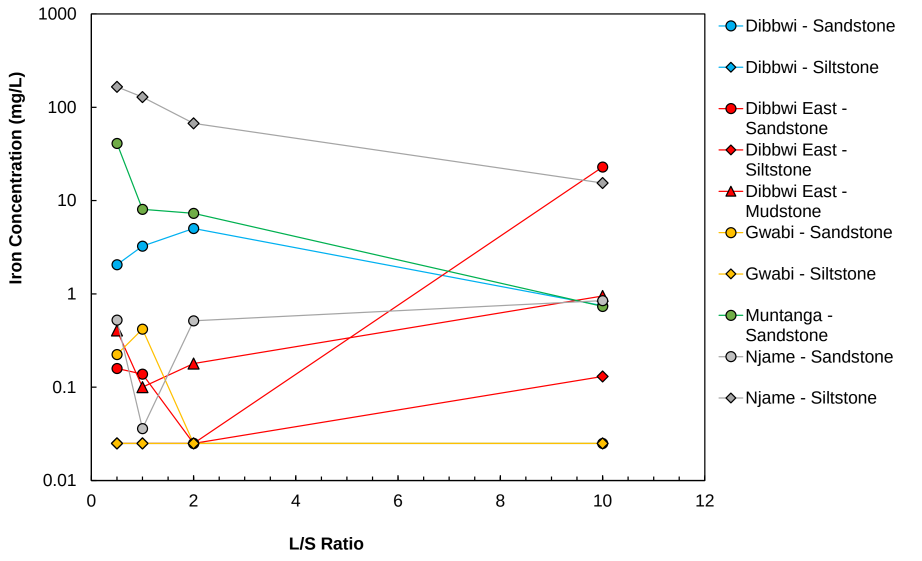


Figure 20-29: LEAF 1316 Test Results Iron vs L/S Ratio

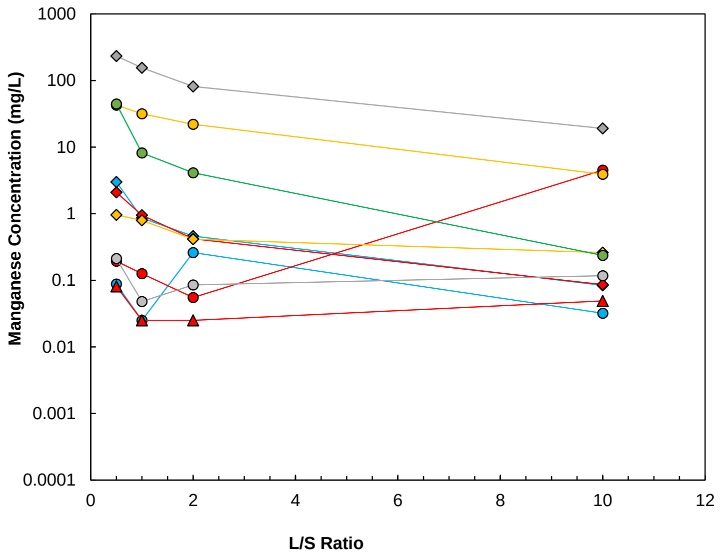


Figure 20-30: LEAF 1316 Test Results Manganese Concentration vs L/S Ratio

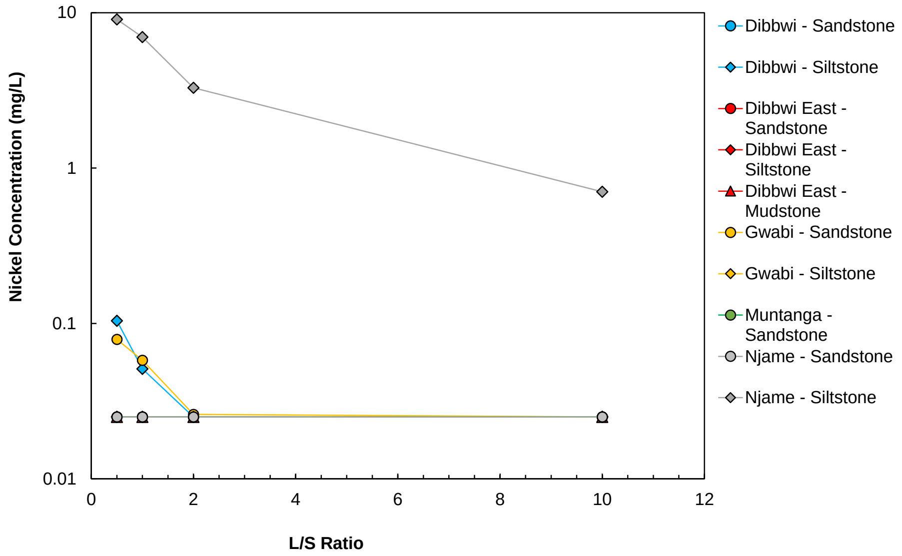


Figure 20-31: LEAF 1316 Test Results Nickel Concentration vs L/S Ratio

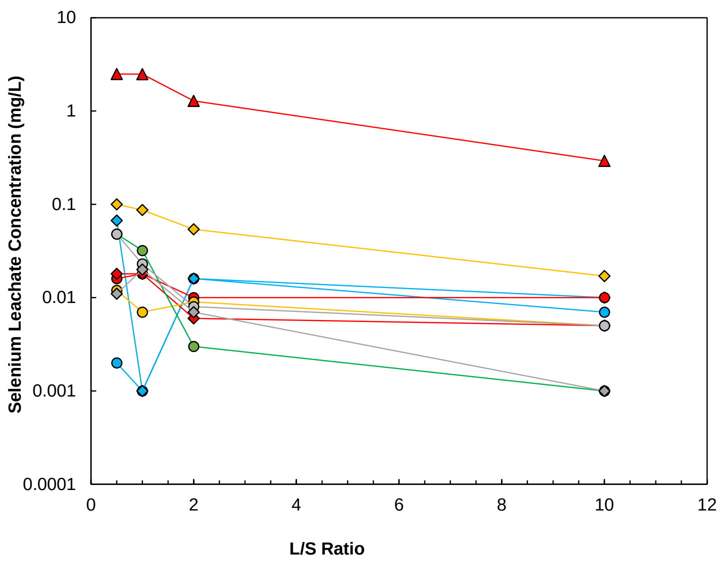


Figure 20-32: LEAF 1316 Test Results Selenium Concentration vs L/S Ratio

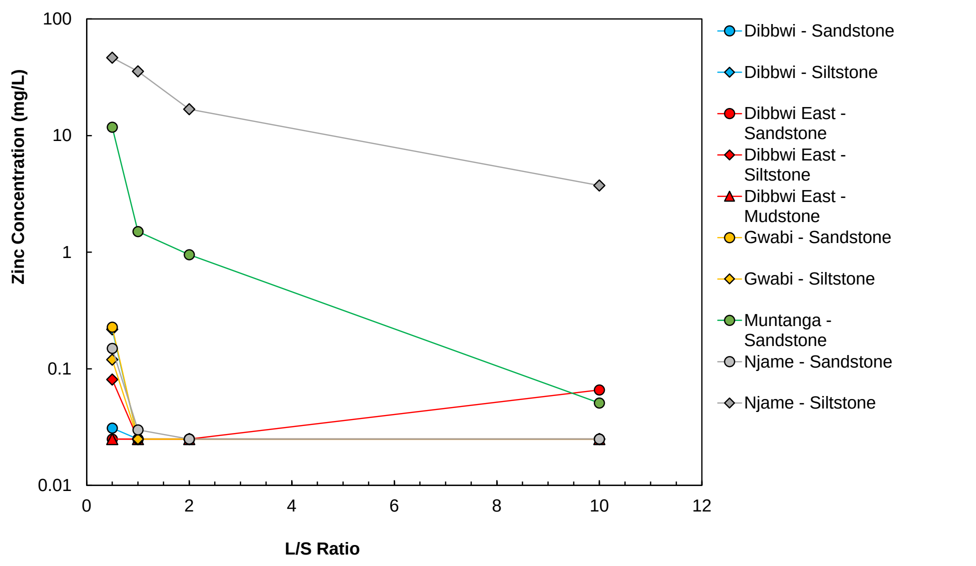


Figure 20-33: LEAF 1316 Test Results Zinc Concentration vs L/S Ratio

20.6.9.Spent ore geochemical testwork results

Water chemistry results for the third wash cycle conducted on the 6m columns prior to their dismantling are presented in Table 20-5 for reference. All samples reported acidic pH 1.7 to 2.8. The majority of parameters in this wash cycle were consistently reported below RDL for all samples although there were some instances where the RDL was relatively high, for example for uranium (<2.0 mg/L), chromium (<2.0 mg/L), mercury (<0.01 mg/L) and nickel (2.0 mg/L). Concentrations of iron and manganese were typically high ranging from 7.4 to 220 mg/L and from 11 to 260 mg/L respectively. One sample reported high copper (8.1 mg/L) but all other samples reported concentrations of copper close to or at RDL. Sulfur concentrations were also high (180 to 1 000 mg/L which would equate to approximately 540 to 3 000 mg/L as sulfate).

Table 20-6 presents ABA and multi-element analyses for each of the column residue samples. These results are based on samples which were reconstituted from the 1 m intervals following column dismantling. Total sulfur ranges between 0.14 and 0.74 % and sulfide sulfur ranges between 0.06 and 0.60 % although sulfate is consistently at RDL (<0.01 %) possibly due to inconsistencies in reporting limits or presence of other oxyanion forms of sulfur (Figure 20-34). AP calculated from sulfide sulfur ranged between 1.9 and 19 kg $CaCO_3$/t. NP ranged between 3.7 and 11 kg $CaCO_3$/t. Figure 20-35 presents the AP versus NP results and shows that two samples have uncertain acid generating potential (Comp 1 Dibbwi East Oxidised and Comp 5 Dibbwi Main Oxidised) all other samples were PAG. If AP was calculated from total sulfur, to be conservative based upon the differences in the total sulfur and sulfide results, all samples would be classified as PAG. Net neutralisation potential (NNP) ranges between -8.0 and 1.8 kg $CaCO_3$/t which results in an uncertain acid potential classification.

Uranium was not analysed in the reconstituted solid samples but was analysed in the individual 1 m intervals during the column dismantling. Concentrations of uranium ranged from 9.1 to 59 ppm relative to an average crustal abundance of 1.8 ppm. The results of the multi element analysis on the reconstituted solid samples showed concentrations of aluminium, cobalt, chromium, copper, magnesium, manganese, nickel, lead, vanadium and zinc within less than three times average crustal abundance. Cadmium is consistently reported at RDL (<10 ppm), but the RDL is more than twelve times average crustal abundance so cannot be assessed. Molybdenum concentrations are elevated relative to average crustal abundance (1.5 ppm); concentrations range between 21 and 27 ppm.

The TCLP leachate analyses on the column residue samples are presented in Table 20-7. Most parameter results are low; concentrations of arsenic, boron, cadmium, cobalt, mercury, molybdenum, nickel, lead, phosphate, antimony, selenium, silver, tin, thorium, vanadium and zirconium are below RDL in all samples, although detection limits for mercury and selenium are relatively high (0.1 mg/L). Uranium concentrations range from <0.1 to 0.4 mg/L. These laboratory generated leachates are not directly comparable to any water quality standards but just for reference to provide context to the results, the WHO drinking water standard for uranium is 0.03 mg/L. Fluoride concentrations are high ranging from 240 to 3 000 mg/L; for reference, the Zambian Effluent Discharge (ZED) limit for fluoride is 2 mg/L. Manganese concentrations are also relatively high, ranging from 0.5 to 12 mg/L (for reference, the ZED limit for manganese is 1 mg/L).

There are no obvious trends apparent between the different column samples.

Table 20-5: Analysis of Wash 3 Solution from Column Rinse

Description	pH	Ag	Al	As	As	B	Ba	Be	Bi	Ca	Cd	Cd	Cl	Co	Cr	Cu	F	Fe	Hg	K
		ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
Comp 1 Wash 3	1.7	0.01	3.2	2.0	0.049	0.01	0.03	0.01	0.01	130	2.0	0.01	77	2.0	2.0	2.0	0.5	7.4	0.01	10
Comp 2 Wash 3	2.2	0.01	45	2.0	0.022	0.01	0.01	0.01	0.01	513	2.0	0.01	64	2.0	2.0	8.1	2.3	200	0.01	0.4
Comp 3 Wash 3	2.5	0.01	7.0	2.0	0.043	0.01	0.01	0.01	0.01	98	2.0	0.01	31	2.0	2.0	2.0	0.5	216	0.01	7.4
Comp 4 Wash 3	2.6	0.01	5.1	2.0	0.015	0.01	0.01	0.01	0.01	527	2.0	0.01	16	2.0	2.0	2.2	2.0	207	0.01	0.4
Comp 5 Wash 3	2.8	0.01	2.0	2.0	0.041	0.01	0.01	0.01	0.01	143	2.0	0.01	14	2.0	2.0	2.0	0.5	59	0.01	0.5

Description	Mg	Mn	Mo	Mo	Na	Nb	Ni	Pb	Pb	PO4	S	Sb	Se	Si	Sn	U	V	Zn
	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
Comp 1 Wash 3	6.2	11	2.0	0.01	54	0.01	2.0	2.0	0.01	33	183	0.01	0.01	73	0.01	2.0	2.0	2.0
Comp 2 Wash 3	133	130	2.0	0.01	50	0.01	2.0	2.0	0.01	26	1040	0.01	0.01	63	0.01	2.0	2.0	2.0
Comp 3 Wash 3	18	23	2.0	0.01	16	0.01	2.0	2.0	0.01	15	393	0.01	0.01	62	0.01	2.0	2.0	2.0
Comp 4 Wash 3	114	263	2.0	0.01	20	0.01	2.0	2.0	0.01	13	1020	0.01	0.01	67	0.01	2.0	2.0	2.0
Comp 5 Wash 3	30	12.9	2.0	0.01	14	0.01	2.0	2.0	0.01	14	277	0.01	0.01	93	0.01	2.0	2.0	2.0

Blue italics indicate results less than the reported detection limit (RDL)

Table 20-6: Summary of ABA and Solids Elemental Analysis on Spent Ore Materials (Reconstituted Column Samples)

Sample Name	Description	AP	NP	NNP	NP/AP	Total S	Sulfate S	Sulfide S	Al	Al_2O_3	CaO	Cd	Co	Co	Cr	Cr_2O_3	Cu	Cu
		kg $CaCO_3$/t				%	%	%	%	%	%	ppm	%	ppm	ppm	%	%	ppm
Average Crustal Abundance (Mason 1966)>>									8.1			0.2		25	100			55
W0119104/1	Comp 1 res.	1.9	3.7	1.8	2.0	0.14	0.01	0.06	3.9	7.4		10	0.05	11	159	0.07	0.05	25
W0119104/2	Comp 2 res.	19	11	-8.0	0.57	0.74	0.01	0.60	4.6	8.7	0.4	10	0.05	10	180	0.07	0.05	58
W0119168/1	Comp 3 res.	11	4.5	-6.5	0.40	0.64	0.01	0.36	4.3	8.2	0.1	10	0.05	10	202	0.07	0.05	10
W0119168/2	Comp 4 res.	8.1	6.0	-2.1	0.73	0.46	0.01	0.26	2.2	4.1	0.3	10	0.05	10	215	0.07	0.05	21
W0119168/3	Comp 5 res.	5.7	7.4	1.7	1.3	0.24	0.01	0.18	3.9	7.3	0.1	10	0.05	10	151	0.07	0.05	10
Sample Name	**Description**	**Fe_2O_3**	**Mg**	**MgO**		**Mn**	**MnO**	**Mo**	**Ni**	**Ni**	**Pb**	**PbO**	**SiO_2**	**TiO_2**	**V**	**V_2O_5**	**Zn**	**Zn**
		%	%	%		ppm	%	ppm	%	ppm	ppm	%	%	%	ppm	%	%	ppm
Average Crustal Abundance (Mason 1966)>>			2.1			950		1.5		75	13				135			70
W0119104/1	Comp 1 res.	3.6	0.12	0.20		-	0.17	22	0.05	51	26	0.06	0.32	0.09	46	0.05	10	34
W0119104/2	Comp 2 res.	3.1	0.18	0.30		305	0.06	23	0.05	61	26	0.06	0.40	0.09	49	0.05	10	17
W0119168/1	Comp 3 res.	2.6	0.07	0.12		132	0.06	21	0.05	42	24	0.06	0.42	0.09	45	0.05	10	19
W0119168/2	Comp 4 res.	3.3	0.11	0.18		-	0.10	27	0.05	46	10	0.06	0.25	0.09	42	0.05	10	20
W0119168/3	Comp 5 res.	1.7	0.07	0.12		63	0.06	24	0.05	44	21	0.06	0.38	0.09	45	0.05	10	12

Blue italics indicate results less than the reported detection limit (RDL)
AP calculated from sulfide sulfur x 31.25
Orange bold indicates sample is classified as having uncertain acid generating potential according to NP/AP or NNP
Red bold indicates sample is classified as PAG according to NP/AP or NNP

White	Indicates <3 times average crustal concentrations
Green	Indicates between 3 and 6 times average crustal concentrations
Yellow	Indicates between 6 and 12 times average crustal concentrations
Pink	Indicates >12 times average crustal concentrations
Grey	Indicates result at RDL but RDL exceeds 3 times average crustal abundance

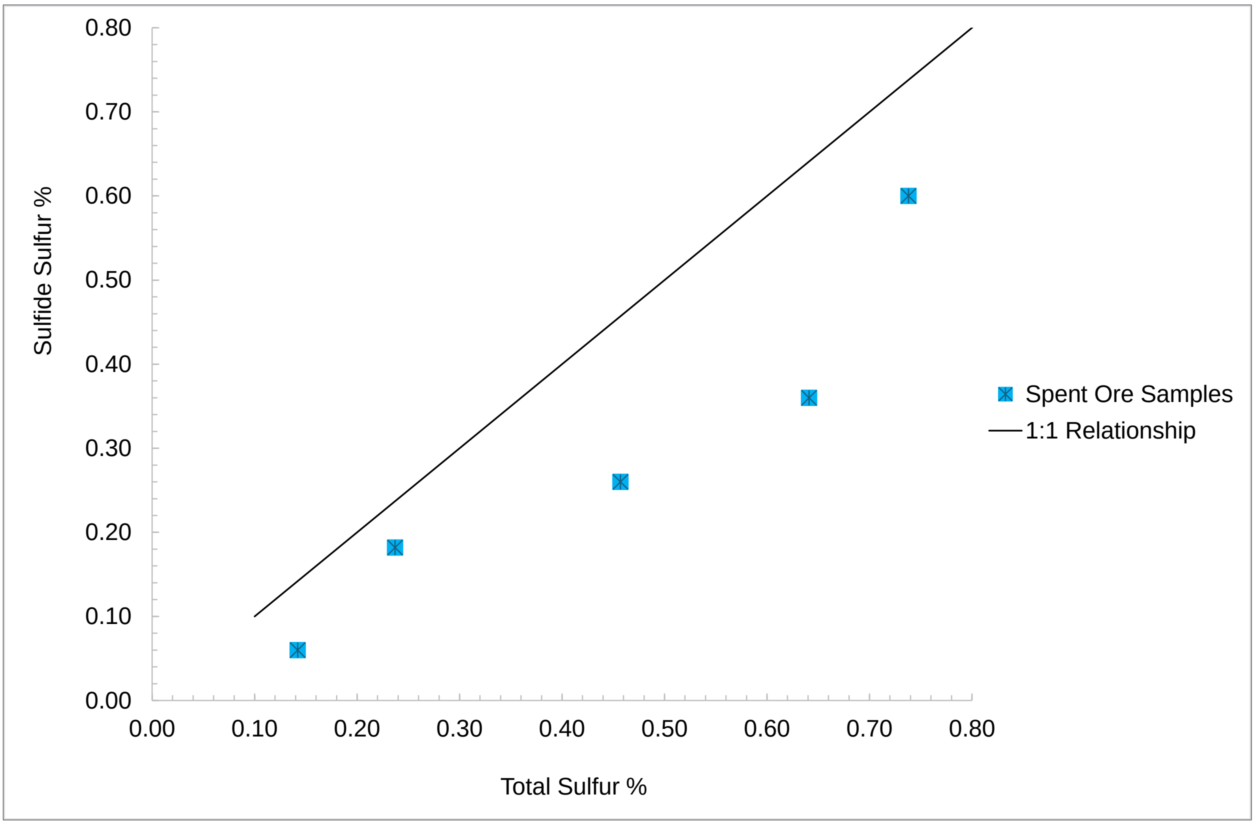


Figure 20-34: Total sulfur vs sulfide sulfur for spent ore samples

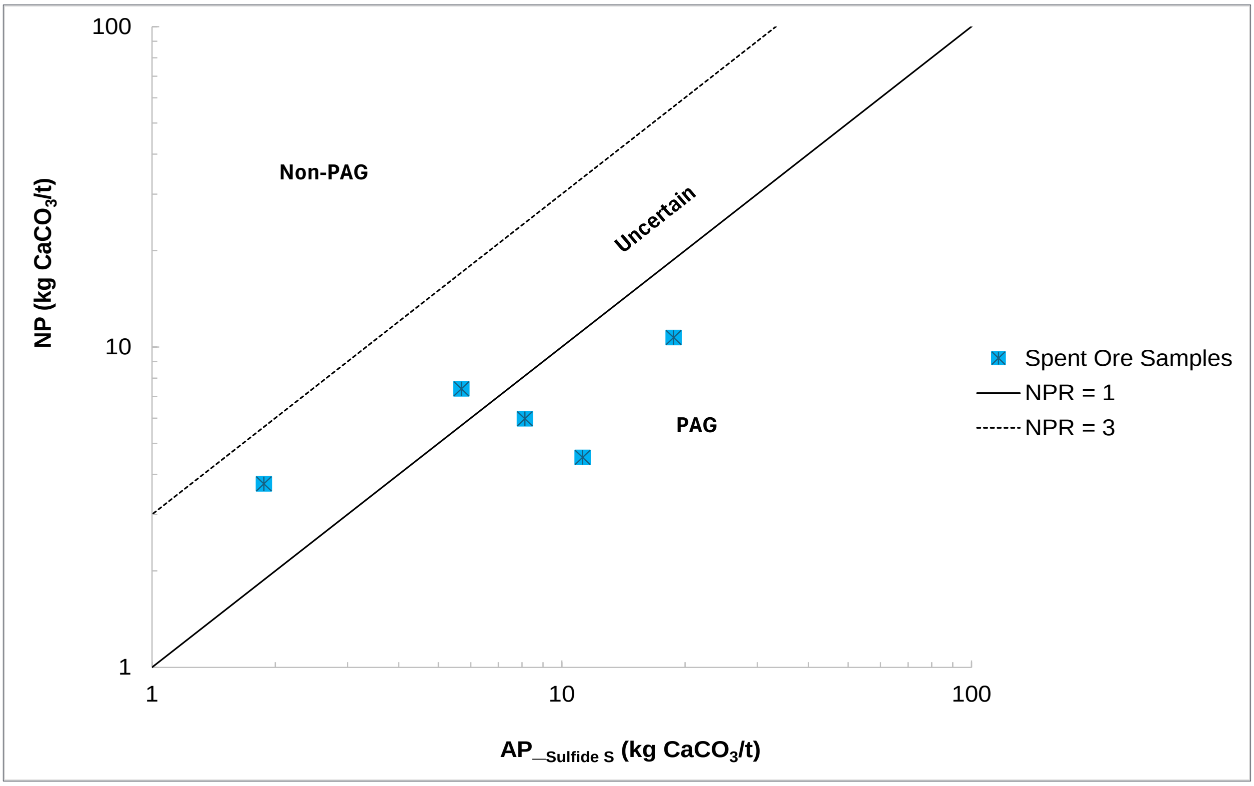


Figure 20-35: AP vs NP for spent ore samples

Table 20-7: Summary of TCLP Leachate Analysis on Spent Ore Materials (Reconstituted Column Samples)

Sample Name	Description	Ag	Al	As	B	Ba	Ca	Cd	Cl	Co	Cr	Cu	F	Fe	Hg	Mg	Mn
		mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
W0119104/1	Comp 1 res.	0.1	0.93	0.1	0.1	1.0	9.5	0.1	0.33	0.1	0.1	0.15	300	0.05	0.1	4.5	7.3
W0119104/2	Comp 2 res.	0.1	0.75	0.1	0.1	0.27	78	0.1	1.1	0.1	0.1	0.18	3001	0.05	0.1	14	3.9
W0119168/1	Comp 3 res.	0.1	1.0	0.1	0.1	0.22	73	0.1	3.1	0.1	0.22	0.1	243	3.7	0.1	11	11
W0119168/2	Comp 4 res.	0.1	1.1	0.1	0.1	0.26	73	0.1	4.6	0.1	0.23	0.1	298	1.3	0.1	11	12
W0119168/3	Comp 5 res.	0.1	2.9	0.1	0.1	1.1	9.8	0.1	0.97	0.1	0.16	0.1	311	4.2	0.1	5.2	0.5
Sample Name	**Description**	**Mo**	**Ni**	**Pb**	**PO4**	**S**	**Sb**	**Se**	**Si**	**Sn**	**Sr**	**Th**	**Ti**	**U**	**V**	**Zn**	**Zr**
		mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
W0119104/1	Comp 1 res.	0.1	0.1	0	0.5	0.5	0.1	0.1	11	0.1	0.18	0.1	0.1	0.16	0.1	0.30	0.1
W0119104/2	Comp 2 res.	0.1	0.1	0.1	0.5	3.8	0.1	0.1	13	0.1	0.10	0.1	0.1	0.38	0.1	0.16	0.1
W0119168/1	Comp 3 res.	0.1	0.1	0.1	0.5	68	0.1	0.1	9.9	0.1	0.11	0.1	0.1	0.37	0.1	0.17	0.1
W0119168/2	Comp 4 res.	0.1	0.1	0.1	0.5	70	0.1	0.1	8.6	0.1	0.12	0.1	0.1	0.39	0.1	0.20	0.1
W0119168/3	Comp 5 res.	0.1	0.1	0.1	0.5	12	0.1	0.1	10	0.1	0.1	0.1	0.1	0.1	0.1	0.15	0.1

Blue italics indicate results less than the reported detection limit (RDL)

20.6.10. Gypsum waste and processing bleed stream testwork results

Table 20-8 presents ABA results for the gypsum waste stream samples. Mintek laboratories repeated the ABA testwork due to missing parameters and results for round 1 (Rnd 1) and round 2 (Rnd 2) are included. Paste pH ranged between 6.8 and 7.6. Total sulfur was reported at 20 % in both the pH 2 and pH 4 solids. As expected in these gypsum samples, sulfur is present as sulfate with concentrations reported at 56 % to 58 % as SO_4. Sulfide sulfur was reported at 0.06 to 0.07 % in the Rnd 1 sampling but anomalously higher (0.12 to 0.15 %) in the Rnd 2 sampling. Since these samples have been leached with sulfuric acid and hydrogen peroxide, trace amounts of sulfide are not expected and may be due to analytical error. AP is calculated from sulfide sulfur in Rnd 1. NP results were also vastly different between Rnd 1 and Rnd 2 (24 to 25 kg $CaCO_3$/t in Rnd 1 and -9.7 to -4.7 kg$CaCO_3$/t in Rnd 2) this affects the classification of acid generating potential which ranges from non-PAG through to uncertain or PAG. At the time of writing, these results are under investigation with Mintek as the sulfide analysis of the gypsum is clearly an error.

Multi-element analyses results for the gypsum solids are presented in Table 20-9. The results are compared to average crustal abundance to provide an indication of enrichment. As expected, the gypsum tailings primarily comprised calcium and sulfate. Calcium concentrations ranged between 23 and 24 % which is between three and six times average crustal abundance. Uranium concentrations were also elevated in the gypsum solids ranging from 59 ppm in the pH 2 solid sample to 560 ppm in the pH 4 solid sample. The uranium results were greater than twelve times average crustal abundance. The following parameters were consistently below detection limit in both gypsum solid samples: aluminium, cobalt, chromium, copper, magnesium, manganese, nickel, lead, silicon, titanium, vanadium and zinc. The detection limits for cobalt, chromium, copper, nickel and lead were above three times crustal abundance so enrichment relative to ACA could not be assessed.

Analyses of the liquors produced in the TCLP tests showed that most parameters were reported less than detection limit. Uranium concentrations ranged from <2.0 mg/L in the pH 2 gypsum sample to 8.5 mg/L in the pH 4 gypsum sample. Fluoride concentrations were elevated in the TCLP leachates, ranging from 270 to 310 mg/L.

Shake flask results undertaken on the gypsum samples are presented in Table 20-11. The test provides an indication of any readily soluble constituents. pH ranged from acidic to mildly acidic (pH 3.3 to 5.8). Lowest pH was reported in the sample labelled as pH 4 gypsum and suggests that some buffering reaction is taking place as the pH 2 gypsum sample is reacted during the test. Most parameter results are low; concentrations of aluminium, arsenic, cadmium, cobalt, chromium, copper, iron, lithium, magnesium, manganese, molybdenum, nickel, lead, titanium, vanadium and zinc were consistently below RDL although the RDL for arsenic, cadmium, chromium and lead were relatively high (2.0 mg/L).

Uranium concentrations ranged from <2.0 to 44 mg/L with highest concentrations reported in the pH 4 gypsum sample. The laboratory generated leachates are not directly comparable to any water quality standards but just for reference to provide context to the results, the WHO drinking water standard for uranium is 0.03 mg/L. The pH 2 gypsum sample reported fluoride at 31 mg/L Fluoride concentrations were reported at <0.50 and 31 mg/L in the pH 4 and pH 2 gypsum samples respectively; for reference, the Zambian Effluent Discharge (ZED) limit for fluoride is 2 mg/L.

Table 20-8: Summary of ABA results on gypsum waste stream materials

Sample Name	Description	Paste pH	Total Sulfur	Sulfate [as SO4]	Sulfide Sulfur [Rnd 1]	Sulfide Sulfur [Rnd 2]	NP [Rnd 1]	NP [Rnd 2]	AP	NP [Rnd 1] /AP	NP [Rnd 2] /AP	NNP [Rnd 1]	NNP [Rnd2]
		pH Units	%	%	%		kg $CaCO_3$/t		kg $CaCO_3$/t			kg $CaCO_3$/t	
W0119779/1	pH 2 solids	7.6	20	56	0.06	0.15	25	-9.7	1.9	13	-5.2	23	-12
W0119784/1	pH 4 solids	6.8	20	58	0.07	0.12	24	-4.7	2.2	11	-2.1	22	-6.9

Table 20-9: Summary of solids elemental analysis on gypsum waste stream materials

Sample Name	Description	Al	Ca	Co	Cr	Cu	Fe	Mg	Mn	Ni	Pb	Si	Ti	V	Zn	Th	U
		%	%	%	%	%	%	%	%	%	%	%	%	%	%	ppm	ppm
Average Crustal Abundance (Mason 1966)>>		8.1	3.6	0.0025	0.01	0.0055	5.0	2.1	0.095	0.0075	0.0013	28	0.44	0.014	0.007	7.2	1.8
W0119779/1	pH 2 solids	0.05	24		0.05	0.05	0.073	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.7	59
W0119784/1	pH 4 solids	0.05	23	0.05	0.05	0.05	0.16	0.05	0.05	0.05	0.05	0.12	0.05	0.05	0.05	1.7	560

Blue italics indicate results less than the reported detection limit (RDL)

Indicates <3 times average crustal concentrations
Indicates between 3 and 6 times average crustal concentrations
Indicates between 6 and 12 times average crustal concentrations
Indicates >12 times average crustal concentrations
Indicates result at RDL but RDL exceeds 3 times average crustal abundance

Table 20-10:SPLP Test Results on Gypsum Waste Stream Materials

Description	Chloride	Fluoride	Phosphate	Sulfate	Al	As	Ba	Ca	Cd	Co	Cr	Cu	Fe	Hg	Mg	Mn	Mo	Ni	Zr
	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
GR2 pH 2 gypsum ppt	0.22	270	0.50	520	0.05	0.10	0.10	730	0.10	0.10	0.10	0.10	0.13	0.10	0.26	0.10	0.44	0.10	0.10
GR2 pH 4 gypsum ppt	0.36	310	0.50	580	0.05	0.00	0.10	820	0.00	0.10	0.10	0.10	0.65	0.10	0.36	0.10	0.46	0.10	0.10

Description	Pb	S	Sb	Se	Si	Sn	Sr	Ti	U	V	Zn
	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
GR2 pH 2 gypsum ppt	0.10	600	0.10	0.10	0.05	0.10	7.0	0.10	2.0	0.10	0.20
GR2 pH 4 gypsum ppt	0.00	670	0.10	0.10	0.05	0.00	4.0	0.10	8.5	0.00	0.52

Table 20-11: Summary of shake flask solution analyses on gypsum waste stream materials

Sample Name	Description	pH	Conductivity	Fluoride	PO4	Chloride	Al	As	Ca	Cd	Co	Cr	Cu	Fe
		pH	mS/m	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
W0119266A/1	Shake sln pH2	5.8	218	31	8.7	4.2	2.0	2.0	603	2.0	2.0	2.0	2.0	2.0
W0119266A/2	Shake sln pH4	3.3	246	0.50	2.2	8.0	2.0	2.0	653	2.0	2.0	2.0	2.0	2.0

Sample Name	Description	K	Li	Mg	Mn	Mo	Na	Ni	Pb	S	Si	Ti	U	V	Zn
		mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
W0119266A/1	Shake sln pH2	5.5	2.0	3.3	2.0	2.0	5.5	2.0	2.0	483	3.1	2.0	2.0	2.0	2.0
W0119266A/2	Shake sln pH4	2.3	2.0	2.0	2.0	2.0	5.6	2.0	2.0	535	2.3	2.0	44	2.0	2.0

Blue italics indicate results less than the reported detection limit (RDL)

20.6.11. Ore sorter rejects

Ore sorter reject material will report to the waste rock dumps at each of the deposits. Reject material from the ore sorting process was not available for geochemical characterisation so is not currently included in this assessment. Analysis should be undertaken to cross check that the ore sorter rejects materials do not differ significantly from the waste rock materials which have been characterised.

20.6.12. Predictive numerical calculations

High-level numerical calculations were performed to assess the possible water quality of lakes developing in the Dibbwi East and Muntanga open pits once operations cease. The predictive water quality for each pit are based on the mine design (December 2024 and January 2025; Ukwazi), the hydrogeological model (January 2025, SRK), and the geochemical (static) test results for each of the major lithological unit (section 20.6.8).

During operations, the pits are expected to be dewatered and thus relatively dry as water from precipitation runoff and groundwater inflows are expected to be collected in a pit sump. Once the open pit mining operations cease, dewatering will be discontinued while diversions of streams will remain in place, leading to a gradual formation of a lake in each pit.

Pit lake chemistry prediction is based on mixing of relative proportions of the chemical loads from different sources such as groundwater inflow, pit wall runoff, and direct precipitation onto the lake. These source contributions to the pit lake are derived from the water balance and are schematically illustrated in a conceptual model presented in Figure 1 (example of Dibbwi East pit and WRD 1).

A series of assumptions and scaling factors were applied to the input data and these are summarized in SRK (2025).

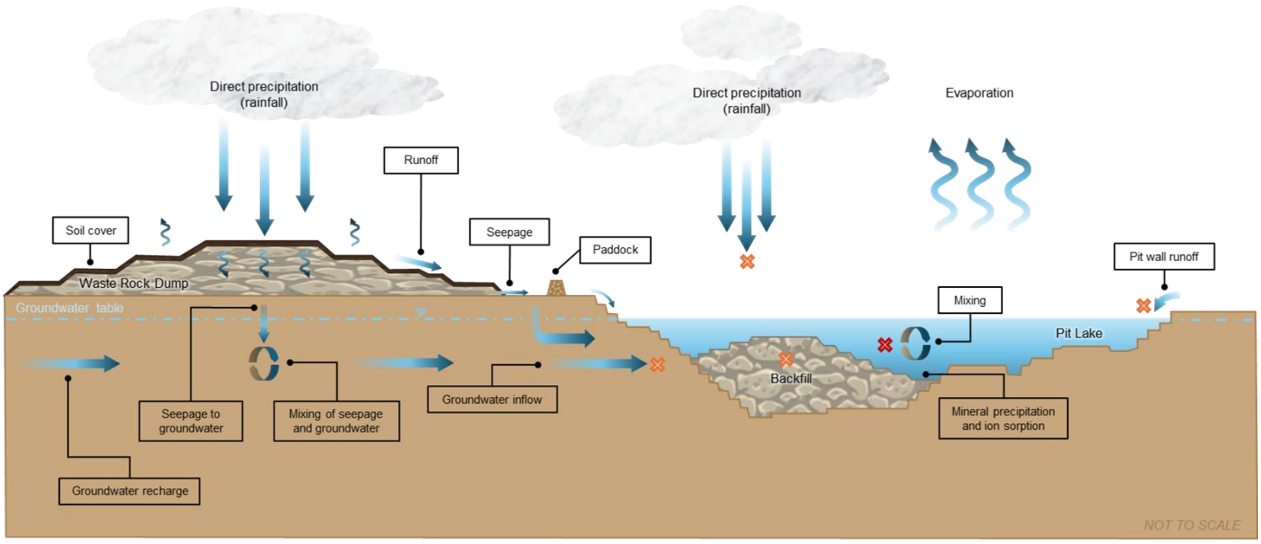


Figure 20-36: Conceptual model of pit lake

20.6.12.1. Results

Predicted pit lake chemistries for the Dibbwi East and Muntanga pits determined by very simplified, high-level model calculations are presented in Table 20-12 for selected years in 10-year time steps and screened against water quality guideline values. Parameters for which guideline values are available are not exceeded in the predicted pit lake concentrations at any modelled time step.

Chemical loads resulting from the weathering of exposed pit wall rock through rainfall and the chemical loads introduced from the inflowing groundwater present the main chemical contribution to the pit lake chemistry. During the initial years of rapid groundwater rebound, the pit lake chemistry mostly mirrors that of the groundwater in each pit. As groundwater influx stabilises at lower rates over time, while rainfall continues at the same annual volumes, contributions from exposed pit wall rock weathering increase and modify the pit lake chemistry towards slightly higher concentrations of readily leached parameters. Such parameters include, for example, chloride, aluminium, potassium, manganese, and uranium, that were leached during static testing and thus also tend to show higher concentrations in the predicted pit lake compared to scaled groundwater.

Although Dibbwi East pit lake pH is initially slightly acidic (pH 5.2 in year 2), the pH gradually increases to pH 6 in year 42. For Muntanga, predicted pit lake water pH increases from 6.2 in year 2 to pH 6.7 in year 12. These pH conditions are in line with what is expected from the geochemical characteristics of the tested lithologies that had low to uncertain acid rock drainage potential. However, it is not clear whether sandstone and siltstone with higher potential for acid production would produce acidic runoff if exposed to oxidative weathering over longer periods of time than the 48 hours of the static leach testing. This is particularly important for the Muntanga pit lake, for which the predicted chemistry is based on one static leach test sample of potentially acid generating sandstone (which produced a static leach test pH of ~7).

Overall, predicted sulfate concentrations in both pit lakes remain below 10 mg/L after year 2, which is generally very low and may not represent the actual concentrations in the pit. Nitrate loads predominantly mirror those in scaled groundwater, which may be the main source of nitrate as blast residuals are most likely flushed from freshly blasted wall rock with the first rainfall. Chloride is lowest in scaled groundwater, hence contributions to the predicted chloride concentrations in the pit lakes of < 1 to 10 mg/L mostly originate from pit wall runoff and rainwater.

Predicted major cation concentrations are generally very low in both pit lake (< 1 and < 10 mg/L) and generally below water quality guideline concentrations. Aluminium, calcium, potassium, and magnesium loads from exposed pit wall rock weathering result in slightly increased predicted concentrations in the pit lakes compared to the scaled groundwater values, while iron loads from the wall rock shows only a minor effect on the overall predicted pit lake chemistry. Sodium loads introduced through rainwater and pit wall runoff result in only minor increases in the predicted pit lake concentrations compared to scaled groundwater.

Predicted trace element concentrations are generally below water quality guideline concentrations. Load contributions predominantly originate from the weathering of exposed pit wall rock. Predicted concentrations in Dibbwi East for example, manganese and uranium, remain below 0.02 mg/L after year 5.

Predicted chemical compositions of the Dibbwi East and Muntanga pit lakes show no exceedances over the Kafue Catchment Water Quality Standards, Zambian Bureau of Standards for Drinking Water, or Zambian Bureau of Standards for Effluent Discharge Limits. Overall, the high-level predictions indicate slightly acidic to circumneutral pit lakes (pH ~5 to 6.7) with major and trace element concentrations becoming more diluted over time despite expected evapoconcentration.

Sulfate and uranium contents are predicted to be very low, with concentrations of < 10 mg/L sulfate and < 0.03 mg/L uranium in Dibbwi East, and < 1 mg/L sulfate and < 0.0003 mg/L uranium in Muntanga as soon as 2 years after flooding of the pits. Although there are no guidelines provided for uranium in the water quality standards used, these concentrations are below the provisional WHO drinking water guideline values of 0.03 mg/L for uranium (WHO, 2022) – for Dibbwi East from year 3 after flooding pits, and for Muntanga right from the first year of flooding the pit.

Table 20-12: Results of pit lake predictions

Predicted Pit Lake Chemistry	Water Quality Guidelines				Predicted Pit Lake Chemistry for Dibbwi East					Predicted Pit Lake Chemistry for Muntanga		
Parameter	Units	KC WQS	ZABS DWS	ZABS EDL	After 2 years	After 12 years	After 22 years	After 32 years	After 42 years	After 2 years	After 6 years	After 12 years
pH		6 - 9	6 - 8	6 - 9	5.2	5.8	5.9	6	6	6.1	6.6	6.7
Total Dissolved Solids	mg/L	350	1000	3000	31	8	6.5	5.1	5.4	2.3	0.59	0.45
Total Alkalinity as $CaCO_3$	mg/L	400	--	--	18	4.4	3.4	2.5	2.7	2.2	0.6	0.47
Cl	mg/L	50	250	800	3.9	1.2	1	0.91	0.99	0.18	0.078	0.049
F	mg/L	--	--	--								
SO4	mg/L	60	400	1500	8.2	2	1.5	1.2	1.2	1.1	0.3	0.23
Nitrate as N	mg/L	6	10	50	0.12	0.028	0.021	0.016	0.016	0.016	0.0044	0.0034
Ammonia as N	mg/L	--	--	--	0.0052	0.0014	0.0013	0.001	0.0011	0.0016	0.00059	0.00052
Phosphate	mg/L	0.1	--	--	0.0077	0.0018	0.0014	0.001	0.001	0.001	0.00026	0.0002
Ag	mg/L	--	0.05	0.1	0.00044	0.00011	0.000081	0.000061	0.000063	0.000074	0.000021	0.000017
Al	mg/L	--	0.2	2.5	0.011	0.0029	0.0024	0.0019	0.0021	0.019	0.0068	0.0059
As	mg/L	0.05	0.01	0.05	0.00038	0.000092	0.000071	0.000053	0.000056	0.00006	0.000017	0.000014
B	mg/L	--	--	0.5	0.012	0.0033	0.0029	0.0023	0.0025	0.00085	0.00029	0.00026
Ba	mg/L	--	0.7	0.5	0.003	0.00078	0.00063	0.00049	0.00052	0.00063	0.0002	0.00017
Be	mg/L	--	--	--	0.0013	0.00037	0.00032	0.00026	0.00029	0.00041	0.00015	0.00013
Ca	mg/L	60	200	-	6.6	1.7	1.3	1	1.1	0.6	0.16	0.12
Cd	mg/L	0.002	0.003	0.5	0.000099	0.000026	0.000021	0.000016	0.000018	0.000023	0.0000075	0.0000064
Co	mg/L	0.1	0.5	1	0.0014	0.00039	0.00034	0.00027	0.0003	0.00043	0.00015	0.00013
Cr	mg/L	-	0.05	0.1	0.0014	0.00039	0.00034	0.00027	0.00029	0.00042	0.00015	0.00013
Cu	mg/L	1.5	1	2	0.0011	0.00028	0.00023	0.00018	0.00019	0.00024	0.000079	0.000067
Fe	mg/L	0.7	0.3	3	0.065	0.016	0.012	0.0087	0.009	0.0096	0.0026	0.002
Hg	mg/L	-	0.001	0.002	0.00013	0.000033	0.000026	0.00002	0.000022	0.000027	0.0000085	0.0000072
K	mg/L	--	--	--	0.81	0.22	0.17	0.14	0.15	0.092	0.028	0.021
Li	mg/L	--	--	--	0.0048	0.0012	0.00089	0.00066	0.00069	0.00059	0.00015	0.00012
Mg	mg/L	40	150	500	0.99	0.25	0.19	0.15	0.16	0.13	0.038	0.028
Mn	mg/L	0.2	0.1	1	0.057	0.015	0.012	0.0092	0.0098	0.006	0.0018	0.0014
Mo	mg/L	--	--	5	0.0016	0.00044	0.00038	0.0003	0.00033	0.00045	0.00016	0.00014
Na	mg/L	20	200	-	3.5	0.91	0.68	0.56	0.59	0.53	0.15	0.11
Ni	mg/L	-	--	0.5	0.0018	0.00047	0.0004	0.00032	0.00034	0.00047	0.00016	0.00014
P	mg/L	--	--	6	0.02	0.005	0.0038	0.0028	0.0029	0.0029	0.0008	0.00062
Pb	mg/L	0.002	0.01	0.5	0.00036	0.000089	0.000068	0.000052	0.000054	0.000061	0.000018	0.000014
Sb	mg/L	--	--	--	0.00021	0.000051	0.00004	0.000031	0.000032	0.000037	0.000011	0.0000091
Se	mg/L	-	0.01	0.02	0.00073	0.00019	0.00016	0.00012	0.00013	0.000075	0.000022	0.000018
Si	mg/L	--	--	--	0.21	0.06	0.052	0.042	0.046	0.056	0.02	0.018
Sn	mg/L	--	--	2	0.00044	0.00011	0.000081	0.000061	0.000063	0.000068	0.000019	0.000015
Sr	mg/L	--	--	--	0.023	0.0057	0.0044	0.0034	0.0035	0.0028	0.00076	0.0006
Te	mg/L	--	--	--	0.00044	0.00011	0.000081	0.000061	0.000063	0.000068	0.000019	0.000015
Th	mg/L	--	--	--	0.000052	0.000014	0.000013	0.00001	0.000011	0.000016	0.0000059	0.0000052
Ti	mg/L	--	--	--	0.00077	0.0002	0.00016	0.00013	0.00014	0.00029	0.0001	0.000086

Predicted Pit Lake Chemistry	Water Quality Guidelines				Predicted Pit Lake Chemistry for Dibbwi East					Predicted Pit Lake Chemistry for Muntanga		
Tl	mg/L	--	--	0.5	0.00028	0.000069	0.000054	0.000041	0.000043	0.000047	0.000014	0.000011
U	mg/L	--	--	--	0.026	0.0071	0.0062	0.005	0.0055	0.0004	0.00013	0.00011
V	mg/L	--	--	1	0.0014	0.00039	0.00034	0.00027	0.0003	0.00042	0.00015	0.00013
W	mg/L	--	--	--	0.000067	0.000019	0.000016	0.000013	0.000015	0.000016	0.0000059	0.0000052
Zn	mg/L	0.02	3	10	0.072	0.017	0.013	0.0096	0.0099	0.0099	0.0026	0.002
Zr	mg/L	--	--	--	0.00044	0.00011	0.000081	0.000061	0.000063	0.000068	0.000019	0.000015

20.6.13. Conclusions

20.6.13.1. Geochemical characterisation

The geochemical characterisation study showed that waste rock from Muntanga, Dibbwi and Dibbwi East is predicted to have a low potential for acid generation. According to modified NP/AP some of the samples are classified as PAG however, according to NPTIC/AP results just one Dibbwi East sandstone sample is classified as PAG, a further three samples have uncertain acid generating potential and all other samples are classified as non-PAG. The non-PAG classification is supported by the NAG test results and low sulfide content of these materials. Paste pH was also typically greater than 5.5.

Solid phase uranium was elevated in the waste rock samples relative to average crustal abundance across all deposits with highest concentrations reported in samples from Gwabi, Muntanga and Dibbwi East. Other parameters reporting some enrichment in one of more of the deposits included boron, molybdenum, selenium and tungsten.

Spent ore materials were classified as uncertain to PAG and showed elevated uranium and molybdenum relative to average crustal abundance. Uranium, manganese and fluoride concentrations were elevated in the TCLP leachates from the spent ore.

The gypsum waste stream materials showed elevated calcium and uranium relative to average crustal abundance. SPLP and shake flask extractions undertaken on the gypsum materials showed elevated uranium and fluoride in the leachates.

20.7. Closure

20.7.1. Closure planning - Introduction

A conceptual rehabilitation and closure plan (“CRCP”) has been prepared to support the EIA for the project and to provide the cost estimate to be included in this feasibility study. The CRCP will be used as a basis for developing the detailed closure, aftercare and surrender documentation required during the life of mine. This section summarises the main elements of the CRCP.

The general objectives of the closure and rehabilitation of the mine site will be to ensure public safety and reclaim the land to a usable condition consistent with surrounding land use objectives, more detail on the post closure land uses selected for the various Project facilities can be found in the CRCP. The closure plan has been developed in line with regulations in place in Zambia and is cognisant of good international practice.

This section provides an overview of:

- Closure considerations: The closure plan has been developed in line with regulations in place in Zambia and is cognisant of good international practice
- Site decommissioning and closure actions
- Post closure monitoring
- Closure cost estimate
- Closure opportunities.

The CRCP is based on the project description collated by AMC and SRK based on the engineering design by the project engineers and environmental and social data collected as part of the environmental and social studies being undertaken for the project. The CRCP will be reviewed and updated with detailed closure planning documents as necessary during the life of the project.

20.7.2. Closure considerations

For the current CRCP the general objectives will be to:

- Maintain worker health and safety throughout closure activities, including concurrent closure;
- Protect public health and safety
- Reduce the visual impact of the facilities as far as practicable
- Demonstrate chemical stability compatible with site conditions
- Demonstrate physical stability compatible with site conditions
- Create a self-sustaining ecosystem compatible with site conditions
- Minimise need for reclamation maintenance
- Minimise negative impact on retrenched employees and local economy
- Maintain community relations and positive community impacts

- Reduce closure liability during operations through a concurrent closure program, this is currently an opportunity being explored by GoviEx.

20.7.3. Design criteria and closure assumptions

The Project design is currently at a feasibility level, consequently, a number of assumptions about the closure, rehabilitation and reclamation approach for the site have been made. As the project moves through construction and into operation, the assumptions in the CRCP will be better understood and future revisions of the CRCP can reflect the improved knowledge base. Future iterations of the closure plan and the detailed closure plan that is developed for the project will include adaptations that are required for the proposed closure actions based on updated data provided from climate change modelling.

The following general assumptions have been made in relation to closure planning:

- The mine life is around 12 years
- The duration of the active closure period is two years
- The duration of post-closure operating, monitoring and administration costs is currently estimated to be five years and will be dependent on the results of post closure monitoring
- No facilities will be handed over for use by the local populace at closure, it has been assumed that all mine related infrastructure will be removed, however this will be discussed with various stakeholders as the mine operations progress
- Wherever possible soil and cover materials will be salvaged or re-used within the project area
- Revegetation of areas will use a seed mix that is recommended by ecologists and regulators and will fit in with the local ecosystems
- Demolition and debris waste of an inert nature will be reused in rehabilitation activities (as appropriate), or recycled, recovered or disposed of (in descending order of preference). Non-hazardous and hazardous waste generated during decommissioning will be disposed of on site, including a dedicated Cat 3 waste facility
- There is no salvage value for any materials from the decommissioning of the infrastructure
- No water treatment will be required for the WRDs, spent ore facility or pit lakes, this assumption will be updated based on the results of predictive numerical geochemical modelling.

20.7.4. Closure actions

Based on the preliminary closure objectives, current industry standards, the current project description and the preliminary understanding of conditions at the site, the following initial concepts have been developed for the key project components. Table 20-13 is a summary of the actions that will be carried out for each of the project facilities, a full breakdown of each of the closure actions can be found in the Project CRCP.

Table 20-13: Summary table of main closure actions

Aspect	Closure Action Summary
WRDs	▪ WRDs will be permanent features ▪ WRDs to be constructed to not require significant recontouring ▪ Cover the facilities with 30 cm of growth media ▪ Revegetate top surface and slopes with regulator approved seed mix.
Open pits	▪ Open pits to be retained ▪ Safety berm to be retained/placed around perimeter with appropriate storm water drainage ▪ Additional action for Gwabi - construct a channel to connect the Kafue River to the Gwabi open pit, creating a flow through hydraulic system.
Process plant	▪ Structures will be dismantled and demolished or removed for re-use or sale ▪ Foundation will be broken up and buried to a depth of at least 500mm, if necessary, the area regraded ▪ Where required decontaminate the soils under the footprint ▪ Cover the facilities with 30 cm of growth media ▪ Revegetate top surface and slopes with regulator approved seed mix.
Spent ore facility	▪ The spent ore pad will be a permanent feature ▪ Minimal profiling where needed to establish safe slopes ▪ Routing of all run-off to existing perimeter areas ▪ Cover the facility with 1m of inert waste rock closure capping and 30 cm of growth media ▪ Revegetate top surface and slopes with regulator approved seed mix.
HLF	▪ The on-off leach pads will be rehabilitated ▪ Remove the water and when the ponds are empty remove the liners ▪ Where required decontaminate the soils underlying the pads ▪ Cover the footprint of the facility with 1 m of inert waste rock closure capping and 30 cm of growth media ▪ Revegetate top surface with regulator approved seed mix.
Offices, warehouses,	▪ For infrastructure that is not being handed over for an alternative post closure use ▪ Structures will be dismantled and demolished or removed for re-use or sale

Aspect	Closure Action Summary
workshops and fueling stations	▪ Footprints decontaminated where required ▪ Foundation will be broken up and buried to a depth of at least 500 mm, if necessary, the area re-graded ▪ Cover the facilities with 30 cm of growth media ▪ Revegetate top surface and slopes with regulator approved seed mix.
Roads	▪ Roads not required for post closure community use will be ripped and, if necessary, re-graded to local topography
Waste management	▪ Hazardous and non-hazardous landfills will be constructed to dispose of the waste generated during the active closure activities ▪ A dedicated Cat 3 landfill will be constructed during the operational phase of the mine, the small volume of Cat 3 waste generated during the active closure phase will be added to this landfill.
Closure studies	▪ An allowance has been made in the closure cost estimate to support the studies required for the development of the detailed closure plan that will be necessary prior to the operation entering the closure phase.
Post-closure Monitoring	▪ A five-year post closure monitoring period has been costed for. This will allow the Project to be confident that the facilities that remain in the post closure environment will be chemically and geotechnically stable.
Maintenance	▪ An allowance has been provided for in the closure cost estimate based on SRK's previous experience for similar projects in Zambia for the replacement/ repair of covers and revegetation during the post closure monitoring period.

20.7.4.1. Social closure

Further information will be gathered during future reviews of the CRCP to enable appropriate social management measures for the closure and post-closure phases of mine life to be defined. In the next version of the mine closure plan, particularly after the mine has been established and the socio-economic influences are realised more information should be provided with respect to proposed measures such as commitments to develop re-training plans and alternative livelihood opportunities post-closure. The post closure land use for the site will also be developed holistically to minimise the socio-economic impacts of the mine's closure.

20.7.5. Post-closure monitoring and maintenance

Several aspects of the closure plan will need to be monitored on an ongoing basis and will require care and maintenance, these will include:

- Groundwater monitoring piezometers (installed in addition to the operational network if deemed required) - Groundwater quality sampling
- Surface water quality sampling associated with agreed discharge consent criteria and locations
- Local wells
- Annual visual inspection of the WRDs, spent ore pad, berms and other facilities as required
- Revegetation care and maintenance.

In addition to the above activities one-off assessments may be required before the site can be relinquished and finally closed, for example biodiversity surveys or social impact assessments depending on the agreed post closure land use objectives.

Water quality monitoring will be required after closure to ensure the effectiveness of the closure approach. At a minimum, the post closure monitoring period is expected to last five years, with the frequency of monitoring decreasing over this period as follows:

- Monthly monitoring for the first 12 months
- Quarterly monitoring for the next 48 months
- Additional annual monitoring if required to ensure closure objectives have been met.

Post closure monitoring would include the following as a minimum on the same frequency as the water quality monitoring:

- Visual inspections of rehabilitated areas to ensure the site remains stable and free of significant erosion
- Monitoring the performance of re-planted vegetation and where necessary replanting, re-contouring, and altering the seed mix as required.

20.7.6. Closure cost estimate

The closure cost estimate has been developed on a first principles basis, using equipment, material and labour unit rates where available from the project engineers, and where not available (for example seed mix rates) by using a cost data file appropriate for rehabilitation works in Zambia. The footprint areas and slope angles for each of the facilities were also used along with the unit rates to develop the life of mine closure cost estimate. The cost estimate was generated using the standardised reclamation cost estimator ("SRCE") software which is globally recognized for its use in the development of closure costs. The SRCE model combines the physical parameters for site facilities and then uses productivity data for mine fleets to develop the time required for rehabilitation activities and then uses the Project unit rates to convert the time into a cost.

In accordance with the assumptions listed above, the total direct closure costs for the project are estimated at USD23 million. With add-ons such as engineering, design, and construction plans, the total is approximately USD30 million.

A summary of this cost is provided in Table 20-14 and a full breakdown of the closure cost is contained in the CRCP.

Table 20-14: Summary of LOM closure costs

Aspect	Cost [USD M]
WRDs	4.5
HLPs	0.5
Spent ore facility	2.5
Open pits	2.0
Gwabi water management	1.0
Roads	0.5
Disturbed areas	2.0
Plant demolition	3.0
Infrastructure demolition	0.5
Waste management – inc Cat 3 disposal and gypsum pond cover	1.5
Closure studies	1.0
Monitoring	2.0
Maintenance	3.0
Total	**24.0**
Contingency (25%)	6.0
Grand total	**30.0**

20.7.7.Opportunities

As it currently stands, it is assumed all the infrastructure onsite will be demolished and reused/ recycled/ recovered/ disposed of offsite, and no scrappage value recovery or reuse has been assumed in the costings. Some of the infrastructure constructed for the project could continue to provide benefits for the community or local authorities if it was left in place (repurposed) once the mining operations have ceased. In addition to providing value to local communities or facilitating future industrial development in the region, the transfer of certain facilities would reduce the overall cost of decommissioning for the company and may reduce the residual liability of the project post closure. As noted above the transformer and power lines will be handed transferred to ZESCO on construction and will hence remain in place post closure.

Although no formal discussions have been held with stakeholders regarding transfer of these facilities, GoviEx will continue to conduct stakeholder negotiations to identify infrastructure with potential for beneficial future use and that could be transferred to the authorities or other third parties.

SRK notes that until agreements are in place with the relevant regulators and local stakeholders, the above items should be regarded as an opportunity only and the closure costs will only be modified in future versions of the CRCP once the agreements have been put in place.

The current cost estimate and closure planning has all the closure actions scheduled at the end of mine life. There is an opportunity to reduce the closure cost estimate, reduce the mine's operational footprint and determine the efficacy of the proposed closure actions during mine life by conducting concurrent rehabilitation activities.

The intention is for the CRCP to be reviewed and updated every three years as a minimum during the life of the mine.

20.8. Carbon footprint

20.8.1. Introduction

As part of the FS, a greenhouse gas ("GHG)" accounting assessment was undertaken to understand the potential emissions from the project. This GHG accounting assessment provides a detailed evaluation of emissions across relevant scopes, adhering to established standards such as the Greenhouse Gas Protocol. Quantifying emissions for the Project supports transparency, guides sustainability initiatives, and ensures compliance with regulatory and voluntary reporting frameworks.

The primary objectives of this assessment are to:

- Identify and quantify GHG emissions across key activities.
- Categorise emissions into scopes (Scope 1: direct emissions and Scope 2: indirect energy).

The emissions assessment has been compared against the Equator Principles (EP4) and International Finance Corporations Performance Standards (PS3). The following should be noted with each guidance document:

- EP4 – Under EP4, if the project has been classified as a 'Category A' project and the projects emissions are expected to exceed 100 000 tCO_2e, a GHG emissions assessment, Physical and Transition Climate Change Risk Assessment (CCRA), and a GHG Alternative Analysis are required. If the project emissions are expected to be less than 100 000 tCO_2e then only a GHG Emissions Assessment and Physical CCRA are required. A Transition CCRA can be undertaken if the Equator Principles Finance Institutions require it.
- IFCPS3 – Consideration should be given for alternatives. The implementation of technically and financially feasible and cost-effective options to reduce project related GHG emissions during the design and operation of the project should be considered. As stated in the IFC Performance Standards, '*These options may include, but are not limited to, alternative project locations, adoption of renewable or low carbon energy sources, sustainable agricultural, forestry and livestock management practices, the reduction of fugitive emissions and the reduction of gas flaring*', as applicable to a project. For a project expected to produce more than 25 000 tCO_2e annually, the client will quantify direct emissions from the facilities owned or controlled within the physical project boundary, as well as indirect emissions associated with the off-site production of energy used by the project. Quantification of GHG emissions will be conducted by the client annually in accordance with internationally recognized methodologies and good practice.

20.8.2. GHG Methodology

Table 20-15 provides an overview of the methodology used for the GHG accounting for the Muntanga project.

Table 20-15: GHG Methodology

Item	Description
Methodology	GHG Protocol[3]
Organisational boundary	Muntanga Site
Operational boundary	Scope 1 and Scope 2 emissions arising from site activities.
Emission sources assessed	Scope 1 - Mobile and stationary combustion Scope 2 - Purchased electricity
Activity data	Scope 1 – Diesel (litres) Scope 2 – Electricity consumption in kilowatt hours (kWh)
Life of Mine (LOM)	12 years
GHGs identified	Carbon dioxide (CO_2), methane (CH_4), nitrous oxide (N_2O)
Emission factor reference	Diesel - Sourced from the Department for Environmental, Food and rural Affairs[4] Electricity consumption – Carbonfootprint.com[5]
Emission factors for diesel	CO_2e – 2.66155 $kgCO_2e$/litre CO_2 – 2.62818 $kgCO_2e$/litre CH_4 – 0.00029 $kgCO_2e$/litre N_2O – 0.03290 $kgCO_2e$/litre
Emission factor for electricity	0.109 kgCO2e/kWh

[3] GHG Protocol Corporate Standard - https://ghgprotocol.org/corporate-standard
[4] DEFRA GHG Conversion Factors - https://www.gov.uk/government/publications/greenhouse-gas-reporting-conversion-factors-2024
[5] https://www.carbonfootprint.com/docs/2024_07_international_electricity_factors_1.xlsx

Item	Description
Global warming potential	AR5[6] as it is built into the emissions factors used from DEFRA: CO_2 – 1 CH_4 – 28 N_2O - 265
Equation	Emission=Activity Data × EF
Comparison Criteria	Equator Principles 4 – Guidance Note on Climate Change Risk Assessment (CCRA) IFC PS3 – Resource efficiency and pollution prevention

The emissions for land use change have been omitted at this stage as it is unclear the size of area to be cleared for construction and operational activities at this stage. However, these should be included in the inventory in each year that land is cleared during construction and operation phases. A study completed by African Mining Consultants Limited (AMC) on carbon storage stocks in the mining area should be referenced when including emissions from land use change. The report provides emission factors that can be used for this assessment.

20.8.3.GHG Results

The total GHG emissions per source for the Muntanga project is presented in Table 20-16 and Table 20-17. Across the LOM, the emissions will total 363 110 tCO_2e of which 81 % is from mobile combustion and 19 % from purchased electricity. The annual average has been estimated around 30 259 tCO_2e. The annual GHG emissions are presented in Table 20-17. Apart from the latter two years in the LOM, the annual emissions are comparable year on year.

Emissions from purchased electricity are the same year on year, except in the first and last years due to the ramp up and down in operations in those years respectively. It should be noted that as the Zambian grid incorporates more renewable energy options into the grid, the emissions from the use of purchased electricity at the mine will decrease accordingly.

On an annual basis the project is lower than the Equator Principles threshold of 100 000 tCO_2e per annum which would mean that threshold requirements for the project are a Physical Climate Change Risk Assessment and annual GHG emissions assessments.

Table 20-16: Summary of GHG Results for LOM

Scope	Source	GHG Emissions (tCO_2e)	% of total emissions
1	Mobile combustion	293 925	81
2	Purchased Electricity	69 185	19
Total emissions		**363 110**	**100**

Table 20-17: Annual GHG Emission

Year	Activity Data		Emissions (tCO_2e)		Annual Emissions (tCO_2e)	% of total emissions
	Fuel Usage Mobile Combustion [litres]	Electricity used [kWh]	Scope 1 Mobile combustion	Scope 2 Purchased electricity		
1	9 226 268	41 391 000	24 555	4 152	29 067	8.0
2	10 986 210	55 188 000	29 238	6 016	35 254	9.7
3	10 939 827	55 188 000	29 115	6 016	35 131	9.7
4	10 048 491	55 188 000	26 743	6 016	32 759	9.0
5	9 763 211	55 188 000	25 984	6 016	32 000	8.8
6	9 578 275	55 188 000	25 491	6 016	31 507	8.7
7	10 231 503	55 188 000	27 230	6 016	33 246	9.2

[6] AR5 GWPs are used for this assessment, however it should be noted that the GHG Protocol have accepted the use of AR6.

Year	Activity Data		Emissions (tCO_2e)		Annual Emissions (tCO_2e)	% of total emissions
	Fuel Usage Mobile Combustion [litres]	Electricity used [kWh]	Scope 1 Mobile combustion	Scope 2 Purchased electricity		
8	9 716 849	55 188 000	25 860	6 016	31 876	8.8
9	9 116 639	55 188 000	24 263	6 016	30 279	8.3
10	9 667 005	55 188 000	25 727	6 016	31 744	8.7
11	7 091 309	55 188 000	18 873	6 016	24 889	6.9
12	4 075 500	41 391 000	10 846	4 152	15 358	4.2
Average	**9 203 424**	**55 188 000**	**24 494**	5 765	**30 259**	**8.3**
Total	**110 441 086**	**634 662 000**	**293 925**	**69 185**	**363 110**	**100**

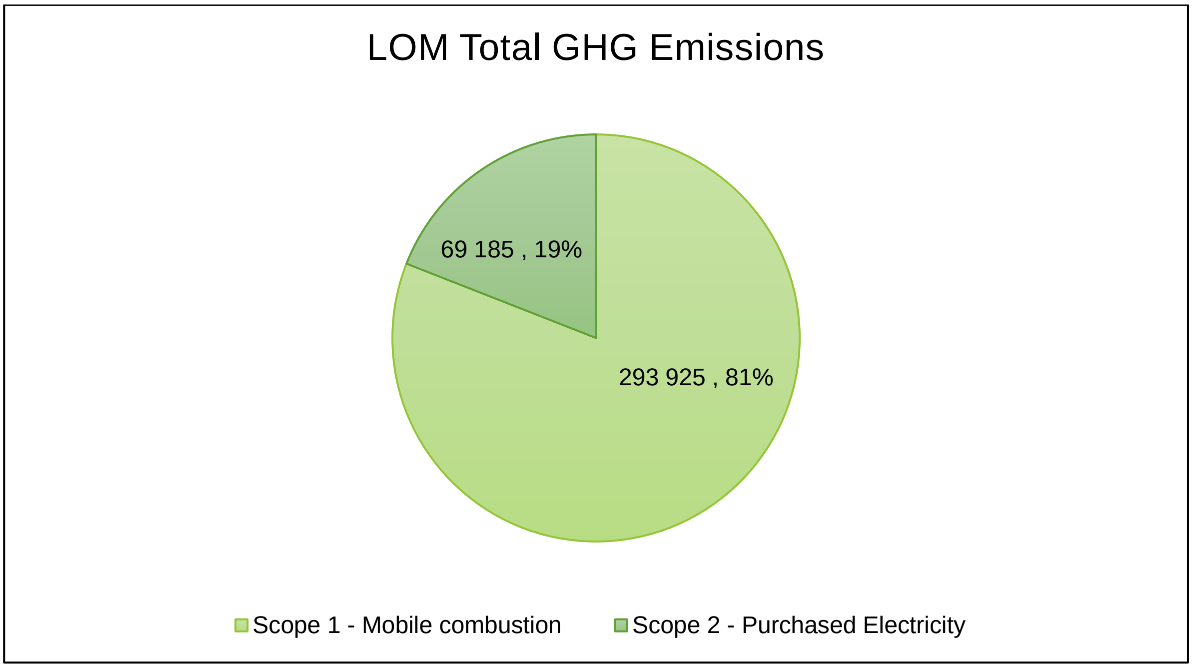


Figure 20-37: Total emissions split by source

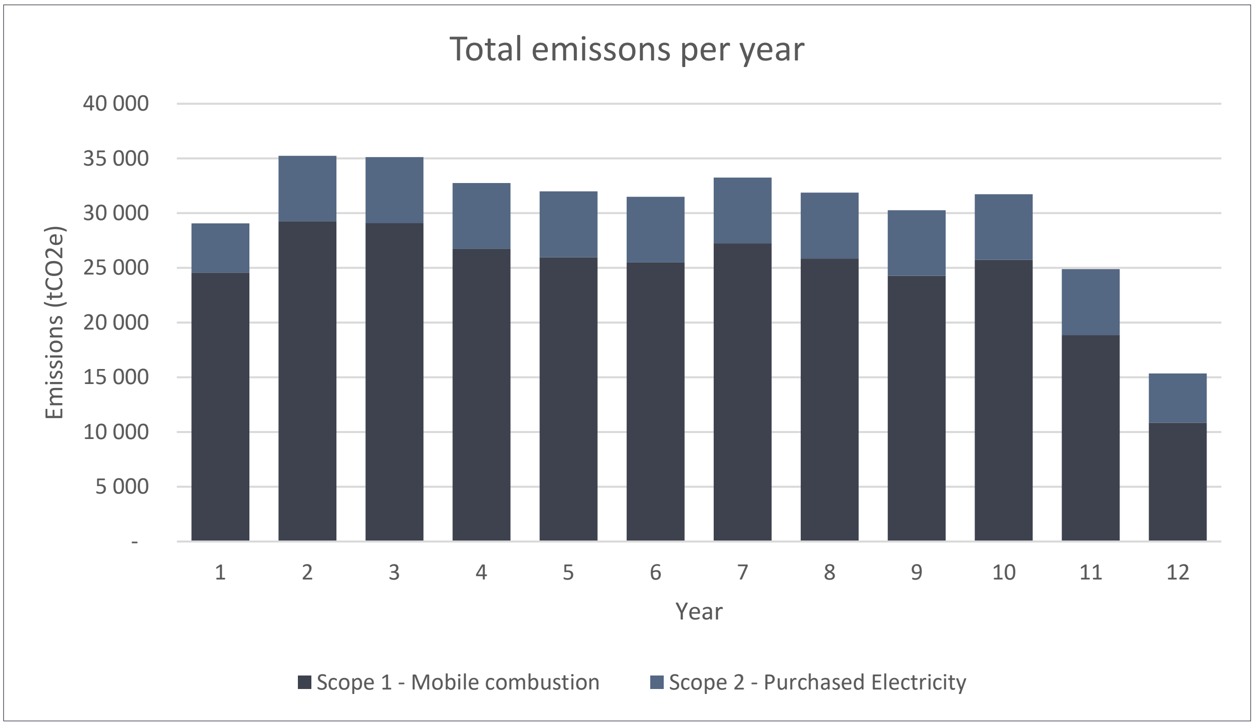


Figure 20-38: Emissions split by source over LOM

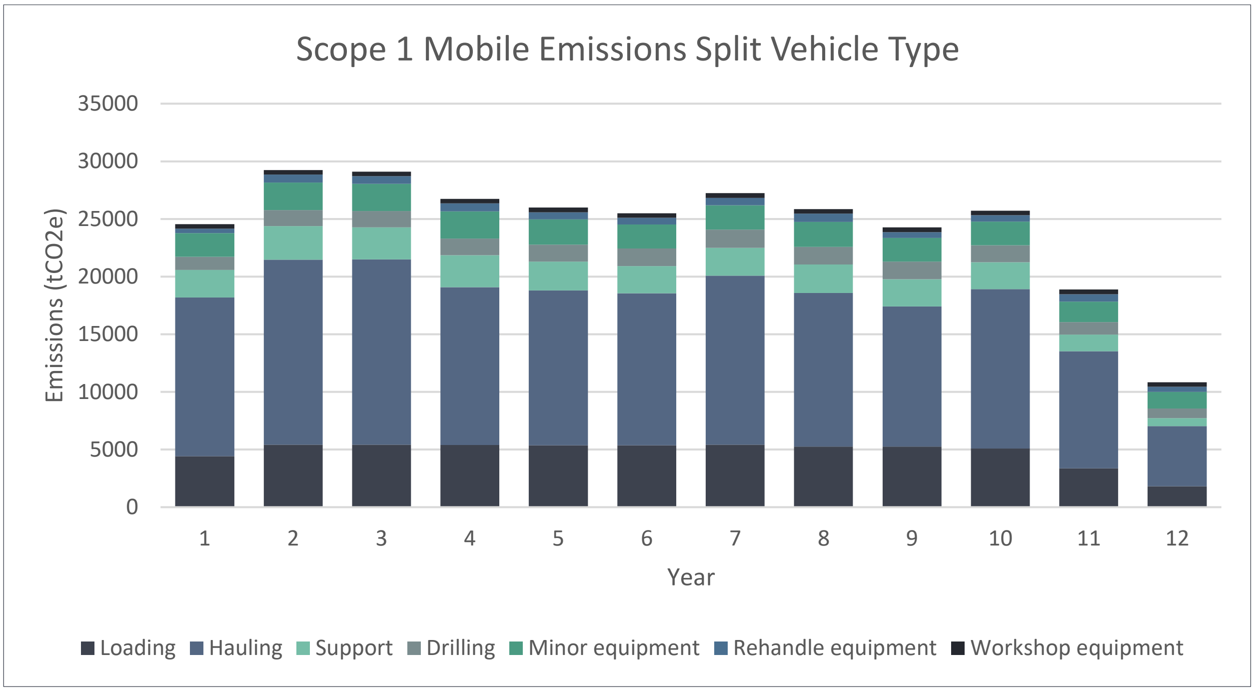


Figure 20-39: Scope 1 mobile emissions split by vehicle type

20.8.4.Benchmarking

Table 20-18 provides benchmarking data of five uranium companies: Paladin Energy, Kazatomprom, Cameco, Bannerman Energy, and Orano that will be compared to GoviEx. The analysis focuses on the emissions data, production output, and GHG intensity per pound of U_3O_8. The GoviEx emissions intensity is based on annual average data over the LOM. GoviEx's emission intensity is 0.01 tCO_2e/lb U_3O_8. The Muntanga project has lowest emissions intensity when compared to peers selected for the benchmarking.

Table 20-18: Annual GHG Emission

Company	Scope 1 [tCO_2e]	Scope 2 [tCO_2e]	Total emissions [tCO_2e]	Production [lbs.]	GHG Intensity [$tCO_2e/lb\ U_3O_8$]
GoviEx	24 494	5 765	30 259	2 200 000	0.01
Paladin Energy[7]	18 994	19 063	38 057	517 597	0.07
Kazatomprom[8]	120 618	596 634	717 252	46 296 451	0.02
Cameco[9]	128 673	181 397	310 070	17 638 201	0.02
Bannerman[10] Energy	50 975	25 640	76 615	2 614 846	0.03
Orano[11,12]	175 687	162 977	338 664	15 754 215	0.02

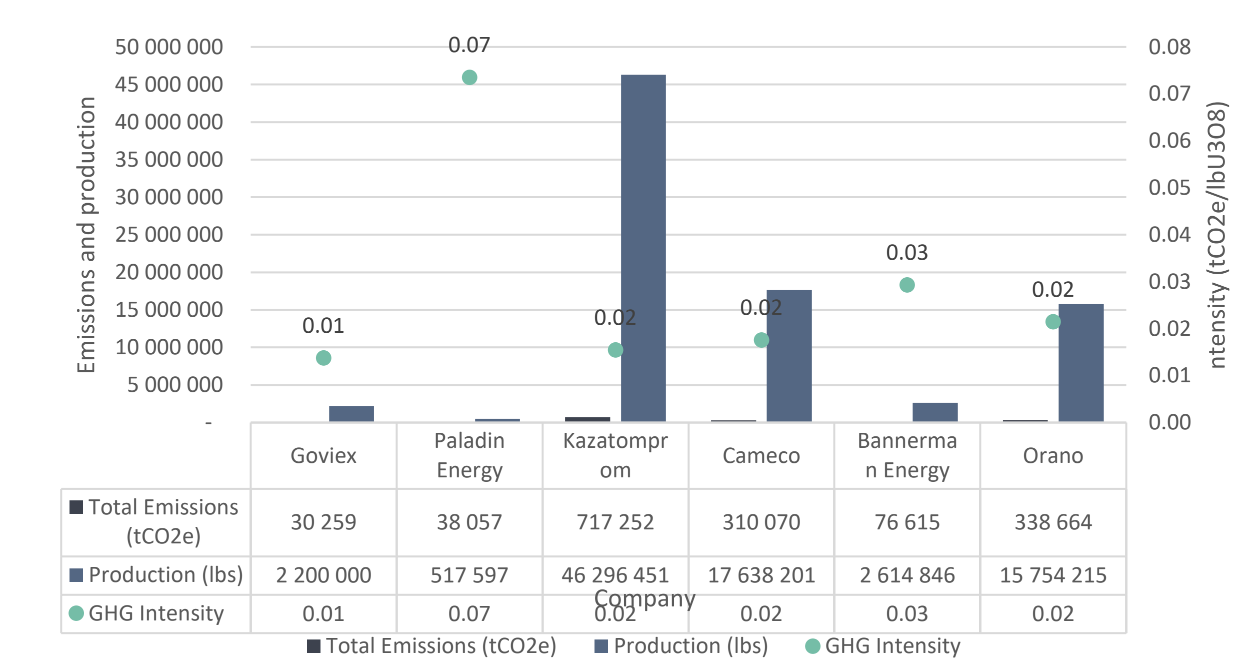


Figure 20-40: Benchmarking Muntanga emissions against peers

20.8.5.Zambia Commitments

Zambia's NDC was updated on the July 30, 2021, for the period 2015 to 2030. Zambia plans to reduce its GHG emissions by 25 % (at business-as-usual levels of international support prevailing in 2015) by 2030 against the 2010 base year. Alternatively, Zambia with substantial international support, plans to reduce emissions by 47 % by 2030. Zambia will focus on three programmes to mitigate emissions:

- Sustainable forest management
- Sustainable agriculture
- Renewable energy and energy efficiency

Table 20-19 presents a scenario of what Zambia's cumulative GHG emissions would be once the Muntanga project is operational. Assuming Zambia's emissions are similar to the country's 2023 emissions throughout the LOM, the Muntanga project would only contribute ±0.1 % to Zambia's total emissions.

[7] https://www.paladinenergy.com.au/wp-content/uploads/2024/10/2024.10.28-2024-Sustainability-Report_Web_Spreads.pdf
[8] https://www.kazatomprom.kz/storage/00/kazatomprom_iar_2023_eng.pdf
[9] https://www.cameco.com/sites/default/files/documents/Cameco-2023-Sustainability-Report.pdf
[10] https://bannermanenergy.com/sustainability/
[11] https://cdn.orano.group/orano/docs/default-source/orano-doc/groupe/publications-reference/publication-groupe/orano_annual-activity-report_2023.pdf
[12] https://cdn.orano.group/orano/docs/default-source/orano-doc/expertises/producteur-uranium/rapport-de-responsabilité-sociétal/orano_mining_rse_2023_en_v6_ld.pdf

Table 20-19: GoviEx's emissions contribution to Zambia's emissions

Scenario	Average annual GoviEx Emissions	Zambia's 2023 Emissions	Cumulative Emissions	Potential increase
tCO_2e	**30 510**	30 484 449	30 514 708	±0.1 %

20.8.6.Conclusions

The major emitting source is mobile combustion, which contributes 81 % of the total emissions, with purchased electricity contributing 19 % . Emissions have been calculated to be 363 110 tCO_2e for the LOM. On an annual basis the project is lower than the Equator Principles 2020 (EP4) threshold of 100 000 tCO_2e per annum, which would mean the threshold requirements for the project are a Physical Climate Change Risk Assessment and annual GHG emissions assessments. The emissions from the mine will increase Zambia's annual emissions by ±0.1 % should Zambia's emissions remain around 30 million tCO_2e.

20.8.7.Opportunities to reduce GHG emissions

Table 20-20 shows opportunities that were identified to reduce the Project's GHG emissions. However, the Project equipment configuration remains diesel equipment, because there is not a base of similar equipment and support in Zambia, project life is relatively short, electrical supply is constrained, capital costs are higher and/or technology is new or unproven. These options may be considered later in the project life when equipment is due for replacement, and if further resources are developed to warrant expenditure.

Table 20-20: Opportunities to reduce GHG emissions

Reduction Opportunity	Practicality	Reasons for option consideration or rejection
Open Pit: Reduce diesel consumption by installing trolley-assist or IPCC	Yes – proven technology but needs sufficient mine life and a suitable pit development sequence to deliver results	Potential to reduce scope 1 emissions by up to 60 %
Open Pit: Switching haul-trucks from diesel to battery, hydrogen fuel cells.	New technology, likely only from 2027	Potential to reduce scope 1 emissions by up to 80 %
Open Pit: Automated Haulage Systems	Moderate as needs reliable technology platforms (LIDAR, etc.)	Potential to reduce scope 1 emissions due to haulage by ~10-15 %
Open Pit: Electric Drill Rigs	Yes – electric drills are proven technology	Potential to reduce scope 1 emissions from drilling equipment by up to 100 % if 100 % electricity from renewable sources
Biofuels	No	Not practical in terms of location
Hydrogen Fuel Cell Vehicles	Not at this time	Still in trial at this stage

20.8.8.Recommendations

Based on the assessment undertaken, the following are recommended:

- Emissions estimation should be undertaken during the construction, operational and closure phases of the mine to confirm what the actual GHG emission are from operating activities. The following emissions sources should be included in the inventory during the abovementioned mine phases: wastewater treatment, landfill, aircon and refrigeration, usage of oil and lubricants, and land use change. Based on the outcome of the inventory a materially assessment can be undertaken to exclude emissions that are less than 5 % of the total emissions from the mine
- As the project is assessed to produce less than 100,000 tCO_2e/year, therefore under EP4, the following need to be undertaken:
  - Once off Physical CCRA
  - GHG assessment on an annual basis throughout the LOM
- Develop a GHG management plan to manage GHG emissions
- As the project is over a 12-year timeframe, the following should be considered to manage GHG emissions:
  - Optimise equipment use to lower fuel and electricity consumption
  - Consider energy efficient lighting and equipment
  - Optimise transportation emissions by reducing number of trips and fuel during materials handling activities, where practical.

20.9. Climate change and project design

As described in Section 18.3 of the report, the impact of climate change on floodlines was assessed for the Northern Njame Mine catchment which served as a proxy of the entire region. The 1:50-year and 1:100-year flood lines were revised based on the existing HEC-RAS model and the estimated peak flow rates by incorporating the projected impacts of climate change. The highest increase in rainfall for the near-term scenario was chosen as input into the model, in order to represent the worst-case scenario of flooding. The highest increase projected for the near-term is +3.9 % for the one-day maximum rainfall. The results of the climate change impact assessment indicated a 3.9 % increase in the one-day maximum rainfall resulted in a difference of <10 mm flood level, which is an insignificant change in the peak flow rates.

The design of the WRDs and spend ore dump have considered the longer-term impacts of climate change and the closure design criteria have factored in a potential increase in a one-day rainfall event.

20.10. Risks to the Project

The following risk factors have the potential to impact project delivery. These are being assessed and addressed as part of the environmental impact assessment process and mitigation and management measures integrated into the FS.

The concept of double materiality is applied, with potential ESG impacts from the project considered equally to impacts posed by the ESG setting to the project. The following section summarises potential material risks.

Material issues are assumed to be factors that could:

- Stop the project, affect the continuation of operations or obtaining of approvals
- Pose major concern to stakeholders and/or could affect the social licence to operate
- Are out of alignment with corporate strategies or policies and/or
- Result in the need for additional studies or costs that could affect the proposed design and/or operation of the Project and thus the value of the assets (e.g., design changes, operational management requirements, cash flow restrictions, rehabilitation/closure demands).

The potential for materiality has been identified on the basis of:

- Experience of the ESG staff and supporting consultants
- Understanding of the location; proposed operation; regulatory and governance structure; socio-political situation; environmental and social setting and
- Understanding GoviEx and the FS report audience, in particular current expectations from investors around ESG factors and the requirements of international standards for a uranium project.

20.10.1. Resettlement

As described in Section 20.3.2, the project requires the physical and economic resettlement of several small villages. Resettlement can have a long lead time and can delay project development. AMC has completed the baseline data collection and is in the process of defining and communicating a compensation framework. This will be used to negotiate final settlement agreements with affected parties.

Land for resettlement has been identified adjacent to the project areas. Land acquisition is still to be finalised and negotiations could be protracted. GoviEx is aware of the potential difficulties associated with resettlement and AMC have experience of other RAP exercises in Zambia. GoviEx anticipate the RAP approval process will take approximately 12 months followed by the implementation phase. GoviEx anticipate being able to progress key construction activities in parallel with the resettlement programme, but pre-stripping and mining can only commence on completion of the resettlement programme.

An early works programme (including access road construction) will be implemented in parallel with the detailed approvals process to enable construction of resettlement villages to start as soon as the RAP is approved.

20.10.2. Permitting schedule

The project is conducting a full ESIA process on the basis of the updated design. To complete this to IFC standards and obtain the required approvals from the Zambian government will take time and consideration in terms of the overall project schedule. Additional permits for management of waste are likely to be required prior to construction.

Although consultation and review timeframes are stipulated for approval of ESIAs and other permits, there is a threat that the approval process could be protracted. The risk also exists of regulators including unrealistic conditions of approval in the permit resulting in cost implications for the project and/or the need to renegotiate permit conditions within a tight project timeframe. GoviEx engage regularly with the various regulatory bodies and this on-going engagement should mitigate risks of delays associated with ZEMA and other government ministries.

20.10.3. In-migration

A development of this significance in a relatively remote area is likely to attract job seekers who may put pressure on accommodation and other resources in the local area. This will be addressed as part of the ESIA and will require coordination and recruitment procedures by GoviEx and its various contractors to avoid unintended secondary social and environmental impacts.

Direct and indirect employment opportunities will vary across the life of the mine in terms of numbers and skills required. GoviEx has a policy to employ 100 % Zambians where practicable and is committed to sourcing labour as close to the Project as possible. It is estimated that around 800 skilled and semi-skilled jobs will be created during the life of operations with substantially more temporary positions during construction.

Playing a proactive role in this through training suppliers to enhance the quality of their service and products could also result in skill development that is transferable to other industries. If this were linked into existing technical and vocational education and training (“TVET”) initiatives, it would begin the process of ensuring a positive legacy and sustainable benefits. This could impact positively on a large proportion of the disengaged young people in the rural communities through increasing their access to direct and indirect employment opportunities.

The company supported adult litreacy and numeracy education, sponsoring the Back to School Project, an initiative run in partnership with the District Education Board Secretaries (“DEBS”) for the Siavonga and Chirundu Districts. It will focus on providing educational opportunities for adults who may not have had previous access to formal education.

Six students have already received sponsorship for vocational training at the Lusaka Vocational Training College (“LVTC”). GoviEx is seeking to develop a vocational training facility with LVTC closer to the Project.

20.10.4. Water resources

The project will require access to water in a setting where surface water is seasonally scarce. Both surface water quantity and quality will need to be managed carefully taking into consideration other water users in the catchment. Climate change adaptation has been considered in mine design (e.g. considerations for increased seasonal temperatures, reduced water availability and more intense flood events). Despite this, increased storm intensity and potential for more severe droughts could impact water management infrastructure.

Obtaining permits to abstract and discharge water will be key along with water treatment systems to achieve discharge standards. The discharge of groundwater to surface water as part of the mine dewatering may prove challenging. Any water discharged will need to meet the appropriate Zambian discharge standards and the water quality will need to be considered in relation to downstream receptors. The groundwater quality has a different chemical composition to the surface water baseline. This is being addressed as part of the ESIA.

20.10.5. Biodiversity

In line with global standards, major projects will need to demonstrate no net loss (“NNL”) to biodiversity, or a net gain in areas considered critical habitat under IFC PS6[13]. As noted in the fauna and flora baseline, there are species of conservation concern with the wider project area. The area is also known to provide ecosystem services in the form of surface water, groundwater, firewood, savannah fruits and medicinal plants. In addition, local communities generate revenue from the sale of locally produced charcoal. The on-going ESIA process will confirm whether the company needs to achieve NNL or a net gain for biodiversity. Implementing successful biodiversity and ecosystem services initiatives require coordination and buy-in from both local communities and regulators, as well as requiring both financial and technical resources.

The environmental baseline has identified an elephant migration route that passes between the Dibbwi and Muntanga sites. This could lead to more interactions with elephants at particular times of year and will require specific management measures, particularly around traffic management. While elephants tend to avoid areas or human habitation, ongoing habitat loss means more human/ wildlife encounters are inevitable.

[13] International Finance Corporation: Performance Standard 6 (IFC PS6) on Biodiversity Conservation represents international best practice for biodiversity management

21. Capital and operating costs

21.1. Common cost factors

Several cost drivers are common to all operating areas across the mine. These include labour costs, electricity costs and diesel costs.

21.1.1. Labour costs

Labour costs are based on the report by Align Advisors: "Benchmark Salary Report, Zambian Mining Industry 2024". This report defined a full basis for remuneration based on a review of other mining operations in Zambia which provided their equivalent costs. Table 21-1 shows the monthly and annual remuneration for all levels of employment on the Project. The total cost to company ("CTC") includes the base salary, housing allowance, other allowances, income taxes, allowances for PPE, periodic medicals and airtime.

Table 21-1: Remuneration for each job level

Level	Job category	Total CTC [USD/month]	Total CTC [USD/year]
13	GM I	17 894	214 726
12	Senior Manager	14 549	174 583
11	Manager II	11 370	136 436
10	Manager I	8 149	97 791
9	Superintendent II	5 626	67 508
8	Superintendent I	3 325	39 897
7	Senior Professional / Supervisor High	1 903	22 833
6	Full Professional/ Supervisor II	1 424	17 093
5	Supervisor/ Full Professional	1 095	13 138
4	Professional, Senior Clerical Senior Skilled Worker, Team Leader	854	10 251
3	Clerical II Senior Operator/ Senior Technician	670	8 038
2	Clerical I Operator II/ Technician I	567	6 798
1	Operator I / Technician Assistant	494	5 924
0	Unskilled / Labourers	403	4 830

These costs are applied to the labour counts derived for each area of the operation to calculate labour costs.

21.1.2. Electricity cost

Mining companies negotiate a "Special Negotiated Tariff" with ZESCO through a PPA. The tariff will be confirmed once the connection agreement (see Section 18.4.1) is agreed upon and during PPA negotiations. The PPA is anticipated to ensure preferential access to power supply at all times. During the FS, GoviEx has significantly furthered the drafting of the CA and is awaiting a draft PPA document to be supplied by ZESCO for their review. For the FS, an all-in tariff (i.e., fixed and variable components included) of 12.5 USD c/ kWh is used, which is based on recent confidential reference points from Utilink and SRK.

The current draft of the CA calls for ZESCO to contribute 30 % of the total capital cost of the power supply infrastructure. The mode of payment is not yet confirmed but would likely be via a tariff adjustment or rebate and is modelled as such in the economic analysis.

21.1.3. Diesel cost

The diesel price used as the basis for the study is USD0.97 /L of fuel (excluding VAT). The price is based on supply from Dar es Salaam including transportation, port charges, in transit losses, import duties, levies, statutory margins, profit and transportation to site. The price was provided under the condition that fuel is supplied as a service from the selected fuel supplier.

21.2. Mining operations

The mining Capex and Opex costs are driven by the equipment requirements, equipment estimated useful life, labour requirements, usage of consumables (e.g. explosives and diesel) and unit costs described in Section 16.

21.2.1. Mining equipment capital costs

Total mining equipment Capex is USD126.7 million over the LOM, comprising initial capital of USD36.9 million and sustaining capital of USD89.8 million. The mining equipment Capex over the LOM is shown in Table 21-2.

Table 21-2: Annual mining equipment capital cost provisions

Mining Capital[1]	Initial	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Initial Capex	36.9												
Sustaining Capex		35.9	1.19	0.29	1.9	0.3	0.72	37.9	10.8	2.3	0.02	-	-

Includes mine spec add-ons, 15 % import duty, fleet management and extended warranty provisions

Initial capital cost includes all equipment purchased to achieve steady state mining reflected in Table 21-3 while sustaining capital relates to end of useful life equipment replacements as discussed under item 16.2.10 above.

Table 21-3: Equipment capital breakdown

Equipment	Capex [USD million]			
	Initial	Expansion	Sustaining	LOM total
Loading equipment	3.5	5.9	9.4	**18.8**
Hauling equipment	15.2	23.9	35.9	**75**
Support equipment	5.4	4.3	0.7	**10.4**
Drill equipment	2.4	1.2	1.2	**4.8**
Minor equipment	5.3	1.8	5.2	**12.3**
Rehandle equipment		2.6	0	**2.6**
Workshop equipment	1.7	0	1	**2.7**
Total	**33.5**	**39.8**	**53.4**	**126.7**

As expected, excavators and trucks are the largest capital cost contributors. But as reflected in Table 21-4, significant life is left in the load and haul replacement equipment at the end of LOM, indicating opportunities to effect cost per tonne capital savings for consideration in possible future mine expansion plans.

Table 21-4: Capex utilisation as a percentage of estimated useful life

Equipment category	Initial equipment	Sustaining equipment
Loading	100 %	63 %
Hauling	100 %	60 %

21.2.2. Mining equipment operating costs

Operating costs are summarised under the following headings: equipment maintenance costs, fuel costs, blasting costs, labour costs and fixed costs.

Equipment maintenance and fuel account for the majority with a 63 % cost contribution as shown in Figure 21-1.

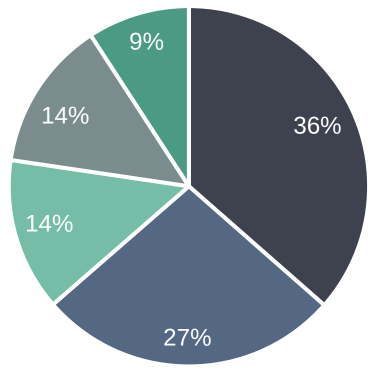


Figure 21-1: Operating costs contributors

The total mining operating cost as tabled in Table 21-5 amounts to USD379 million. The cost per tonne mined (total ore and waste) averages USD2.06 /t over the LOM.

Table 21-5: Annual mining operating cost provisions

Operating costs	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	End
Equipment maintenance	11.26	13.44	13.39	12.63	12.3	12.11	12.74	12.22	11.66	12.08	8.86	5.36	
Fuel	8.58	10.22	10.17	9.35	9.08	8.91	9.52	9.04	8.48	8.99	6.59	3.79	
Blasting	3.66	4.49	4.53	4.6	4.59	4.9	5.1	4.95	4.8	4.52	3.33	2.49	
Labour	4.09	4.51	4.51	4.55	4.53	4.47	4.47	4.39	4.37	4.38	3.75	3.05	
Fixed	6.12	2.55	2.51	2.53	2.53	2.49	2.5	2.54	2.47	2.51	2.88	2.68	0.43
Total	**33.71**	**35.2**	**35.12**	**33.65**	**33.03**	**32.87**	**34.34**	**33.14**	**31.77**	**32.47**	**25.41**	**17.37**	**0.43**
USD/t mined	*2.39*	*2.03*	*2.03*	*1.95*	*1.93*	*1.92*	*1.98*	*1.97*	*1.89*	*1.99*	*2.39*	*3.08*	

21.2.2.1. Equipment maintenance costs

Equipment maintenance costs amount to a total of USD138 million and include equipment insurance, replacement parts, servicing (filters and lubricants), wear checks, bucket/ load body repairs, tyres/ under carriage repairs and replacement, and GET costs as reflected in Table 21-6.

These costs correlate with equipment utilisation expressed machine hours calculated from the production schedule, productivity assumptions and haulage simulations as discussed in Section 16.2.10 above.

Table 21-6: Annual equipment maintenance cost provisions

Operating costs	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Total
Insurance	0.46	0.55	0.55	0.55	0.55	0.55	0.55	0.53	0.52	0.52	0.40	0.25	6.00
Parts	6.50	7.71	7.65	7.22	6.99	6.83	7.18	6.93	6.59	6.84	5.05	3.10	78.60
Filters	0.60	0.72	0.72	0.65	0.63	0.62	0.67	0.63	0.59	0.63	0.46	0.26	7.17
Wear check	0.06	0.07	0.07	0.06	0.06	0.06	0.06	0.06	0.05	0.06	0.04	0.02	0.66
Lubes Cost	0.39	0.47	0.46	0.42	0.41	0.40	0.44	0.41	0.38	0.41	0.30	0.17	4.68
Buckets & Bodies	0.26	0.31	0.31	0.28	0.28	0.27	0.29	0.27	0.26	0.27	0.19	0.10	3.11
Tyres & UC	1.84	2.19	2.19	1.99	1.94	1.91	2.05	1.93	1.81	1.93	1.39	0.76	21.92
GET	1.15	1.43	1.44	1.45	1.44	1.47	1.51	1.47	1.45	1.41	1.02	0.69	15.92

When viewed by activity, excavators and trucks used for loading and hauling are the single largest cost contributors, shown in the Pareto analysis in Figure 21-2.

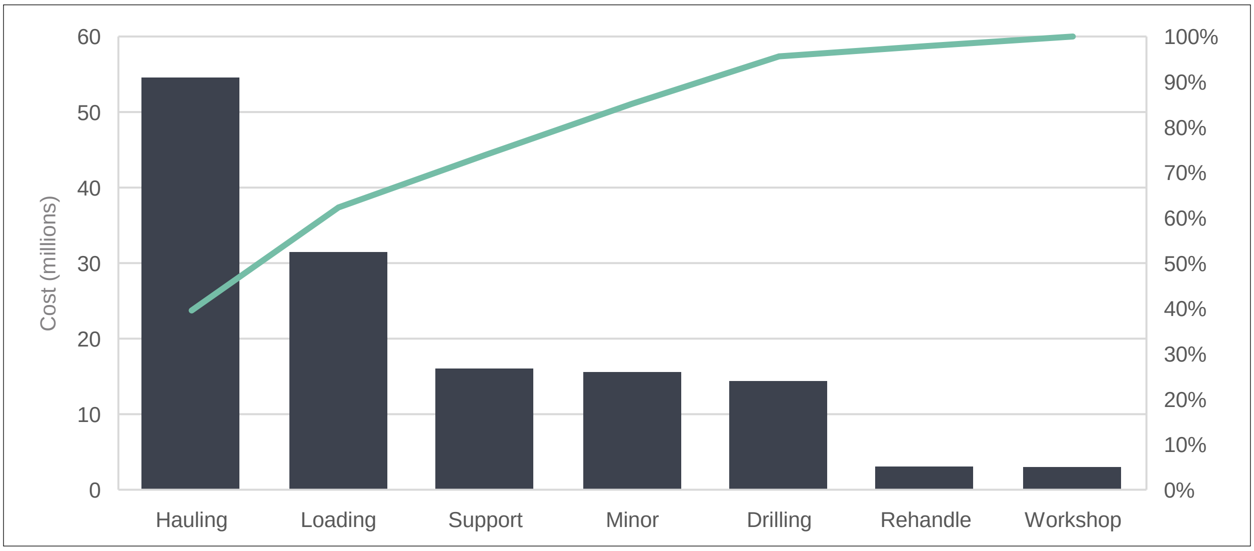


Figure 21-2: Pareto analysis of equipment maintenance costs by mining activity cost

Maintenance costs were based on the following factors:

- **Insurance:** A provision of 0.75 % per year of equipment capital cost
- **Parts:** Yellow equipment parts costing was provided by OEMs in the form of LCCs based on useful component life and rebuild component cost assumptions, with appropriate country factors, freight costs and customs duties applied
- **Servicing (lubes/ greasing, filters wear, checks):** Lubrication requirements were calculated from equipment fluid capacities with services done according to OEM recommended intervals. Oil samples will be taken from all major components at each service interval. Prices were provided by suppliers operating in Zambia
- **Buckets and bodies:** Estimated useful life and replacement costs for buckets and bodies were provided by the OEMs and applied against equipment operating hours
- **Tyres and undercarriage:** Undercarriage provisions made were in accordance with OEM LCC provided, and include provisions for grouser, roller and sprocket bolts and nuts. Tyre pricing was received from suppliers operating in Zambia
- **GET:** For earthmoving equipment and drill rigs, provision was made for GET based on expected life and costs of GET components provided by OEMs.

21.2.2.2. Equipment fuel costs

Fuel costs were calculated from fuel consumptions and equipment utilisations as discussed in Section 16.2.11 above. An average of 9.3 megalitres (“ML”) of fuel is used in mining operations each year, with a total of 106.1ML over the LOM as shown in Table 21-7. The LOM fuel cost amounts to USD 102.7 million at the base rate of USD 0.97 /L.

Table 21-7: Annual consumption of fuel by mining operations

Fuel	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Total
Consumption [ML]	8.9	10.5	10.5	9.6	9.4	9.2	9.8	9.3	8.8	9.3	6.8	3.9	**106.1**

The fuel consumption by activity is in Table 21-7.

Table 21-8: Fuel cost provisions for mining equipment

Equipment:	Loading	Hauling	Support	Drilling	Minor	Rehandle	Workshop
Fuel [ML]	20.11	54.35	9.61	5.71	8.78	2.49	1.66

The fuel price is a variable cost item influenced by several key factors including global crude oil prices, exchange rate, transportation and logistics, government taxes and levies, supply change and storage costs, fuel price adjustment by the energy regulation board and regional and international trade policies. The impact of fuel price fluctuations on the total LOM cost is shown in Table 21-9.

Table 21-9: Impact of fuel fluctuations on mining cost

Price fluctuation	Fuel price per litre	Cost impact [USD million]
Minus 20 %	0.775	-20.5
Minus 10 %	0.872	-10.3
Minus 5 %	0.920	-5.1
Base price	0.968	
Plus 5 %	1.017	+5.1
Plus 10 %	1.065	+10.3
Plus 20 %	1.162	+20.5

21.2.2.3. Blasting costs

Blasting totals USD52 million over the LOM, as shown in Table 21 9. Explosives make up the bulk of this cost.

Table 21-10: Explosives cost provisions

Operating costs	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Total
Bulk explosives	2.9	3.6	3.6	3.6	3.6	3.6	3.6	3.5	3.5	3.4	2.3	1.2	**38.53**
Packaged explosives	0.3	0.4	0.4	0.5	0.5	0.8	0.9	0.9	0.7	0.6	0.7	1.0	**7.59**
Explosives accessories	0.4	0.5	0.5	0.5	0.5	0.5	0.6	0.5	0.5	0.5	0.4	0.2	**5.83**

21.2.2.4. Labour costs

Based on the labour requirements discussed in Section 16, the annual mining labour requirements are shown in Table 21-11. Levels 7 to 11 are largely managerial and professional and remain constant, while levels 0 to 6 are more directly related to production levels and vary with LOM.

Table 21-11: Mining labour requirements

Operating costs	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Labour Grade 11	3	3	3	3	3	3	3	3	3	3	3	3
Labour Grade 10	2	2	2	2	2	2	2	2	2	2	2	2
Labour Grade 9	5	5	5	5	5	5	5	5	5	5	5	5
Labour Grade 8	4	4	4	4	4	4	4	4	4	4	4	4
Labour Grade 7	1	1	1	1	1	1	1	1	1	1	1	1
Labour Grade 6	12	12	12	12	11	9	9	9	9	9	9	9
Labour Grade 5	24	25	25	25	25	24	24	24	24	24	21	18
Labour Grade 4	4	4	4	4	4	4	4	4	4	4	4	4
Labour Grade 3	109	126	126	128	130	129	129	128	128	128	103	77
Labour Grade 2	54	62	62	64	64	62	62	62	62	62	47	34
Labour Grade 1	196	232	232	234	234	232	232	220	216	218	171	109
Labour Grade 0	1	1	1	1	1	1	1	1	1	1	1	1
Total	**414**	**477**	**477**	**482**	**482**	**476**	**476**	**463**	**459**	**460**	**370**	**267**

Applying the labour rates from mining labour rates to Table 21-11 gives the total mining labour cost of USD51 million over the LOM.

21.2.2.5. Fixed costs

As summarised in Table 21-12, the total LOM fixed cost is USD35 million.

Table 21-12: Annual fixed cost provisions

Operating costs	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	End
Establishment	3.682	0.030	-	0.033	-	-	-	-	-	0.013	-	-	-
Disestablishment	-	0.001	-	-	0.037	-	-	0.062	0.003	0.020	0.517	0.453	0.431
Running costs	2.435	2.515	2.514	2.499	2.492	2.488	2.501	2.476	2.465	2.473	2.359	2.222	-

The fixed costs comprise the following:

- **Establishment costs:** includes the costs of initial medical fitness evaluations, induction training, first PPE issue and external operator training both on an offsite and further include a provision for OEM artisan development training specific to selected equipment makes and models. One month of unproductive time is allowed while getting ready and authorised to start working on the mine. Services providers' establishment costs are included
- **Disestablishment costs:** Includes final exit medicals and one month's associated unproductive time per employee. Severance costs are allowed in the labour rates. Disestablishment costs include a provision to transport equipment from site when replaced and at the end of the project
- **Running costs:** Includes the explosives, tyre and fuel/ lube service providers, maintenance, hardware/ software, pit dewatering, insurance excess and office administration.

21.3. Mining infrastructure

21.3.1. Mining infrastructure capital expenditure

21.3.1.1. Cost summary

Table 21-13 provides a summary of the capital costs associated with the mining-specific infrastructure. This estimate, based on tendered prices, meets the accuracy requirements of a FS. The table presents the major cost buckets, categorized across a total of 11 work packages. It is important to note that these costs exclude contingencies, which will be incorporated at the overall project level.

Table 21-13: Mining infrastructure Capex

Central mining infrastructure Capex	USD
Earthworks & civils	4 766 817
SMPP (General works package)	2 578 894
EC&I	1 652 353
Buildings & facilities	2 050 237
Vendor packages	734 459
EPCM	1 034 715
Total	**12 817 474**

21.3.1.2. Basis of estimate

The mining infrastructure scope was developed and packaged into eleven tender packages, which were then released to the market as a request for quotations ("RFQ"). Each package comprised a bill of quantities ("BOQ") and/or pricing schedule, along with detailed specifications. These packages were distributed to reputable contractors and suppliers with a proven track record of successful project delivery in Zambia.

While the overall response rate to the RFQ packages was satisfactory, no quotations were received for the Electrical, Control, and Instrumentation ("EC&I") package. As a result, this package was priced using applicable database rates.

The RFQ documents provided to contractors and suppliers included comprehensive project details, drawings, specifications, BOQs/pricing schedules, and a clear scope of works for each package. A separate document outlining site-specific conditions and other specific requirements to be considered by the contractor was also included in the RFQ.

The contractors' tendered prices incorporate preliminary and general costs ("P&Gs"). P&Gs represent the contractor's indirect costs, encompassing mobilisation and demobilisation (including the setup and removal of construction plant and equipment), indirect and non-productive labour, scaffolding, safety equipment, personal protective equipment ("PPE"), transport and travel, accommodation, and supervision. P&Gs also include contractual obligations related to finance costs, insurance, bonds, and work permits.

Indirect costs also include the engineering, procurement, and construction management ("EPCM") costs associated with this specific scope of work. These costs were determined using the professional fee tables published by the Engineering Council of South Africa, which provides a fair and recognised basis for professional services contracted within the Republic of South Africa.

21.3.1.3. Battery limits

The scope included in the above estimate is specifically limited to the mining infrastructure. Typical inclusions within this scope are:

- Perimeter fencing
- Haul roads
- Bulk earthworks for mining infrastructure complex terrace
- Surface water management for mining infrastructure complex area, including a pollution control dam
- Mining production and mining maintenance facilities (wash bays, workshops, offices, change houses, etc.)
- Reticulation and supply to the various facilities, from the point of supply.

All water and power will be supplied to the perimeter of the mining infrastructure complex. This scope includes the internal reticulation and distribution of these utilities within the complex. Costs associated with establishing the external supply points up to the perimeter are covered under the processing plant capital estimate. Similarly, this scope covers the conveyance of sewage from within the mining complex to a designated tie-in point, where it will connect to the Processing Plant's reticulation system for further treatment.

21.3.2. Mining infrastructure operating costs

All costs for mining operations, including infrastructure, are accounted for in Section 21.2.2, except for maintenance. Provision is made for mining infrastructure maintenance as an annual percentage of the Capex. Table 21-14 shows the percentage applied for each category of mining infrastructure.

Table 21-14: Mining infrastructure maintenance provision

Classification	Allocation % of Capex /year
Mechanical equipment maintenance	5.0
Mechanical fixed infrastructure (tanks, dams)	2.5
Earthworks maintenance	0.5
Civil and structures maintenance	1.0
Electrical equipment maintenance	2.5
Dams and trenches	5.0
Roads	5.0
Facilities/ Buildings	1.0
Miscellaneous	0.5
Low maintenance infrastructure	0.3

The resulting maintenance cost is USD613 000 per year.

21.4. Processing plant capital expenditure

The estimate was compiled using deterministic estimating methodologies in which components of the project scope definition are quantitatively surveyed, generated, and priced using the most realistic unit prices available. Deterministic estimating methods are typically used for Class 3 through to Class 1 estimates (AACEI Classification).

This type of estimate is used for budgetary purposes and appropriation or control baseline and will have to form the basis for the completion of a capital budget estimate ("CBE") prior to project execution.

The works are to be executed on greenfield sites. The main facilities for the recovery of uranium oxide will be located close to the Muntanga and Dibbwi East pits.

21.4.1.Battery limits

Reference must be made to the execution SOW that clearly defines the incoming and outgoing battery limits of the SOW for the project.

A summary of the plant battery limits for the CPF is given below.

- Incoming
  - Bin grizzly at central plant for receiving ROM by dump truck
  - Pregnant resin receiving flange connection at central plant for transport tanker-truck hose offloading
  - Crushed ore stockpile emergency feed by FEL
  - Potable water plant distribution pump flange and boreholes
  - Raw water feed to raw water tank
  - Solution from dams/ ponds interface with pumps (penstock discharge flanges, submersible pump stilling wells)
  - Reagent supply in liquids, slurry and fine powder format from transport vehicle offloading hose connecting flanges
  - Reagent supply in solids - bulk bags hoist hooking points and drums handler hoists hooking points
  - Fuel storage fill points
  - Mobile equipment operating limits
  - The battery limit of E&I is the outgoing terminals of the 11kV MV switchgear supplied by others.
- Outgoing
  - Discharge from ore sorters discard conveyor
  - Agglomerated ore discharge conveyor discharge to heap leach
  - Barren resin load-out flange at central plant for transport
  - Gypsum conveyor discharge
  - Drummed product loaded with hoist onto pallets for storage and transport
  - Sewage water plant intake sump
  - Water distribution points
  - Possible residual dust emissions after dust suppression
  - Stack emissions to atmosphere.

The plant battery limits for the satellite facilities are given below.

- Incoming
  - ROM uranium material into the MMD Navicore ® unit feed bin
- Outgoing
  - Sorted ore stockpile.

21.4.2.Estimating criteria

The CCE has been compiled based on a full EPCM execution strategy. The capital costs have been developed from a range of sources, including finalised PFDs, defined P&IDs, defined mechanical layouts, finalised major mechanical equipment list, preliminary structural steel/ civil layout drawings, preliminary electrical and instrumentation bills of quantities and multiple source formal pricing enquiries from designated vendors and contractors to obtain the required class of estimate.

21.4.2.1. Estimating accuracy

The capital cost estimate has been prepared in accordance with the SGS Bateman Estimator's Best Practice Guide PCNG-0920-002 Rev 1 definition for a Class 2 estimate classification giving an accuracy range of -15 % to +20 % at a probability of no more than 80 % after contingency is applied and this conforms to the Association for the Advancement of Cost Engineering International ("AACEI") Class 2 estimate classification guidelines. SGS Bateman has excluded the project contingency from the CCE based on the Client's instructions.

Table 21-15: Cost estimate classification matrix for the process industries

		Primary characteristic	Secondary characteristic		
Estimate class	**Also known as**	**Key technical deliverables Input to cost estimate**	**Expected accuracy range**	**Methodology**	**Purpose of estimate/End usage**
Class 2	Feasibility study Detailed estimate Definitive estimate.	Complete: mass balances, PFDs, P&IDs, mechanical equipment list, site plot plans & site general arrangement (“GA”) drawings, specs & data sheets, electrical single line diagram, equipment list for all engineered items, detailed BOQs/MTOs for all disciplines, motor list, instrument list, execution plans. Preliminary/complete: mechanical layouts, civil/structural/ piping layouts, isometric drawings, control philosophy.	Low: -5 % to -15 % High: +5 % to +20 %	Detailed MTOs, unit costs with some forced detailed take-offs, planning, defined work packages, firm bids/tenders.	Appropriation or control, detailed control baseline for monitoring variation to budget

For standardisation, the AACEI recommended practice No. 18R-97 is used as a base guideline with specific requirements for SGS Bateman added.

21.4.2.2. Base date

The base date of the estimate is September 2024. Forward escalation is excluded.

21.4.2.3. Base currency/ exchange rate

The capital estimate has been compiled in South African Rand (“ZAR”) and presented in United States Dollars (“USD”). Prices obtained in other currencies have been converted to ZAR using a spot rate of exchange given by the Client on the 8th of August 2024 via email, and then to USD.

Table 21-16: Currency forecast for Muntanga

Currency	RoE used
ZMW:USD	26.00
ZAR:USD	18.50
EUR: USD	1.09

No foreign currency or rate of exchange variations were allowed for in this estimate. It will fall within the client’s SOW to make adequate provision and risk allowance for rate of exchange variations.

21.4.2.4. Scope definition

The estimate is based on the scope as defined within this document and by the engineering documentation such as:

- Completed PFDs
- Defined P&IDs
- Preliminary civil/ structural layouts
- Preliminary civil/ structural BOQ
- Defined plot plan
- Defined mechanical layouts
- Finalised major equipment list
- Detailed minor equipment list
- Develop the main cable routing on plant layout drawings
- Preliminary load list
- Preliminary electrical BOQ
- Control system topology
- Instrument list
- Preliminary C&I BOQ
- Preliminary line and valve list
- Major pipe routes identified with preliminary layouts produced
- Preliminary piping BOQ.

21.4.2.5. Pricing basis

Pricing for the direct works is based on a variety of sources as follows:

- Single and/or multiple sources fixed and firm quotations
- Single and/or multiple budgetary quotations
- Provisional sum allowances
- Factored or estimated allowances.

21.4.2.6. Presentation of capital cost

The overall capital cost estimate is compiled using Microsoft Excel spreadsheet format.

21.4.2.7. Capital estimate structure

The cost estimate is compiled in line with the work breakdown structure (“WBS”) and project cost breakdown structure (“CBS”).

Table 21-17: Project area summary

Area description	Total [USD]
Direct field costs	
Bulk Earthworks and Plant Roads	3 747 557
Civil Works	12 037 999
Process Plant Buildings	3 347 539
Structural Steelwork	4 134 698
Platework & Liners	4 513 604
Conveyors	4 263 027
Mechanical Equipment	27 996 183
Piping & Valves	13 692 629
Electrical	6 140 663
Instrumentation	3 642 853
Plant and Vehicles	1 072 805
BE P&G s	885 259
CIVIL P&G s	8 318 923
SMPP P&G s	15 494 585
SITE FAB TANKS P&G s	651 794
E&I P&G s	996 120
Transportation of Equipment (DAP) to site	3 766 764
Commissioning Spares	1 007 084
Vendor assist during Constr & Comm	1 925 094
Total direct field costs	**117 635 180**
Home office & indirect field costs	
Home Office Resources - Engineering	2 218 331
Home Office Resources - Drawing Office	1 583 636
Home Office Resources - Management & Services	1 897 472
Construction Management	5 409 911
Commissioning Resources - Site Based	1 028 609
Technical Support (Site Visit) during Construction	425 740
Home Office Traveling & Accommodation	58 658
Site Costs (Mobilization De-Mobilization & Running Costs)	1 537 175
External Consultants	255 362
Total H.O. & indirect field costs	**14 414 894**
Total net cost	**132 050 074**
Other Costs	
Bonds Guarantees etc @ 0.25 % of TNC	330 125
Insurance	2 329 892
Total other costs	2 660 017
Overall project cost	**134 710 091**

21.4.3.Direct costs

Material and labour quantities were obtained as described below and converted to the CCE by the application of unit cost and unit rates or formal budgetary/ fixed and firm quotations for the supply, fabrication, construction and installation of the various materials and equipment.

21.4.3.1. Bulk earthworks and plant roads

The bulk earthworks and plant roads bill of quantities has been priced using current market-related rates competitively tendered by earthworks/ civil contractors. Preliminary and general costs are included in the contractor's tendered price.

The contractor’s indirect costs (P&Gs) cater for the contractor’s mobilisation and demobilisation including the establishment and later removal of construction plant and equipment, contractor’s manual indirect and non-productive labour, scaffolding, safety equipment, personal protective equipment, transport and travel, accommodation and supervision including contractual requirements relating to finance costs, insurance, bonds, and work permits.

21.4.3.2. Civil works

Civil works quantities for the project have been developed from preliminary site layout drawings. The civil BOQs have been priced using current market-related fixed and firm rates competitively tendered by civil contractors. P&G costs have been included in the contractor’s tendered price.

21.4.3.3. Architectural/ building

All BOQs for buildings have been priced using market-related fixed and firm prices competitively tendered by suppliers of modular building solutions fit for the project. P&G costs have been included in the contractor’s tendered price.

21.4.3.4. Structural steel

Structural quantities were developed from the preliminary 3D model. The structural BOQs were priced using current market-related fixed and firm rates competitively tendered by structural, mechanical, piping and plate work (“SMPP”) contractors. P&G costs have been included in the SMPP contractor’s tendered price.

The structural steelwork cost includes supply, fabrication, surface protection, delivery to site and shop detail drawings and final painting.

21.4.3.5. Plateworks and liners

Platework items, specifically tanks, were broken down into two separate work packages. Shop-fabricated plate work items, including tanks, hoppers, chutes, and sumps form part of the SMPP contractors’ scope of supply while site-fabricated tanks form part of the Site Fab Tanks’ scope of supply with each contractor responsible for the installation of their scope respectively. Both contractors have allowed for P&G costs for their respective SOWs.

- SMPP Contractor Scope (X_M001)
  - Shop-fabricated plate work
    - Tanks
    - Hoppers
    - Chutes
    - Sumps, etc.
  - Responsibilities
    - Fabrication, supply, surface protection, freight, and installation of all shop-fabricated items, including associated linings (rubber linings, internal surface treatments, and liner plates where required).
  - P&G costs
    - These have been included in the contractor's tendered price for their SOW.
- Site Fab Tanks Contractor Scope (X_M002)
  - Site-fabricated tanks
    - Tanks fabricated on-site, which are separate from the shop-fabricated tanks
  - Responsibilities
    - Fabrication and installation of site-fabricated tanks, including associated preliminary and general costs as part of their tender
- Cost Estimation
  - Quantities
    - Derived from the preliminary 3D model, data sheets and mechanical layout drawings
  - Pricing
    - Unit rates for pricing plate work items, including tanks and associated linings, were based on the SMPP contractor’s competitively tendered price
    - The contractor’s prices include the costs for supply, detailing, fabrication, surface protection, freight, and installation
    - The specific items included in this cost (for both shop and site-fabricated tanks) account for rubber linings, internal treatments, and any necessary liner plates.
- P&G costs
  - Both contractors have accounted for their respective P&G costs in their respective tendered prices, covering overheads and any non-direct project expenses
- Key points
  - Two separate work packages for shop-fabricated and site-fabricated tanks

- Both contractors are responsible for the installation of items within their scope
- Pricing for the plate work items includes full scope costs from supply to installation, including surface protection and linings where necessary
- The quantities and pricing were based on 3D models, data sheets and mechanical layout drawings, with unit rates from the SMPP contractor's tender.

21.4.3.6. Mechanical equipment

Pricing of mechanical equipment has been based on fixed and firm prices for the items identified in the mechanical equipment list developed from the initial PFDs, subsequent P&IDs, plant layouts and general plant arrangement drawings. Some package suppliers, requiring extensive design work to submit a tender, responded with FS budgetary quotations without comprehensive requested detailed design information for this phase.

Enquiries were issued to multiple vendors, however in some cases with specific technology supply packages, only a single supplier responded suitably.

SGS Bateman's procurement department revalidated the vendor packages in July 2024. Revalidated proposals were received from the selected vendors only.

BOQs were developed for equipment installation and using current market-related fixed and firm rates competitively tendered by SMPP contractors. P&G costs were included in the capital estimate based on the SMPP contractor's tendered price.

21.4.3.7. Piping and valves

The piping quantities were prepared by the engineer based on the P&IDs. The quantities were developed from a preliminary 3D piping model routed as per the preliminary plant layout. The piping quantities were priced for supply and installation using the SMPP contractor's tendered price. The pricing for the supply of valves and special piping items is based on enquiries issued to multiple vendors. Pipe supports (including u-bolts/shoes/ goal posts/ angles/ channels etc. 0kg to 5kg) are included under the piping BOQ and priced by the SMPP contractor.

The installation pricing for the valves, special piping items and pipe supports were priced using current market-related fixed and firm rates competitively tendered by the SMPP contractor. P&G costs have been included in the SMPP contractor's tendered price.

21.4.3.8. Electrical

Electrical equipment (e.g. transformers, MCCs, switchgear, UPS, etc) was developed and issued to multiple vendors for budget pricing. Electrical bulk material supplies (cables, cable trays, etc.) were priced using E&I contractor's tendered pricing. P&G costs have been included in the contractor's tendered price.

21.4.3.9. Control and instrumentation

Instrument equipment (e.g. systems integrator, field instruments, etc) was developed based on the topology diagrams and P&IDs and issued to multiple vendors for market-related budget pricing. C&I bulk material supplies were prepared based on P&IDs and preliminary plant layout drawings and priced using E&I contractor's tendered pricing. P&G costs will be included in the contractor's tendered price.

21.4.3.10. Preliminary and general costs (note)

Due to the change in scope, with the Satellite Plant being removed, P&G costs have been calculated for the Central Plant by using a factored percentage of the initial P&G costs included in the tendered prices submitted by the various contractors. The following contract packages were adjusted using factored percentages:

- Bulk earthworks and plant roads
- Civil work
- SMPP (split on weighted average cost per discipline)
- E&I.

21.4.4. Allowances

Growth allowance is normally applied to estimates where material take-offs are performed. Where part of the conceptual cost estimate is done in more detail, then the allowance for growth will be agreed upon in consultation with the discipline engineers for only those specific portions of the estimate.

Quantity and price growth allowance have been applied to the estimate based on the degree of engineering completed and the quality of pricing information that supports the estimate. The average growth allowance for the capital cost estimate is 8.2 % for all direct field costs, a total of USD8.9 million.

21.4.5. Transport

The costs for sea freight and road freight are included in the estimate for delivery of equipment from the country of origin to the site based on rates received from the specific vendors. Where no rates were given an estimated percentage of the supply costs was used for delivery to the site. Transport costs have been allocated at the package level. Transport costs are based on delivery and place ("DAP") according to Incoterms ® 2020. No allowance has been made for duties and taxes. Total transport costs are USD52 million

21.4.6. Spares

The estimate provides for commissioning spares. The spare cost has been allocated to each discipline at the package level using market-related pricing.

The following spares and consumables will be included in the capital cost estimate:

- Commissioning or initial spares (included in the CCE)
- Operational spares (excluded in the CCE – included in Opex)
- Critical spares (excluded in the CCE).

21.4.7. First fills (oils, lubricants)

The cost for the first fill of lubricants is included in the vendor's quotations. Discipline engineers have verified if the first fills provided by the vendors are technically compliant with the project requirements.

21.4.8. Vendor assistance

The cost for vendors' assistance during construction and commissioning has been included in the vendor's quotations. Discipline engineers have verified that the vendors' assistance cost provided by vendors is technically compliant with the project requirements.

Vendor assistance typically includes the following:

- Installation costs
- Commissioning costs
- Accommodation, transport and messing and
- Airfares per person per visit.

21.4.9. Indirect field costs

Indirect costs are generally time or duration-based and include items that are necessary for the completion of the project but are not related to the direct construction costs.

The project indirect field costs ("IFC") are dependent on the project duration.

21.4.9.1. Engineering, design and project management

These costs cover the project management, engineering and procurement, construction management and commissioning assistance up to C3 (EPCM) costs directly associated with the implementation of the project. The manhour loading has been developed based on SGS Bateman's experience and in accordance with the project execution schedule. No Allowance has been made for the training of local personnel and defect liability period.

21.4.9.2. Bonds, guarantees etc

An allowance for bonds and guarantees based on a factored percentage is included in the estimate.

21.4.9.3. Project insurance

Insurance included in the estimate is an allowance for project-related risks which are insurable. It is dependent on project variables and project-specific circumstances. It typically includes the following:

- Contractors All Risk on construction and site activities typical cover – this depends on the extent of cover required
- Third Party Liability insurance typical cover

- Medical Evacuation and CasEvac typical cover. This depends on the area, location, and the detailed circumstances
- Marine Cargo and difference in excess typical cover.

The following risks have not been included in the estimate and thus excluded due to the specific requirements the owner may have. These should be strongly considered in addition to those listed above:

- Delay in start-up insurance ("DSU")
- Project-specific required professional indemnity
- Advance loss of profits ("ALOP").

The insurance estimate allowance should be finalised by the performance of an insurance review by specialised parties, once the exact requirements of the owner and the project are available in more detail. An allowance based on a factored percentage of the total net cost has been included in the estimate as a guide. However, formal quotations for project insurance should be sourced and issued by the client.

21.5. Process plant operating expenditure

21.5.1. Operating cost summary

The Opex cost estimate has been developed for each pit to allow for the different processing options and individual reagent consumptions. The applicable Opex costs when treating or from the various pits are summarised in Table 21-18. The estimate includes reagents and consumables, fuel, labour, maintenance materials and power consumption.

Table 21-18: Opex cost estimate

Cost component	CPP Dibbwi East Oxide ore	CPP Dibbwi East Reduced ore	CPP Muntanga
	Total	Total	Total
	[USD/a]	[USD/a]	[USD/a]
Fixed costs	5 174 400	5 174 400	5 179 188
Labour	1 307 346	1 307 346	1 307 346
Maintenance	2 145 888	2 145 888	2 145 888
Waste Disposal	128 859	128 859	133 648
Laboratory	1 592 306	1 592 306	1 592 306
Variable costs	18 871 335	28 210 225	18 568 017
Power	5 372 000	5 372 000	5 372 000
Fuel	281 144	281 144	281 144
Reagents	13 218 191	22 557 081	12 914 872
Total	**24 045 735**	**33 384 625**	**23 747 205**
Cost component	**CPP Dibbwi East Oxide**	**CPP Dibbwi East Red**	**CPP Muntanga**
	Total	**Total**	**Total**
	[USD/t]	**[USD/t]**	**[USD/t]**
Fixed costs	1.5	1.5	1.5
Labour	0.4	0.4	0.4
Maintenance	0.6	0.6	0.6
Waste disposal	0.0	0.0	0.0
Laboratory	0.5	0.5	0.5
Variable costs	5.4	8.1	5.3
Power	1.5	1.5	1.5
Fuel	0.1	0.1	0.1
Reagents	3.8	6.4	3.7
Total	**6.9**	**9.5**	**6.8**
Total	**24 045 735**	**33 384 625**	**23 747 205**
Total plant feed (t/a)	**3 500 000**	**3 500 000**	**3 500 000**
USD/t of total feed to process plant	**6.9**	**9.5**	**6.8**
USD/lb U_3O_8	**11.78**	**16.36**	**11.22**

21.5.2.Basis of estimate

21.5.2.1. Scope

The Opex costs can be categorised as fixed or variable costs.

Fixed costs include:

- Manpower (Labour for plant operation and maintenance only)
- Maintenance and operating supplies.

Variable costs include:

- Power
- Reagents, fuel and consumables
- Waste handling.

21.5.2.2. Accuracy

The accuracy for the Opex cost estimate is as for the capital expenditure ("Capex") cost estimate. The methodology used in preparing the Opex costs in this report was based on the scope, pricing and information available at the time but should be revised as the industry fluctuates, especially in terms of reagent pricing. It must be noted that these costs apply only to the operation of the plant at full capacity, under name-plate design conditions. During commissioning, start-up and ramp-up the unit costs will vary in comparison to when the plant is operating at full capacity.

21.5.2.3. Exchange rate

Opex costs are base-dated November 2024. The estimates are presented in United States dollars ("USD") and the exchange rate used is South African Rand ("ZAR") 18.0/ USD.

21.5.2.4. Exclusions

The following are excluded from this estimate:

- Mining Costs (outside of project scope)
- Tailings handling costs (outside of project scope)
- General, medical and administration costs, other than plant and technical/engineering services
- Security costs
- Duties and taxes on exports of products
- Marketing costs
- Depreciation and replacement capital
- Insurance
- In-country corporation tax
- First, fill reagents costs (included in the capital estimate for owners' costs)
- No provision for annual increases in salary, services and supplies growth has been allowed
- Product dispatch including handling and cost of transport for products from site to destination (outside of project scope)
- Contingency.

21.5.3. Fixed costs

21.5.3.1. Labour

The salary grades of plant personnel used to derive the overall plant labour contingent are shown in Table 21-19.

Table 21-19: Personnel grades (supplied by GoviEx)

Personnel supplied by GoviEx	Grade	Monthly salary [USD]
Unskilled labour	1	417
Skilled labour	2	482
Technical	3	574
Supervisors	4	730
Engineers/technical	5	943
Senior engineers	9	4 957
Manager	10	12 779
Senior manager	11	15 725

Labour contingents have been derived for the selected flowsheet and are inclusive of production, safety, health, environmental and quality (“SHEQ”), laboratory, product dispatch and maintenance staff to meet typical South African regulations in terms of health and safety (“H&S”) standards. It excludes the heap leach and non-technical staffing like administration, finance, human resource, medical and procurement. The production labour force is based on operating the process plant for three eight-hour shifts per day, seven days a week. Shift rotations will comprise a total of three operating crews with allowance made for a standby crew.

The manning structure for production was derived by SGS Bateman for each area, based on previous studies on uranium processing plants, together with input from GoviEx. Allowance has been made for shared resources across areas. The compliment allows for leave to relieve personnel in critical production areas. Maintenance staff is accounted for in the process plant technical and management contingent. Salaries have been calculated based on a basic wage plus additional allowances applied for the labour contingent to establish the total cost to the company. A total of 169 operating personnel is estimated to be required as shown in Table 21-20.

Table 21-20: Total processing plant complement

	Central
Management/ Admin	60
Supervisors	23
Operators	39
Labour	34
Total	**156**

21.5.3.2. Maintenance

In all cases, the cost of maintenance supplies is calculated as a factor of the mechanical equipment supply costs excluding piping and valves, electrical and instrumentation based on previous studies for typical uranium hydrometallurgical plants but excludes the maintenance associated with the acid plant. Generally, maintenance materials are considered to be (7 % to 13 %) of the mechanical equipment supply cost. An estimate of 7 % has been applied. The maintenance labour component has been allowed for in the annual labour estimate.

21.5.4. Variable costs

21.5.4.1. Power

The overall operating power consumption is estimated at 5.37 MW for the central plant and 0.56 MW for the satellites. The power requirements for each process area were estimated from tenders available, database information and reduction factors applied to the loads per area as required.

A unit energy supply rate of USD 0.125 /kWh was supplied by SRK. An overall annual plant availability of 80 % has been used. It is based on all normally operating equipment (i.e., ignoring stand-by units). Allowance has been made for absorbed power using load factors (0.80x) applied to the actual motor kilowatts for drives installed for operational use.

21.5.4.2. Reagents and consumables

Prices for major reagents and consumables for this estimate were based on information supplied from GoviEx and budget-level quotations obtained from chemical suppliers in both South Africa and Zambia.

21.5.4.3. Reagent pricing

No cost has been allocated for raw water supply based on recommendations by GoviEx. This is due to the raw water being supplied internally to the project with the operating costs for the supply being included in the mining scope.

21.5.4.4. Reagent consumptions

Consumption of all the major process reagents was determined by the test work conducted at Mintek. The consumption of hydrogen peroxide in the test columns was however inflated due to the iron that was added to initiate the leaching process. The peroxide consumption was thus adjusted to take into account the iron will be present in the recycled barren solution in the actual plant. The source of reagent consumption data that is used in the Opex calculation is shown in Table 21-21.

Table 21-21: Reagent consumption

Reagent	Consumption source	Comment
Sulfuric Acid (98 %)	6 m column test work	0.98 scale-up factor added
Hydrogen Peroxide (50 %) for leach	6 m column testwork	Leach consumption adjusted for iron concentration
Hydrogen Peroxide (50 %) for precipitation	Testwork	~3x stoichiometric
Caustic Flakes (98 %)	Testwork	
Lime (92 %)	Testwork	
Flocculant	Estimated	For Gypsum precipitate thickening
IX Resin	Estimated annual loss	Purolite MTA8000PPSO4
Nanofiltration membranes	Estimated by vendor	6-month membrane life

21.5.4.5. General consumables

General plant consumables such as filter cloths, plant safety equipment and gear, minor chemicals, laboratory chemicals, office items, packaging and waste bags have been allowed for as 5 % of the overall reagent cost. Details of the delivered prices used, and sources are shown in Table 21-22.

Table 21-22: Reagent and consumable cost summary

Reagents and Consumables	Unit Delivered Cost [DAP] USD/t	Supplier	Source country
Sulfuric acid (98 %)	150	KEMCORE	South Africa
Hydrogen peroxide (50 %) for leach	1 141	Marlyn Chemicals	South Africa
Hydrogen peroxide (50 %) for precipitation	1 141	Marlyn Chemicals	South Africa
Caustic flakes (98 %)	1 084	Marlyn Chemicals	South Africa
Lime (92 %)	150	Newcrest Lime	Zambia
Flocculant	2 854	Protea	South Africa
IX Resin	4 574	Purolite	South Africa
Nanofiltration membranes	3 577	Watercare	South Africa

21.5.4.6. Fuel

The annual fuel costs were estimated for mobile plant equipment used within the plant boundaries and required for daily operations. The fuel cost is based on an estimated fuel consumption rate of 35 litres per hour (“L/h”) for mobile cranes, 3 L/h for forklifts and 25 L/h for light vehicles. Diesel cost of USD0.97 /L per was used in the model.

21.5.4.7. Waste services

An estimated annual allowance made for waste management services based on USD138 /t of U_3O_8 product. This cost was derived from a typical South African uranium plant case.

21.6. Heap leach facilities

21.6.1. Capital cost – Heap leach pad materials handling (stacking and reclaiming)

21.6.1.1. Cost summary

The pre-production capital cost is presented in Table 21-23.

Table 21-23: Pre-production capital cost

Item	Capital cost [USDM]	Remarks
MHS1 stacking system	6.28	Original equipment manufacturers ("OEM") & construction / Installation
MHS2A/2B spent ore reclaiming system	10.72	
Mobile equipment (spent ore reclaiming and movement of grasshoppers)	5.0	E.g. 2 x WA900, WA500 FEL
Total	22.00	

The sustaining capital cost for additional or replacement equipment is presented in Table 21-24.

Table 21-24: Sustaining capital cost for additional/ replacement equipment

Item	Capital Cost [USDM]	Remarks
MHS2B additional conveyors	1.27	Year 7
Mobile equipment replacement	5.00	Year 7
Total	**6.27**	

21.6.1.2. Basis of estimate

Prior to receipt of the final processing testwork and the switch to an on-off (dynamic) leach system, FS development focused on a static HL using a retreat stacking system of grasshopper conveyors in 10m lifts with the stacking system fed from an overland conveyor with motorised tripper.

- 1 500 m Overland with motorised tripper
- 2 x MHS (each MHS comprised 2 x RCV, 10 x HOCV, TCV, SFC and SCV conveyors).

For this design, pricing enquiries were requested from Tier 1 Materials Handling OEM (TAKRAF, RBL-REI, Nepean, etc).

OEM supply pricing was received from three Tier 1 suppliers, and total costs from all three were within ±15 % of the average of costs received, showing a close grouping and, noting that site installation, construction, and vendor representation are excluded, and showing the relatively similar order of magnitude of total cost. With the change of processing strategy to on-off (dynamic), pricing enquiries were resubmitted. The equipment stacking height, operating hours and philosophy, belt capacity and widths remained the same; however, only one supplier responded, and this quotation is the basis for FS capital cost for the supply of equipment.

The OEM costs include:

- Detailed design and specification
- Manufacture, fabrication and delivery of equipment and materials
- Vendor representation for commissioning.

Installation, construction and erection contractor costs, including contractor preliminary and general ("P&G") costs, are based on the pricing enquiries for the processing plant.

Engineering, procurement (OEM), and construction management costs are factored in and cognisant of the detailed engineering for equipment being allowed for in the OEM costs.

21.6.2.Operating cost (heap leach pad and spent ore dump operations) summary

The steady-state (3.5 Mtpa) operating costs for the process plant are shown in Table 21-25.

Table 21-25: Operating costs at 3.5 Mtpa

Item	Annual cost [USD/Annum]	USD/t	Remarks
Stacking - dynamic pads			
Expenses (allowance)	29 121	0.01	HLP office expenses
Stacking system CV - Power	152 054	0.04	Project power tariff
Stacking system CV - Wear parts	70 000	0.02	2 % of OEM equipment
Mobile equipment - Ops	1 557 446	0.44	FEL reclaim
Mobile equipment - Fuel	853 903	0.24	Project fuel rate
Labour	278 732	0.08	Project labour rates
Reclaim - SOD			
Expenses (allowance)	11 876	0.00	HLP office expenses
Reclaiming CV system - Power	242 310	0.07	Project power tariff
Reclaiming CV system - Wear parts	112 000	0.03	2 % of OEM equipment
Mobile equipment - Ops	448 976	0.13	D8 dozer
Mobile equipment - Fuel	282 798	0.08	Project fuel rate
Labour	101 472	0.03	Project labour rates
Total	**4 140 688**	**1.18**	

21.6.2.1. Basis of estimate

- Power cost is 0.125 USD/kWh
- Conveyor absorbed power is 70 % of installed motor capacity and annual cost is based on an average of 70 % of grasshoppers being in continuous operation
- Operating hours for conveyors = 6 570 hours per annum
- Wear parts cost per annum 2.0 % of OEM equipment
- Diesel price is 0.97 USD/L
- Tonnage per annum is 3.5 Mtpa
- Labour: four shifts per day per equipment item and job grades 1 and 2 for operators
- Equipment maintenance labour and operations are undertaken at a central workshop at the plant
- DHLP office expenses cover light vehicle fuel and facilities maintenance
- Operation, wear and replacement of solutions system pumps and pipes considered under the plant costs
- The FEL WA500 for moving mobile conveyors is shared between the stacking and reclaiming operating costs
- Operating costs for mobile equipment are based on the FS pricing for mining operations.

21.7. Access roads

21.7.1. Capital cost

21.7.1.1. Cost summary

The pre-production capital cost is presented in Table 21-26. The capital cost estimate is considered to be a FS estimate and is considered to align with guidance provided by the Association for the Advancement of Cost Engineering International ("AACEI") for a Class 3 Estimate. SRK has excluded the project contingency based on the client's instructions, and owner's costs, which will be estimated by GoviEx.

Table 21-26: Pre-production capital cost

Item	Capital cost [USDM]	Remarks
Muntanga PAR	4.81	0+000 to 10+150
Machinga River Bridge	1.76	Composite construction method
RAP1, RAP2, Kashunde Link	1.71	16 000 m total length
Camp Earthwork & Access Road	0.50	50 x 100 m platform + access road
Total	**8.78**	**Total value**

21.7.1.2. Basis of estimate

The road BOQ has been priced using current market-related requests for budget quotations from experienced and suitable earthworks / civil contractors.

A request for quotation for the construction of Muntanga and Gwabi roads was circulated to six pre-selected Contractors to obtain estimates for road construction. Four quotations were received from both local and international Contractors.

A request for quotation for the construction of the Machinga River Bridge was circulated to four pre-selected Contractors to obtain estimates and four quotations were received.

The request for quotation documents provided to the contractors included details of the project, drawings, specifications, BOQs derived from the preliminary 3D model, and details of all works to be included by the contractor under the proposed package of works. For the Machinga River Bridge, the engineer's tender bill of quantities and drawings were provided.

The battery limits of the works are as follows:

- **Start**: connection to the edge of national road pavement or in the case of RAP roads, connection to a primary access road
- **End**: Muntanga processing plant entry gates or mine entry gates, or in the case of RAP roads, the fence line of the relocation village.

P&Gs are included in the contractor's tendered price. P&Gs are the contractor's indirect costs and cater for the contractor's mobilisation and demobilisation including the establishment and later removal of construction plant and equipment, contractor's manual indirect and non-productive labour, scaffolding, safety equipment, personal protective equipment, transport and travel, accommodation and supervision including contractual requirements relating to finance costs, insurance, bonds, and work permits.

Estimates of indirect costs are as follows:

- EPCM costs have been estimated at 10 % of direct costs.
- Owner's costs and contingency are excluded in accordance with the Client's requests.

21.7.2. Operating costs

The operating costs are summarised as:

- There is an allowance in the sustaining capital schedule for replenishment of the wearing course on the Muntanga primary access road
- Dust suppression is by mobile water bowser included in the mining mobile fleet
- The mining scope of work includes the installation and operatives required to temporarily halt road operations during blasting.

21.8. Power

21.8.1. Estimated costs

21.8.1.1. Capital cost summary

The pre-production capital cost is presented in Table 21-27.

Table 21-27: Pre-production capital cost

Item	Capital Cost [USD]	Remarks
Connection	1.41	Siavonga substation works
Transmission line	7.22	39km, 132kV OHL
Main substation	6.19	132/33/11kV substation
Plant substation	2.42	11kV board
Distribution (selected)	0.59	Camp, dewatering
Total	**17.82**	

21.8.1.2. Basis of capital estimate

The cost estimate represents the engineer's estimate and is based on the unit cost tables for supply as advised in the 2022 Zambia Integrated Resource Plan and benchmarked, and adjusted where necessary, by Utilink for recent similar projects undertaken in Zambia and Southern Africa. Where required costs have been inflated, this is in line with Zambian inflation (2022 to 2024).

These costs are representative of "competitively tendered, as constructed" costs on an EPC turnkey basis (i.e. a "tender design cost") inclusive of the following:

- Mobilisation
- Detailed design and specification
- Manufacture, fabrication and delivery of equipment and materials
- Installation, construction (civil works etc.), and commissioning.

The costs are at an appropriate level of accuracy for this stage of design and subject to review of the SOW, adequate for providing a check cost for future tendering.

The Engineer's costs are exclusive of the following, which need to be considered by SRK/ GoviEx:

- Final engineering updates
- Geotechnical and topographic surveys
- EPC tendering process
- Land acquisition
- Environmental and social surveys and assessments
- Relocations work
- Changes in scope
- Contingency
- Owner's costs
- Project implementation team costs/ ZESCO costs.

The costs include allowances for detailed surveys and owner's engineer costs. A contingency of 10 % was recommended to be added at a project level.

21.9. Water management

The capital expenditure for all surface water management, detailed in Section 18.3, is shown in Table 21-28.

Table 21-28: Water management capex

Item	Total [USD]
Surface water infrastructure works	
Clean surface water channels and dissipators	161 880
Clean surface water storage sump	73 240
Attenuation dam & stormwater diversion berm	606 500
Excess water dams	785 750
Borehole drilling, pumping tests, and associated piping	2 223 500
P&Gs Items (25 %)	962 720
Professional fees	481 360
Total	**5 294 950**

21.10. Resettlement

The budget for the resettlement action plan discussed in Section 20 is a total of USD3.0 million. The breakdown of the RAP budget is shown in Table 21-29.

Table 21-29: RAP budget breakdown

Item	Total [USD]
Compensation	**1 786 503**
Replacement Houses - Central (including Land Preparation)	1 341 272
Water System for the School - Solar Powered at 4 Sites	60 000
Replacement Churches - Central	25 000
Construction of School between Muntanga and Kashundi Resettlement Sites - Standard School in line with the Government Designs with solar	98 846
Construction of 7 Staff Houses at the School - 4 Houses with Toilet based on 3 Roomed RAP houses	94 231
Water System for the School - Solar Powered	15 000
Construction of Police Post between Muntanga and Kashundi Resettlement Sites	49 077
Construction of Staff Houses - 2 Houses with Toilet based on 3 Roomed RAP Houses Designs	26 923
Water System for the Police Station - Solar Powered	15 000
Construction of Clinic between Muntanga and Kashundi Resettlement Sites	19 231
Construction of Staff Houses - 2 Houses with Toilet based on 3 Roomed RAP Houses Designs	26 923
Water System for the Clinic - Solar Powered	15 000
Compensation for livelihood restoration and agriculture activities	**907 364**
Land Value - Central	301 331
Livelihood Restoration Measures and assistance to Vulnerable Groups	410 000
Disturbance Allowance - Central	112 033
Grievance Management	84 000
Implementation costs	**322 651**
Compensation for Land	150 000
Capacity Building and Strengthening of Institutions	35 000
Management	137 651
Total RAP budget	**3 016 518**

21.11. Other costs

21.11.1. Product transport costs

As discussed in Section 19.6.1, quotations were provided that covered transportation costs from the Project to Comurhex in France, Cameco in Canada and ConverDyn in the USA. The average transport cost is USD1.45 /lb U_3O_8 from Muntanga to final conversion destination, based on quotation.

21.12. General and administrative

21.12.1. Capital expenditure

Capex for general and administration ("G&A") is shown in Table 21-30. Note that G&A office and other facilties are included in the main processing plant facilities.

Table 21-30: G&A Capex

Item	Cost [USD]
G&A vehicles	720 000
Kitchen equipment	68 278
Medical capital costs	86 156
Ambulance	150 000
Camp construction and furniture	1 960 369
G&A office furniture	159 030
Environment	554 194
Total G&A capital costs	**3 698 027**

21.12.2. Operating costs

The steady-state G&A labour complement is shown in Table 21-31. The personnel number ramps up gradually during project development and commissioning, reaching the full complement four years after the start of project development.

Table 21-31: G&A labour complement

Position	Level	Number
General adminstration		
General manager	12	1
Administration manager	7	1
IT officers	5	2
Admin assistant	4	2
Secretary	2	2
Drivers	1	4
Cleaners	0	28
Security		
Security manager	7	1
Security assistant	2	2
Internal security	1	28
Human resources		
HR manager	7	1
HR officer	5	1
HR assistants	4	3
Health, safety & environmental		
HSE manager	8	1
CSR engagement manager	7	1
Safety & environmental radiation officers	6	6
Community engagement	3	1
HSE assistants	3	3
Medical		
Medical doctor	7	1
Paramedic	4	3
Nurses	3	3
Stores		
Stores supervisor	4	1
Logistics/ procurement	3	2
Warehouse assistants	2	2
Warehouse assistants	1	6
Laundry		
Laundry supervisor	2	1
Laundry attendants	0	12
Total		**119**

Applying these numbers to the costs per employment level in Section 21.1.1 results in a steady-state G&A labour cost of USD1.3 million per year.

Other G&A costs are shown in Table 21-32.

Table 21-32: Other G&A costs

Other G&A costs	USD/ year
Directors' fees	8 182
Property licenses	80 260
CSR and donations	100 000
Annual fuel (200L/Vehicle per month)	40 320
Rent	19 000
Professional fees (corporate and staff)	12 000
Office supplies and maintenance	68 469
Communication	21 954
Catering	806 014
Camp accommodation management	90 480
Medicals	60 000
Total G&A other costs	**1 306 679**

21.13.Summary

21.13.1. Project capital costs

Based on the Capex descriptions in the section and sustaining capital requirements, the LOM capital cashflows are as shown in the following tables.

The initial Capex is the expenditure required to purchase the initial mining fleet, develop the processing plant and build all roads and infrastructure, up to the point where mining production can commence and revenue is received.

The total initial Capex is USD 282 million as shown in Table 21-33

Table 21-33: Initial development capital

Development capital [USD '000]	Total	2025	2026	2027
Mining equipment	**36 887**	0	0	36 887
Mining infrastructure	**14 099**	570	7 657	5 872
Processing plant	**137 721**	143	44 753	92 825
Heap leach pads	**24 200**	2 663	12 497	9 040
Heap leach stacking and reclaiming	**25 592**	0	11 028	14 564
Power	**20 020**	934	11 829	7 257
Roads	**9 658**	6 843	1 770	1 045
Water management	**5 824**	0	971	4 854
General & Administration	**4 061**	385	1 183	2 493
Resettlement action plan	**3 885**	647	3 237	0
Total initial capex	**281 948**	**12 185**	**94 926**	**174 837**

Sustaining capital is required thereafter to maintain production levels at the target throughout the LOM, including equipment purchases and replacement, and expansion of facilities such as the HLF, waste and spent ore dumps. This totals USD101 million, as shown in Table 21-34.

Table 21-34: Sustaining capital

Sustaining Capital [USD '000]	Total	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
Mining equipment	**93 185**	39 737	0	232	1 189	286	718	39 864	9 341	1 796	22	0
Mining infrastructure	**0**	0	0	0	0	0	0	0	0	0	0	0
Processing plant	**0**	0	0	0	0	0	0	0	0	0	0	0
Heap leach pads	**6 270**	0	0	0	0	0	0	6 270	0	0	0	0
Heap leach stacking and reclaiming	**0**	0	0	0	0	0	0	0	0	0	0	0
Power	**0**	0	0	0	0	0	0	0	0	0	0	0
Roads	**1 209**	0	0	0	0	385	0	0	0	823	0	0
Water management	**8**	0	0	0	0	0	0	0	0	0	0	8
General & Administration	**0**	0	0	0	0	0	0	0	0	0	0	0
RAP	**0**	0	0	0	0	0	0	0	0	0	0	0
Total sustaining Capex	**100 671**	**39 737**	**0**	**232**	**1 189**	**671**	**718**	**46 134**	**9 341**	**2 620**	**22**	**8**

Total LOM capital is USD 383 million.

21.13.2. Operating costs

The LOM Opex is shown in Table 21-35 on a unit cost per ROM tonne, unit cost per pound U_3O_8 and percentage basis. LOM total opex is USD 816 million, with mining and processing costs making up 94 % of the operating costs.

Table 21-35: Operating cost summary

Opex [USD '000]	USD/ ROM t	USD/ lb U_3O_8	% of Total
Mining	9.55	14.94	46.4
Mining infrastructure	0.19	0.29	0.9
Processing	8.37	13.09	40.7
Stacking	0.85	1.34	4.2
Reclaiming	0.35	0.55	1.7
G&A	0.42	0.66	2.1
Power rebate	-0.13	-0.20	-0.6
Product transport	0.93	1.46	4.5
Closure	0.05	0.07	0.2
Total Opex	**20.58**	**32.20**	**100**

22. Economic analysis

22.1. Assumptions

The economic analysis was conducted by building a discounted cash flow ("DCF") model for the project, using the financial assumptions detailed in Table 22-1 and the production, capital expenditure, operating cost and project implementation schedule discussed in this report. The model is built in real terms, based on January 1, 2025 US dollars.

Table 22-1: Financial assumptions applied in valuation

Parameter	Units	Value	Comment
Uranium price	USD /lb U_3O_8	90	
Corporate income tax rate	%	30	Percent of taxable income, sourced from Zambian tax legislation
Government royalties	%	5	Percent of revenue, sourced from Zambian mining legislation
Discount rate	% p.a.	8	See derivation below
Valuation base date	Date	January 1, 2025	
Tax depreciation rate	Years	5	Sourced from Zambian tax legislation
Capital expenditure contingency	%	10	Percent of initial capital expenditure

The approach applied to derive an appropriate discount rate to apply in the valuation was through an assessment of the three key elements of the interest rate (cost of debt), the project risk based on the stage of its development and the country risk; and in the second, on a physical basis combining a risk-free mining factor (between 1 % and 5 %), the assessed geological risk (between 1 % and 8 %), the assessed operating risk (between 1 % and 8 %), and country risk(between 1 % and 8 %).

Assessing the stage of the Muntanga project development in Zambia and assessing the institutional consensus indicates a risk-free mining factor at date of valuation (January 1, 2025) of 4.0 % (10 yr US T-Bill), added to a geological risk of 1 %, operating risk of 1 % and the median country risk of 2 %, for a total of 8 %.

This indicates that a reasonable base discount rate of at least 8 % is appropriate for application in this case. Further confidence is given by Zambia having shown commitment to the mining industry and critical minerals development as core to rebirthing its economy. This is underscored by numerous strategic agreements between Tier 1 companies like First Quantum, Barrick, and International Resource Holdings, that has mitigated strongly any underlaying uncertainty and added risk for mining investment in Zambia.

The ore tonnages, grades, process recoveries and U_3O_8 production for Muntanga, Dibbwi East and the total Project are presented in Table 22-2.

Table 22-2: Mining and production summary

Item	Units	Project total	Muntanga	Dibbwi East
Mining				
Mining period	Years	**12.0**	4.5	12.0
ROM ore production period	Years	**12.0**	4.5	11.3
Total material mined	Mt	**183.8**	18.5	165.3
ROM ore	Mt	**39.6**	8.4	31.2
ROM grade	ppm U_3O_8	**320**	331	317
U_3O_8 content in ore	M lb	**28.0**	6.1	21.9
Processing				
Plant production period	Years	**11.8**	4.6	11.4
Plant feed	Mt	**39.6**	8.4	31.2
U_3O_8 recovery	% of feed content	**90.5**	93.0	89.8
U_3O_8 produced	M lb	**25.3**	5.7	19.6

Figure 22-1 shows the mining production schedule. There is a small amount stockpiling in the first year of mining.

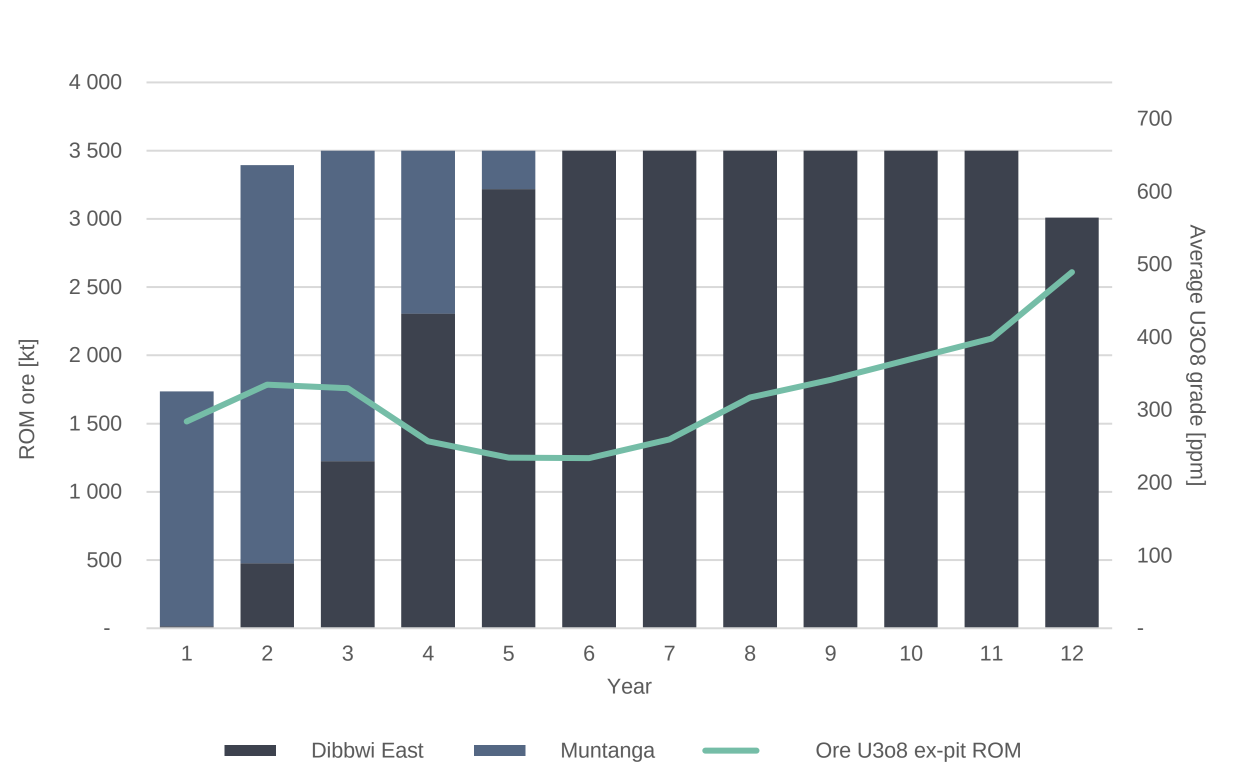


Figure 22-1: ROM tonnage and average U_3O_8 grade from Muntanga and Dibbwi East over the LOM

22.2. Revenue

Applying the recoveries for the different ore types shown in Table 17-20 to plant feed tonnages and grades gives the LOM U_3O_8 production profile in Figure 22-2. The total production is 25.3 Mlb. Note the steady increase in annual revenue from 2028 – 2034 as grades increase at Dibbwi East.

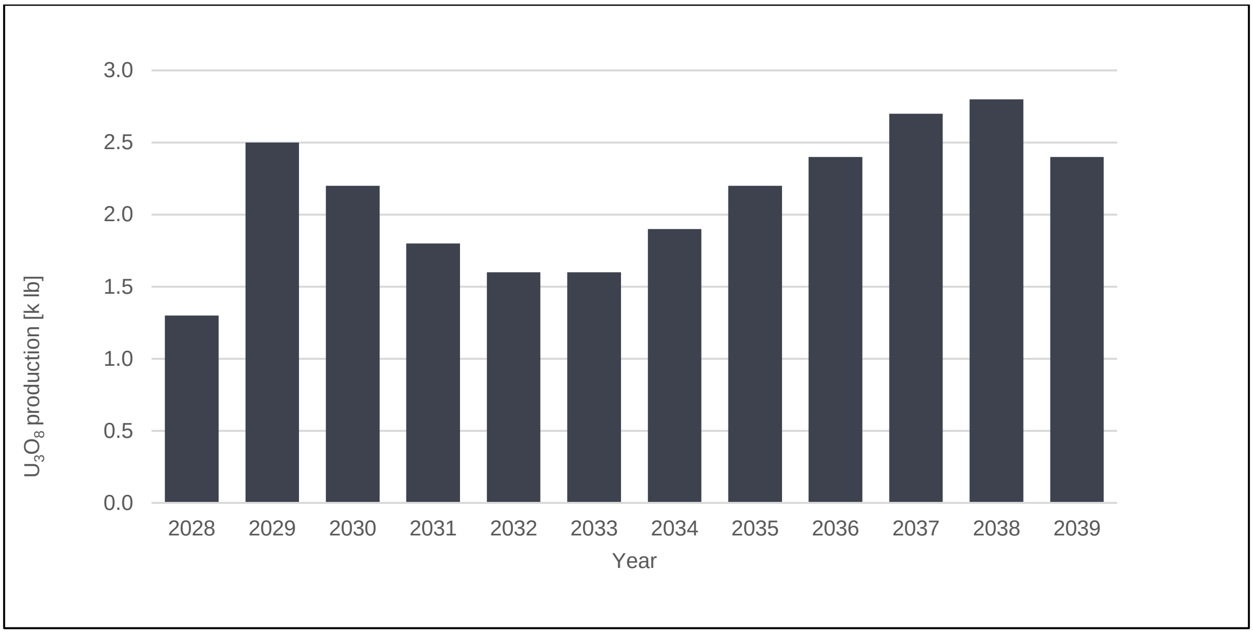


Figure 22-2: LOM U_3O_8 production

At a base price of USD 90 /lb U_3O_8, the resulting LOM revenue is shown in Figure 22-3, totalling USD 2 280 million.

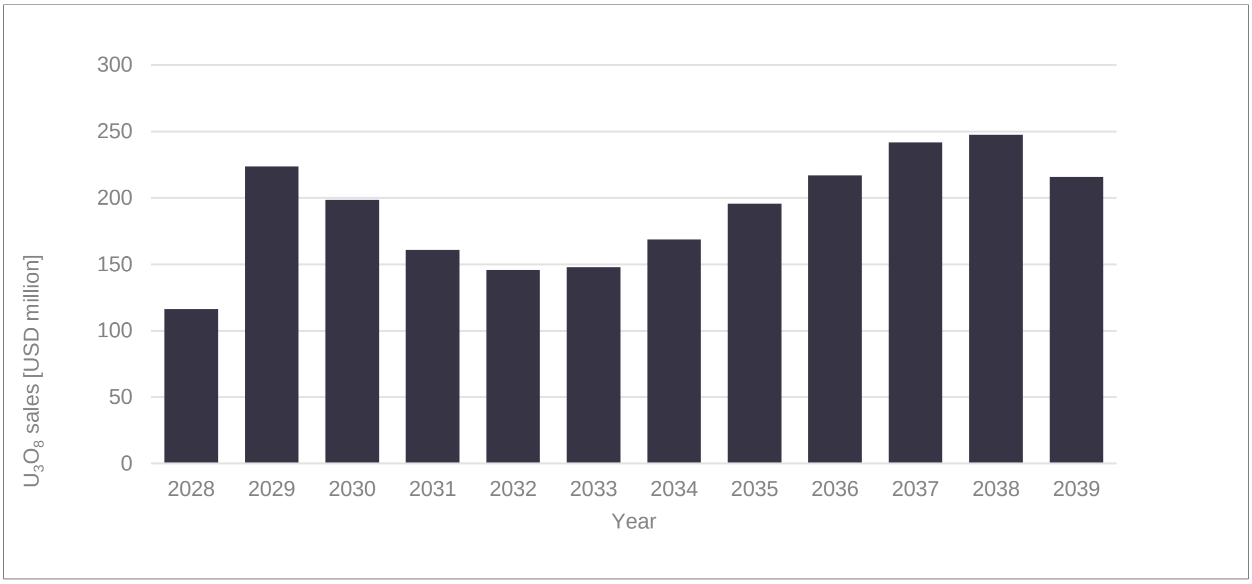


Figure 22-3: LOM revenue

22.3. Cash flow forecasts and financial performance

The LOM cashflow for Muntanga is shown in Figure 22-4**Error! Reference source not found.**. The Project returns USD 672 million in free cash flow, resulting in an NPV (at 8 %) of USD 243 million. On a cash basis, the Project pays back by October 2031, within 3.5 years of first revenue.

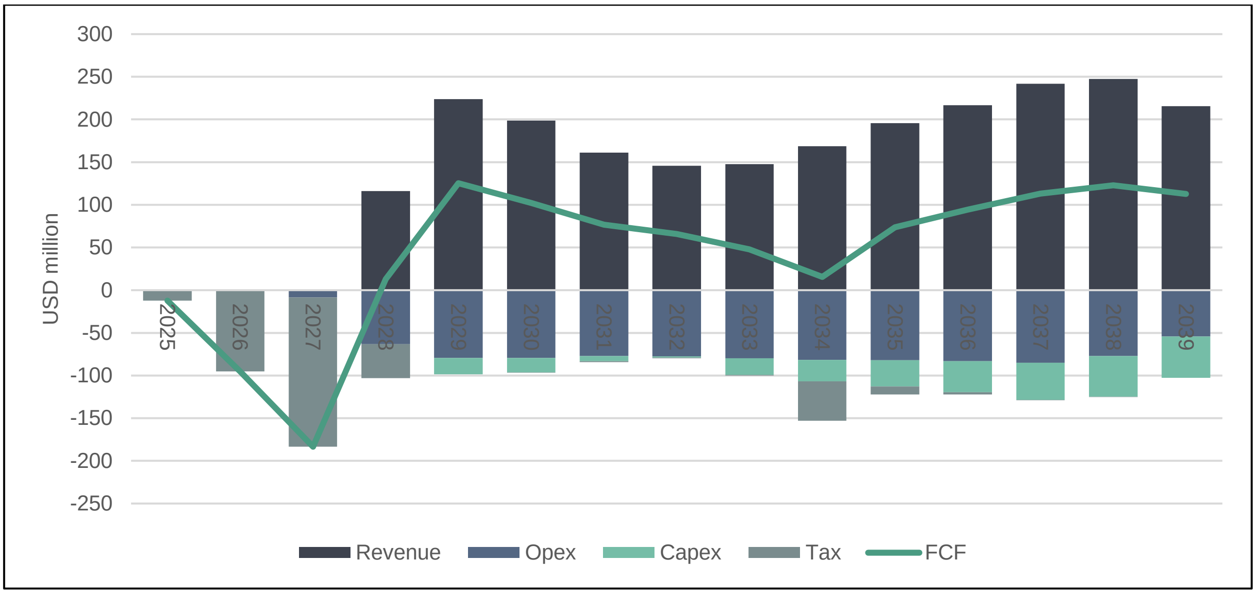


Figure 22-4: LOM cash flows

Table 22-3: LOM cash flow

Year	Units	LOM Total	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039
Physical production																	
Waste	Mt	144.2	0.0	0.0	1.7	13.9	13.9	13.9	13.7	13.4	13.8	13.9	13.3	13.2	12.2	5.8	1.6
ROM Ore	Mt	39.6	0.0	0.0	0.1	2.4	3.4	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	2.1
Plant feed	Mt	39.6	0.0	0.0	0.0	2.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	2.1
Grade	ppm U_3O_8	**320.3**	0.0	0.0	0.0	298.9	356.4	306.5	250.1	228.0	244.0	279.9	303.0	352.5	395.2	406.9	499.4
U_3O_8 produced	M lb	**25.3**	0.0	0.0	0.0	1.3	2.5	2.2	1.8	1.6	1.6	1.9	2.2	2.4	2.7	2.8	2.4
Sales																	
U_3O_8 sales	M lb	25.3	0.0	0.0	0.0	1.3	2.5	2.2	1.8	1.6	1.6	1.9	2.2	2.4	2.7	2.8	2.4
Price	USD/ lb U_3O_8	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0
Revenue	**USD million**	**2 279.8**	**0.0**	**0.0**	**0.0**	**116.2**	**223.8**	**198.7**	**161.0**	**145.7**	**147.8**	**168.8**	**195.8**	**216.9**	**241.8**	**247.6**	**215.7**
Operating costs																	
Mining	USD million	378.5	0.0	0.0	7.9	34.2	35.2	34.9	33.4	32.8	33.4	34.2	33.0	31.6	32.0	23.1	12.7
Processing*	USD million	379.4	0.0	0.0	0.0	20.1	28.1	30.1	31.7	33.9	35.2	35.0	34.7	35.7	35.8	35.8	23.3
Other costs	USD million	57.9	0.1	0.4	0.7	3.3	5.0	4.6	4.0	3.8	3.8	4.1	4.6	4.9	5.3	5.9	7.2
Royalties	USD million	114.2	0.0	0.0	0.0	5.8	11.2	10.0	8.1	7.3	7.4	8.5	9.8	10.9	12.1	12.4	10.8
Total operating costs	USD million	**930.0**	**0.1**	**0.4**	**8.6**	**63.4**	**79.5**	**79.6**	**77.3**	**77.7**	**79.9**	**81.7**	**82.1**	**83.1**	**85.2**	**77.2**	**54.1**
Corporate income tax																	
Tax	USD million	**294.8**	0.0	0.0	0.0	0.0	18.9	16.7	6.0	1.4	19.3	25.2	30.8	36.6	43.4	47.6	48.8
Capital expenditure																	
Mining	USD million	144.2	0.6	7.7	42.8	39.7	0.0	0.2	1.2	0.3	0.7	39.9	9.3	1.8	0.0	0.0	0.0
Processing	USD million	193.8	2.8	68.3	116.4	0.0	0.0	0.0	0.0	0.0	0.0	6.3	0.0	0.0	0.0	0.0	0.0
Infrastructure	USD million	36.7	7.8	14.6	13.2	0.0	0.0	0.0	0.0	0.4	0.0	0.0	0.0	0.8	0.0	0.0	0.0
G&A	USD million	4.1	0.4	1.2	2.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
RAP	USD million	3.9	0.6	3.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Total capital expenditure	USD million	**382.6**	**12.2**	**94.9**	**174.8**	**39.7**	**0.0**	**0.2**	**1.2**	**0.7**	**0.7**	**46.1**	**9.3**	**2.6**	**0.0**	**0.0**	**0.0**
Free cash flow																	
Free cash flow	**USD million**	**672.4**	**-12.3**	**-95.3**	**-183.5**	**13.0**	**125.3**	**102.2**	**76.6**	**65.9**	**47.9**	**15.7**	**73.6**	**94.5**	**113.2**	**122.7**	**112.9**

* Processing costs include HLF stacking and reclaiming

A summary of Project financial performance is shown in Table 22-4, providing LOM and unit cost analysis.

Table 22-4: Financial performance summary

Item	LOM [USD million]	Product unit [USD/ lb U_3O_8]	ROM unit [USD/t ROM]
Revenue			
U_3O_8 Revenue	**2 279.8**	**90.00**	**57.51**
Operating costs			
Mining	378.5	14.94	9.55
Processing*	379.4	14.98	9.57
Other costs	57.9	2.29	1.46
Royalties	114.2	4.51	2.88
Total operating costs	**930.0**	**36.71**	**23.46**
Corporate income tax			
Tax	294.8	11.64	7.44
Capital expenditure			
Mining	144.2	5.69	3.64
Processing	193.8	7.65	4.89
Infrastructure	36.7	1.45	0.93
G&A	4.1	0.16	0.10
RAP	3.9	0.15	0.10
Total capital expenditure	**382.6**	**15.10**	**9.65**
Financial performance			
Free cash flow	**672.4**	**26.55**	**16.96**
Net present value @ 8 %	**242.6**		
Internal rate of return	**20.8%**		

* Processing costs include HLF stacking and reclaiming

22.4. Valuation sensitivity

Table 22-5, Table 22-6, Figure 22-5, and Figure 22-6 show the sensitivity of Project NPV and IRR to changes in the U_3O_8 price, opex and capex. As expected, the Project is most sensitive to changes in the price, followed by opex and capex. The NPV breakeven price is USD 63.32 /lb U_3O_8.

Table 22-5: NPV sensitivity to changes in U_3O_8 price, Capex and Opex

Change in variable	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%
U_3O_8 price	81	122	162	202	**243**	283	323	363	403
Opex	306	290	274	259	**243**	227	211	195	179
Capex	291	279	267	255	**243**	230	218	206	194

Table 22-6: IRR sensitivity to changes in U_3O_8 price, Capex and Opex

Change in variable	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%
U_3O_8 price	12.7 %	14.8 %	16.9 %	18.9 %	**20.8 %**	22.6 %	24.4 %	26.1 %	27.8 %
Opex	23.8 %	23.1 %	22.3 %	21.5 %	**20.8 %**	20.0 %	19.2 %	18.4 %	17.6 %
Capex	25.9 %	24.4 %	23.1 %	21.9 %	**20.8 %**	19.7 %	18.7 %	17.8 %	17.0 %

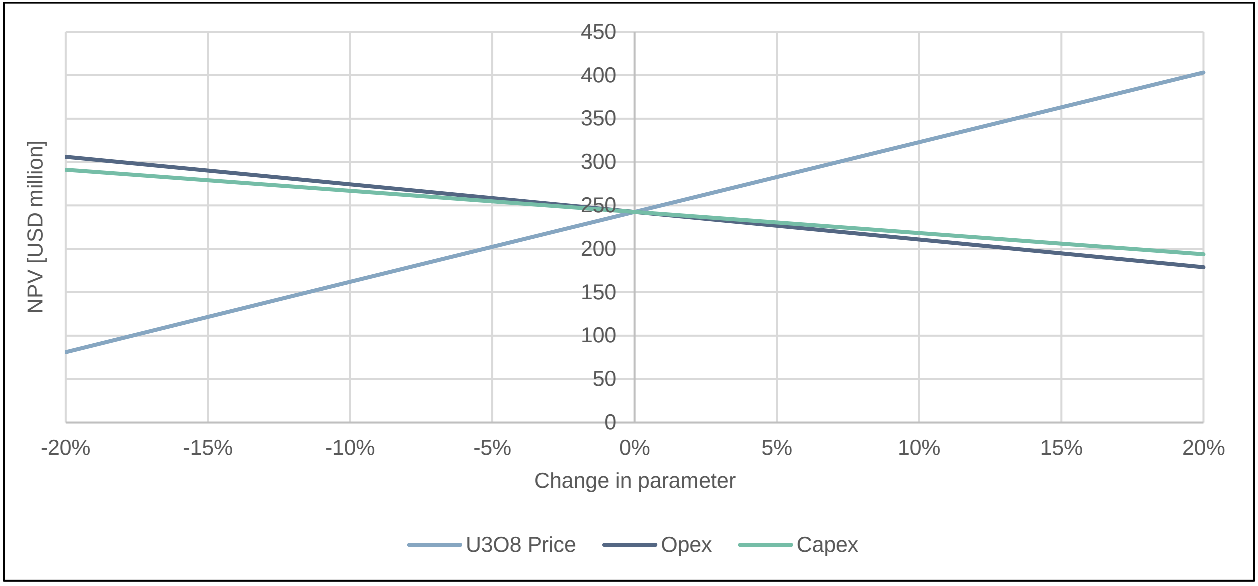


Figure 22-5: NPV sensitivity to changes in U_3O_8 price, opex and capex

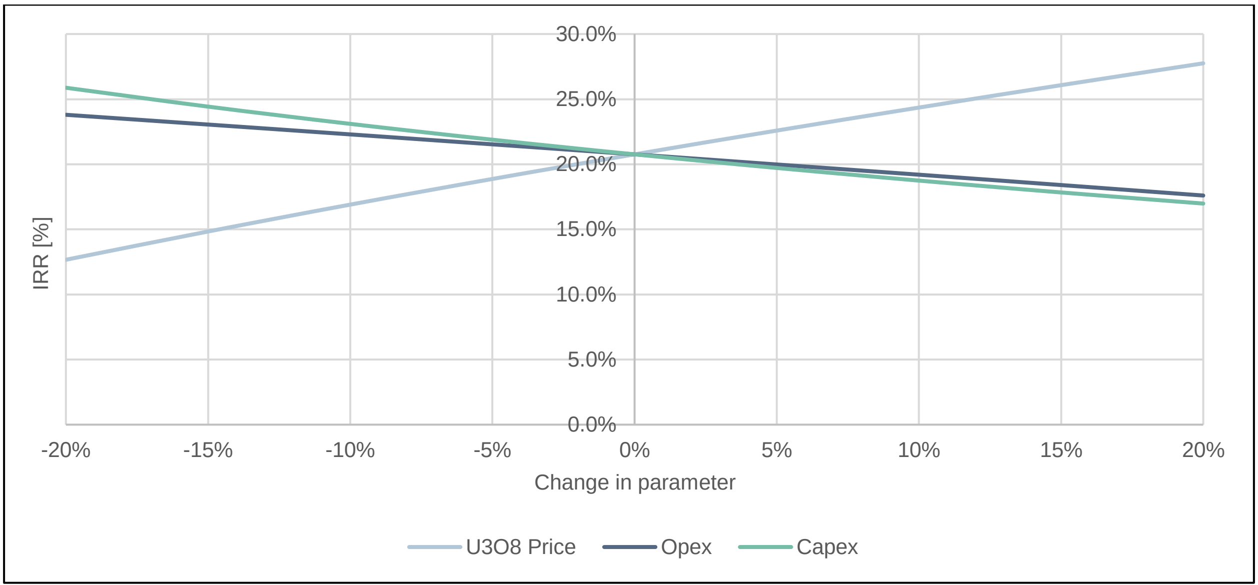


Figure 22-6: IRR sensitivity to changes in U_3O_8 price, opex and capex

22.5. Conclusion

An economic assessment to verify and demonstrate the economic viability of the Mineral Reserves was undertaken. Mineral Reserves declared at a price of USD 90/lb U_3O_8 return a positive NPV of USD 243 million at a discount rate of 8 %, with an IRR of 20.8 %.

23. Adjacent properties

There are no mining properties immediately adjacent to the Muntanga Uranium Project licences.

24. Other relevant data and information

24.1. The satellite pits: Dibbwi, Gwabi and Njame

24.1.1. Introduction

The scope of work for this FS included five pits: the central pits, Muntanga and Dibbwi East; and the satellite pits, Dibbwi, Gwabi and Njame. The FS technical work was done to the same level for all five pits, including mine schedules, heap leach and processing plants, waste rock and spent ore dumps, infrastructure, power supply, water management, access roads, environmental and resettlement plans.

The first mining schedule developed for the Project, based on the approach in the PEA, targeted roughly 3.9 Mtpa of ROM, with 3.4 Mtpa from the central pits and 0.5 Mtpa from the satellite pits, as shown in Figure 24-1. Two plants were planned – a fixed 3.4 Mtpa plant at the central site, and a moveable 0.5 Mtpa plant initially located at Gwabi. Mining started simultaneously at Muntanga, Dibbwi East and Gwabi. The mobile plant would replicate the central plant on a smaller scale, processing ore all the way through to IX resin, which would be shipped to the Central plant for elution and yellow cake production. As Gwabi was reaching the end of its life, mining would start at Njame, and the plant was to be moved to Njame. Similarly, as Njame was reaching the end of its life, mining would start at Dibbwi, and the plant was to be moved to Dibbwi.

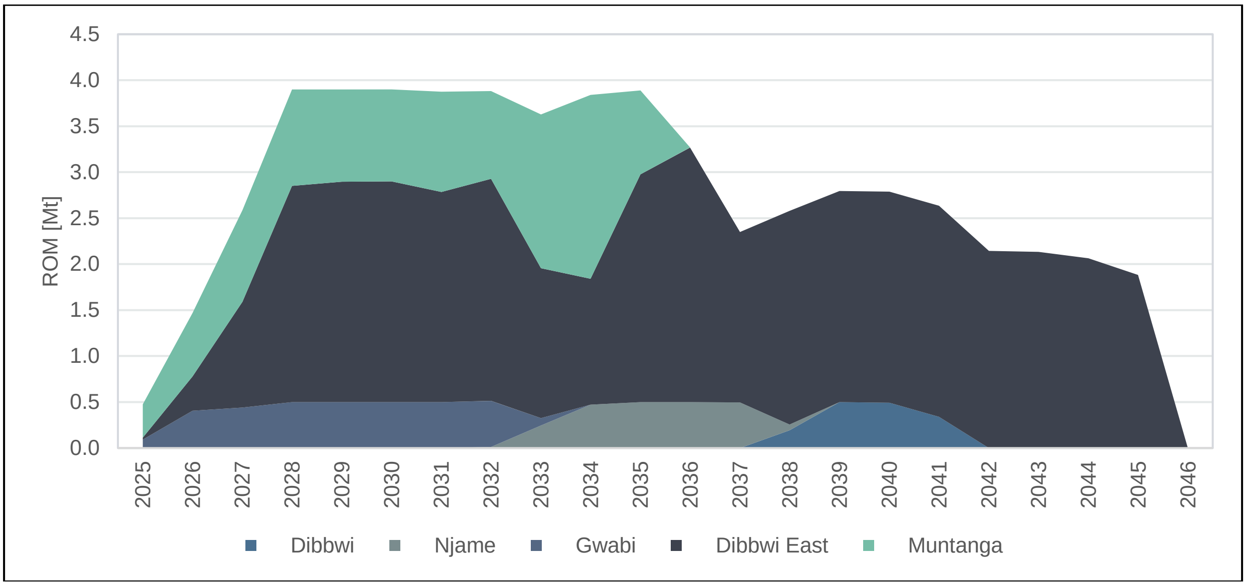


Figure 24-1: Initial consolidated schedule based on PEA strategy

This approach was unsatisfactory for a number of reasons:

1. The feed to the Central plant was irregular, and once Muntanga was depleted after the tenth year of mining, ROM production at Central dropped to 2.0 Mtpa – 2.5 Mtpa. The Central plant was underutilised for the remaining nine years of the LOM.
2. The Capex and Opex of the smaller moveable plant at the satellites were disproportionately high relative to the Central plant, adversely affecting the economic viability of mining the satellite pits.
3. The moveable plant would take at least three months to move from site to site and bring to a fully-operational state, leaving a production gap and incurring high moving costs

A consolidated schedule shown in Figure 24-2 was prepared based on the outcomes of a technical investigation into alternative deployment strategies.

This schedule had the following characteristics:

1. Muntanga was scheduled as quickly as possible due to a low stripping ratio and above average grades, to reduce costs and accelerate revenue
2. After Muntanga was depleted, Dibbwi East would be mined at a target 3.2 Mtpa
3. The satellites would be mined at 0.5 Mtpa
4. There will be no process plant or HLF at the satellites

5. The satellites would be equipped with a radiometric ore sorting system, “Rados”, which would reject any ore with a grade of less than 90 ppm U_3O_8 and reduce the volume of plant feed to 0.35 Mtpa while increasing the feed grade
6. The sorted ore would be trucked with a road-capable side-tipper fleet to the Central plant, where it would be fed into the second crushing stage. The Rados reject will be placed on the WRD
7. Dibbwi would be mined first, given its proximity to Central and hence reduced trucking costs, followed by Gwabi and then Njame.

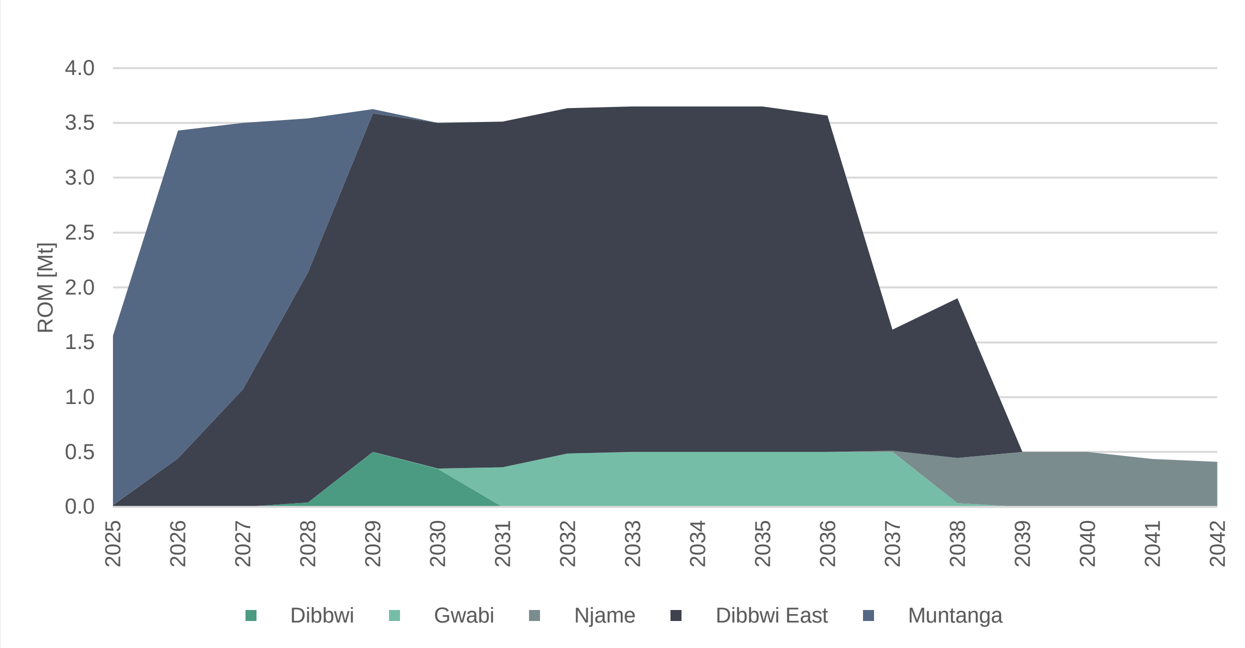


Figure 24-2: The final consolidated mining schedule including satellites

Although this schedule showed improved economics at a positive NPV, it was clear that the satellites were sufficiently value additive. The small scale, higher capital requirements and operating costs, and schedule tail of the satellite operations relative to Central detracted from the financial performance of the overall project. The decision was made to investigate a Central-only schedule, as described in Section 16 and shown in Section 22. Consequently, the satellite pits were not included as part of the Mineral Reserve estimate.

Under improved market conditions, the satellite pits have the potential to be economically attractive. The FS work was sufficiently developed to allow for implementation based on minimal additional technical effort. The findings as part of the FS on the satellite pits are discussed in the remainder of this section.

24.1.2.Mining

Using the same approach and economic factors as described in Section 16, pit designs were completed for the satellite pits. Various technical aspects were considered in the mine design, including the techno-economic pit limits, geotechnical parameters, mining methodology, mining sequence, pit access, ramp placement, equipment capability, production rates and practical mining considerations. The mining-related modifying factors were applied. Table 24-1 shows the estimated ROM schedule based on mineralised material from Measured Mineral Resources (43 %) and Indicated Mineral Resources (57 %) classes. No mineralised material from Inferred Resources were included in the satellite schedule.

The result of these schedules is shown in Table 24-1. A total of 6.5 Mt of ROM ore at an average grade of 300 ppm U_3O_8 was scheduled.

Table 24-1: Satellite pit mining summary

Mining parameter	Units	Gwabi	Njame	Dibbwi	Total
Waste	Mt	6.2	11.2	1.0	18.4
ROM	Mt	3.4	2.3	0.9	6.5
Total mined	**Mt**	**9.6**	**13.5**	**1.9**	**25.0**
Stripping ratio	t:t	1.8	4.9	1.1	2.8
ROM ore grade	ppm U_3O_8	322	300	220	300
Contained U_3O_8	Mlb	2.4	1.5	0.4	4.3
Mining rate	Mtpa	0.5	0.5	0.5	0.5
Mining duration	years	7.3	5.1	1.8	13.8

Pictures of the pits and associated WRDs are shown in Figure 24-3 to Figure 24-5.

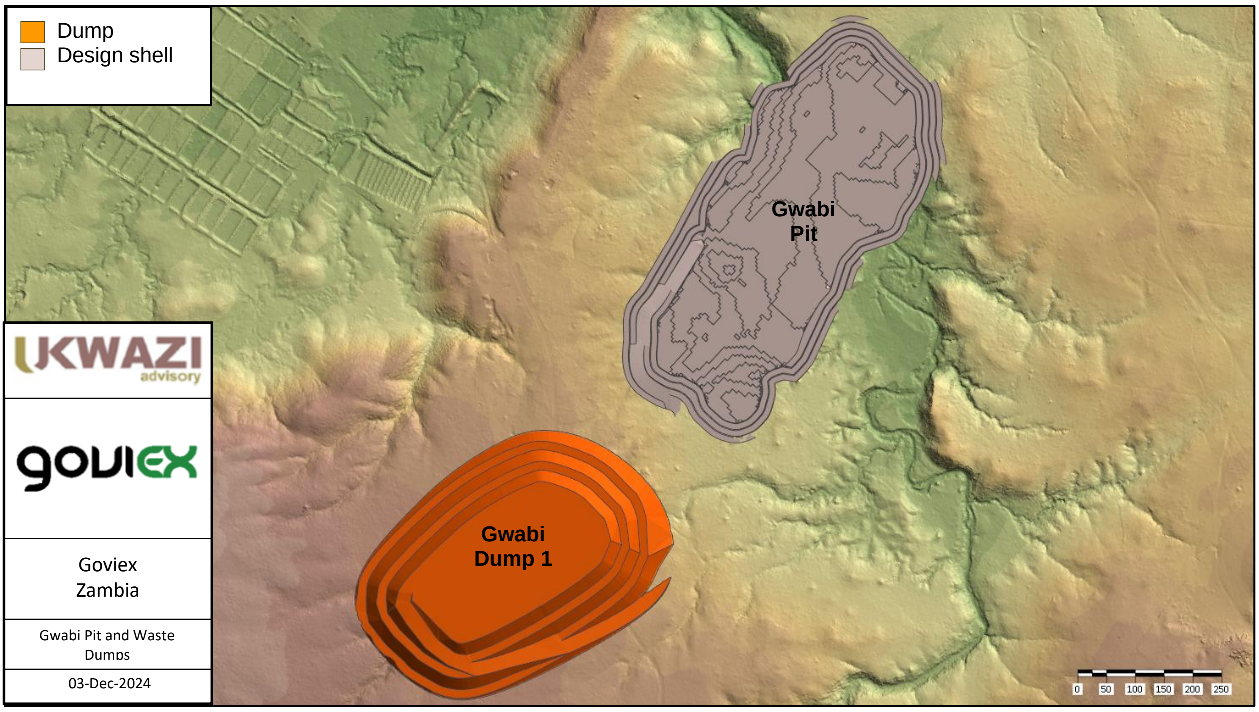


Figure 24-3: Gwabi pit and WRD

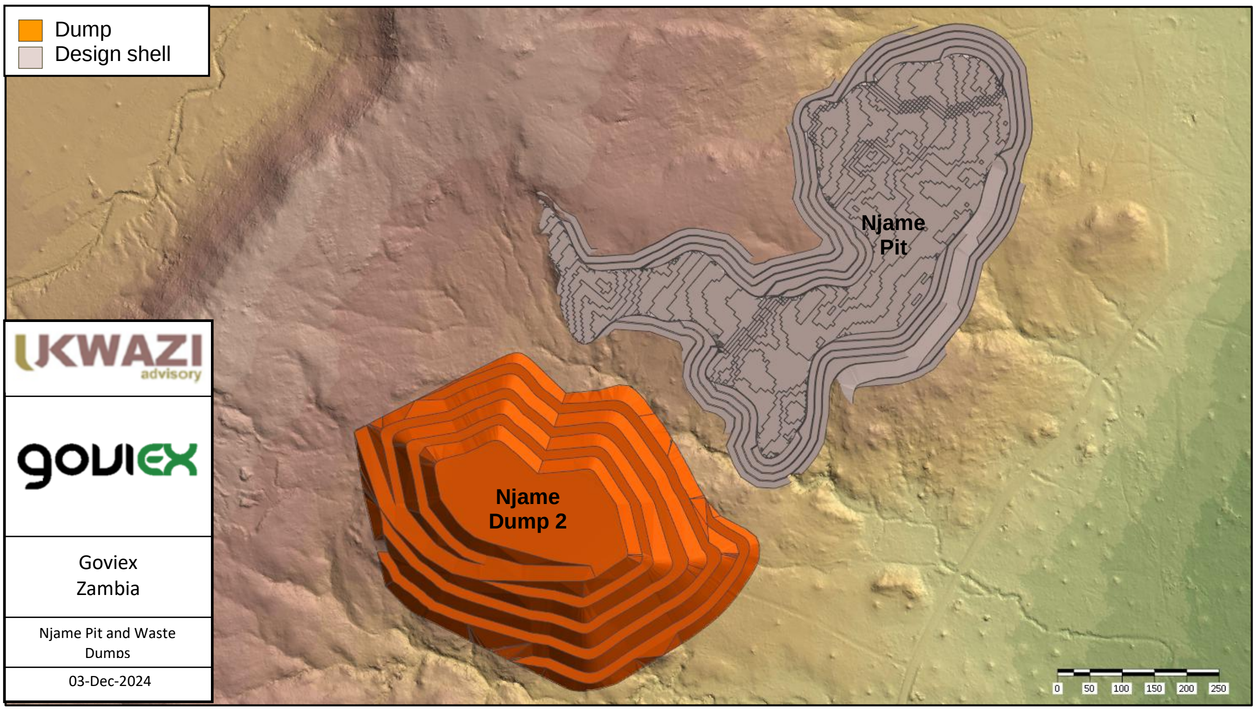


Figure 24-4: Njame pit and WRD

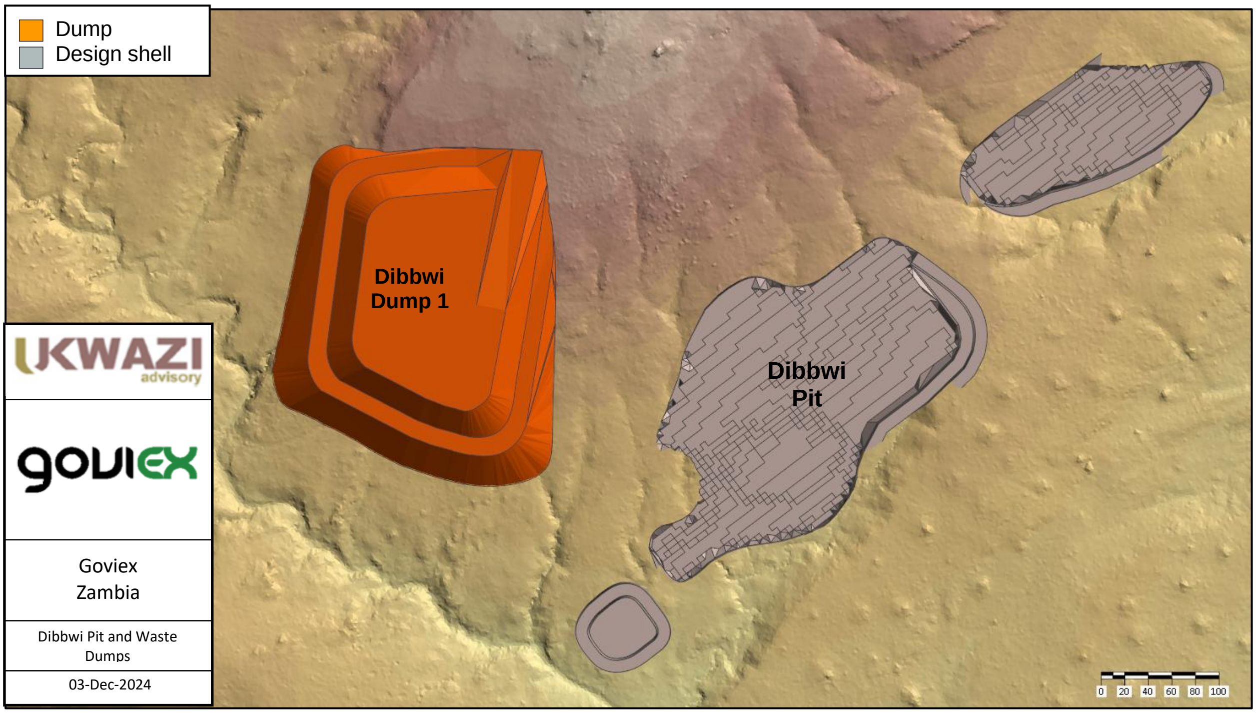


Figure 24-5: Dibbwi pit and WRD

The mining fleet requirement for the satellites is shown in Table 24-2. The unit numbers are displayed as a maximum, based on the consolidated schedule that allows for effective capital expenditure allocations and the movement of equipment between operations.

Table 24-2: Maximum mining equipment requirements for satellite pits

Equipment	Units
Cat 395 BH	2
Cat 745 ADT	6
Cat D8 Dozer	**3**
Cat 140 Grader	2
Cat 730 Water Bowser	2
Sandvik DI650i Drill Rig	1
Waterpump	6
Lighting plant	6
Cat 730 Diesel Bowser	1
Cat 330 Rockbreaker	1
Cat 428 TLB	1
LDVs	15
Minibus	1
Buses	2

24.1.3. Rados ore sorting

At the satellite pits, the ore will be sorted at the site to improve the grade and reduce the ore trucked to the Central plant. The Rados system is a radiometric sorting system, which sorts ore according to uranium content, passing ore above a certain cut-off grade and rejecting ore below. Test results show a 30 % rejection of the material sorted, while retaining 95 % of the U_3O_8 content. The high-grade ore will be trucked by a dedicated road-going fleet to the plant, and the low-grade discard will be deposited on the allocated WRD. The ore sorting system is shown in Figure 24-6.

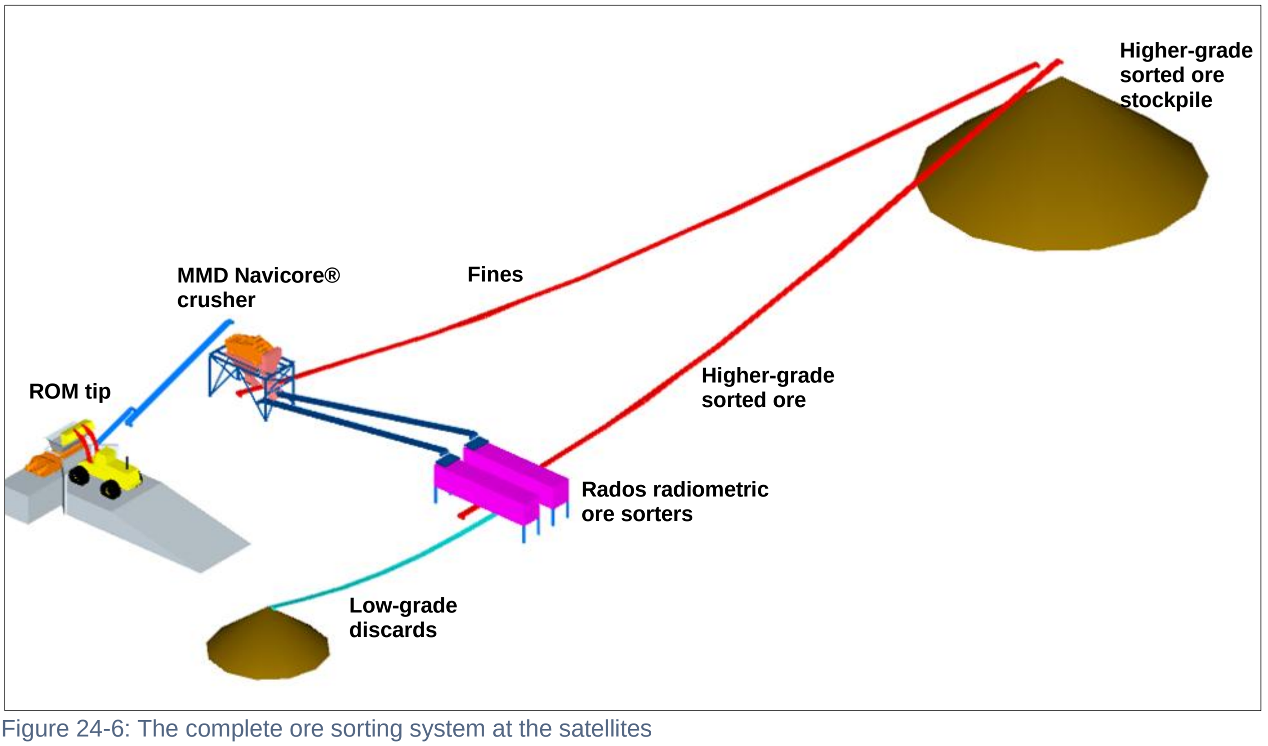


Figure 24-6: The complete ore sorting system at the satellites

Ore mined from the satellite pits will be hauled to the Rados facilty using the mining haulage fleet.

A FEL will discharge ROM material into the MMD Navicore® crusher unit that will reduce the material from 500 mm to 150 mm at a single pass, at a rate of up to 100 tph.

From the NaviCore, a transfer conveyor will be employed to elevate the material to the screen module. This will allow the material to be spread over the width of the screen to increase the screen efficiency. The conveyors will be modular in design and easily relocatable within the mining area, alternatively if extended relocation is required, minimal disassembly will be targeted to facilitate the use of trucks to the new operating location. This material will be transferred via a 1 050 mm conveyor belt to a screen module, which will segregate the material for sorting into the following factions:

- Coarse: 150 mm to 40 mm
- Intermediate: 40 mm to 10 mm
- Fines: <10 mm.

The coarse material will report to a modular Rados Ultima Unit for final sorting, and Intermediate material will report to the Rados Optima unit for sorting. While all the fines will be stockpiled separately.

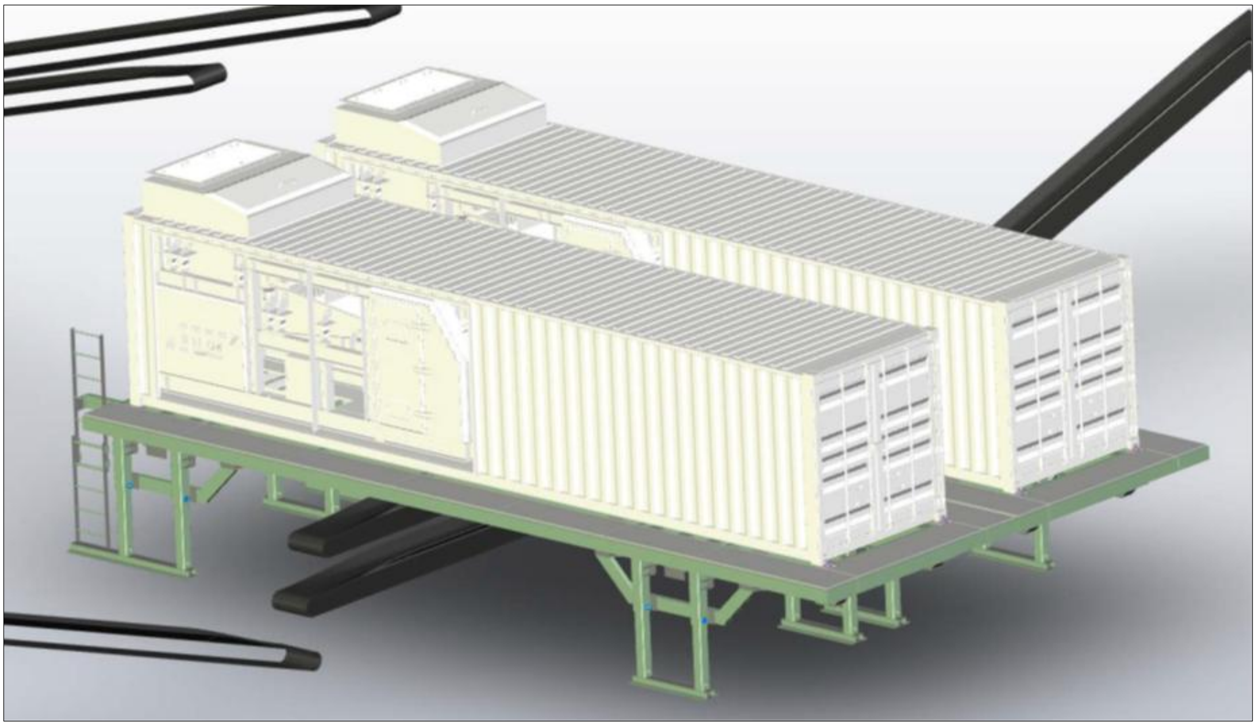


Figure 24-7: Conceptual modular Rados sorter units

The results of the Rados ore sorting are shown in Table 24-3. Trucking volumes are reduced by 2 Mt in total (30 %), while the Satellite plant feed ore grade to Central is increased to 408 ppm U_3O_8 from 308 ppm U_3O_8 (32 %).

Table 24-3: Rados ore sorting recoveries

Ore Sorting	Units	Gwabi	Njame	Dibbwi	Total
ROM to Rados					
ROM ore volume	Mt	3.4	2.3	0.9	**6.5**
ROM ore grade	ppm U_3O_8	322	300	220	**308**
Contained U_3O_8	Mlb	2.4	1.5	0.4	**4.3**
Rados sorted ore					
High-grade sorted ore volume	Mt	2.4	1.6	0.6	**4.6**
Sorted ore grade	ppm U_3O_8	436	408	298	**408**
Contained U_3O_8	Mlb	2.3	1.4	0.4	**4.1**
Rados reject					
Reject low-grade ore to WRD	Mt	1.0	0.7	0.3	**2.0**
Reject grade	ppm U_3O_8	54	50	37	**50**

24.1.4.High-grade ore transport

The high-grade Rados product will be loaded by Komatsu WA500 FELs onto Volvo FH440 trucks equipped with side-tipping trailers and transported to the Central plant, where it will be placed onto a tip at the secondary crushing stage. Dibbwi requires three sidetipper trucks, Gwabi nine and Njame seven.

24.1.5.Ore processing

As it has already been through a primary crushing stage, satellite ore will enter the Central plant at the secondary crushing stage (Section 17.4.2.3). After secondary crushing, it will be treated in the same way as ore from the Central pits (Sections 17.2 to 17.4).

A summary of key parameters informing the overall plant design is provided in Table 24-4. Note that the recoveries vary from ore body to ore body as shown in Table 17-20.

Table 24-4: Summary of key parameters

Parameter	Unit	Value
Ore source		
Dibbwi, Gwabi and Njame pits	Mpa (max)	0.350
Ore grade		
Dibbwi	g/t U_30_8	308
Gwabi	g/t U_30_8	434
Njame	g/t U_30_8	406
Rados ore sorting		
Uranium recovery	%	95.0
Mass pull	%	70.0
Uranium recovery (overall)		
Dibbwi	%	92.2
Gwabi	%	73.1
Njame	%	93.0

24.1.6.Infrastructure

In order to access the Gwabi (and Njame) sites, an upgrade of the existing road infrastructure is required, and a primary access road is to be constructed. A power supply will need to be developed, and water management infrastructure put in place. Mining infrastructure at each of the satellite pits was kept to a minimum, restricted to WRDs, location for the Rados system, and offices, refuelling stations and change houses for mining personnel.

24.1.6.1. Mine infrastructure

The satellite mining sites were designed to supplement the Central mining operation and were designed to rely on the Central for support, avoiding unnecessary duplication of costly infrastructure. The infrastructure at these sites was designed to be portable and simple to allow for efficient relocation from one site to another. This modularity minimises capital expenditure and reduces the environmental footprint. The proposed infrastructure provides a starting point for future development and rapid deployment of the satellite operations if they become economically attractive.

The satellite sites have been designed with a range of essential infrastructure, including:

- Secure perimeter fencing to protect the sites
- Safe and efficient haul road networks for transportation
- A centralised infrastructure complex with offices and workshops
- Stormwater and pollution control measures, including pollution control dams
- Pit dewatering systems to manage groundwater and surface runoff
- Rapid reload areas for safe explosives transfer
- RADOS areas for sorting and stockpiling ore
- Well-marked internal access roads for easy navigation
- Designated parking areas for various vehicles
- Access control points to regulate entry into different areas
- Modular, prefabricated office and general buildings
- Workshops for engineering and maintenance activities
- Wash bays for cleaning equipment
- Tyre management areas for maintenance and storage
- Bulk fuel storage areas for fuel and lubricants
- Refuelling and service stations for daily checks and refuelling
- Water storage and reticulation systems for potable water, process water, wastewater, and firewater
- Sewage reticulation and management systems for treating and reusing sewage
- Electrical infrastructure for power supply and distribution
- Communication networks, including VHF radio and GPS tracking.

24.1.6.2. Gwabi primary access road design

The existing access road alignment was taken forward into FS design. The Gwabi access road is 7m wide and will support a haulage operation. The access road has a total length of 5+790 m long. Road design parameters are presented in Table 18-6. The total change in natural ground level over the length of the road is approximately -25 m with a maximum gradient of 7 %. At the connection with the T2, the elevation is 440 m ASL. Over 5 km, the ground level gradually descends to around 395 m ASL (chainage 5+000) before rising to 415m at the mine entrance gate. A significant drainage crossing is encountered at chainage 2+040 and a longer descent from 4+200 to 5+000. A summary of new versus upgrade is presented in Table 24-5.

Table 24-5: Gwabi road alignment

Chainage start	Chainage finish	Details
0+000	2+500	Upgrade to the existing road (widening)
2+500	3+825	Upgrade to the existing road and minor alignment changes (widening).
3+825	5+790	New alignment and road construction.

In general, the route follows the existing Gwabi access road. A school is located around 375 m from the start of the road and cannot be avoided. Traffic controls will be put in place during the years of operation as advised by the environmental and social consultants. Another school is located close to Gwabi (5 km from the Gwabi junction with the T2); however, the access route diverts to follow a new road alignment from 3 825 km, which avoids the school.

Project usage (>90 %) will be for four-axle rigid highway trucks with a 30 t to 40 t payload delivering crushed and sorted ore material from Dibbwi to Muntanga while Dibbwi is in operation. Other vehicles will be delivery trucks (similar loading) and light vehicles. Maximum truck width will be around 2.6 m. This will be a shared road and speed limits, and operating hours will be restricted in accordance with the permits.

The road geometry for Gwabi is the same as for Dibbwi. Pavement design parameters are presented in Table 18-9. The pavement design for Gwabi will align with that proposed for the Muntanga Access Road.

The access roads for Gwabi and Njame are shown in Figure 24-8.

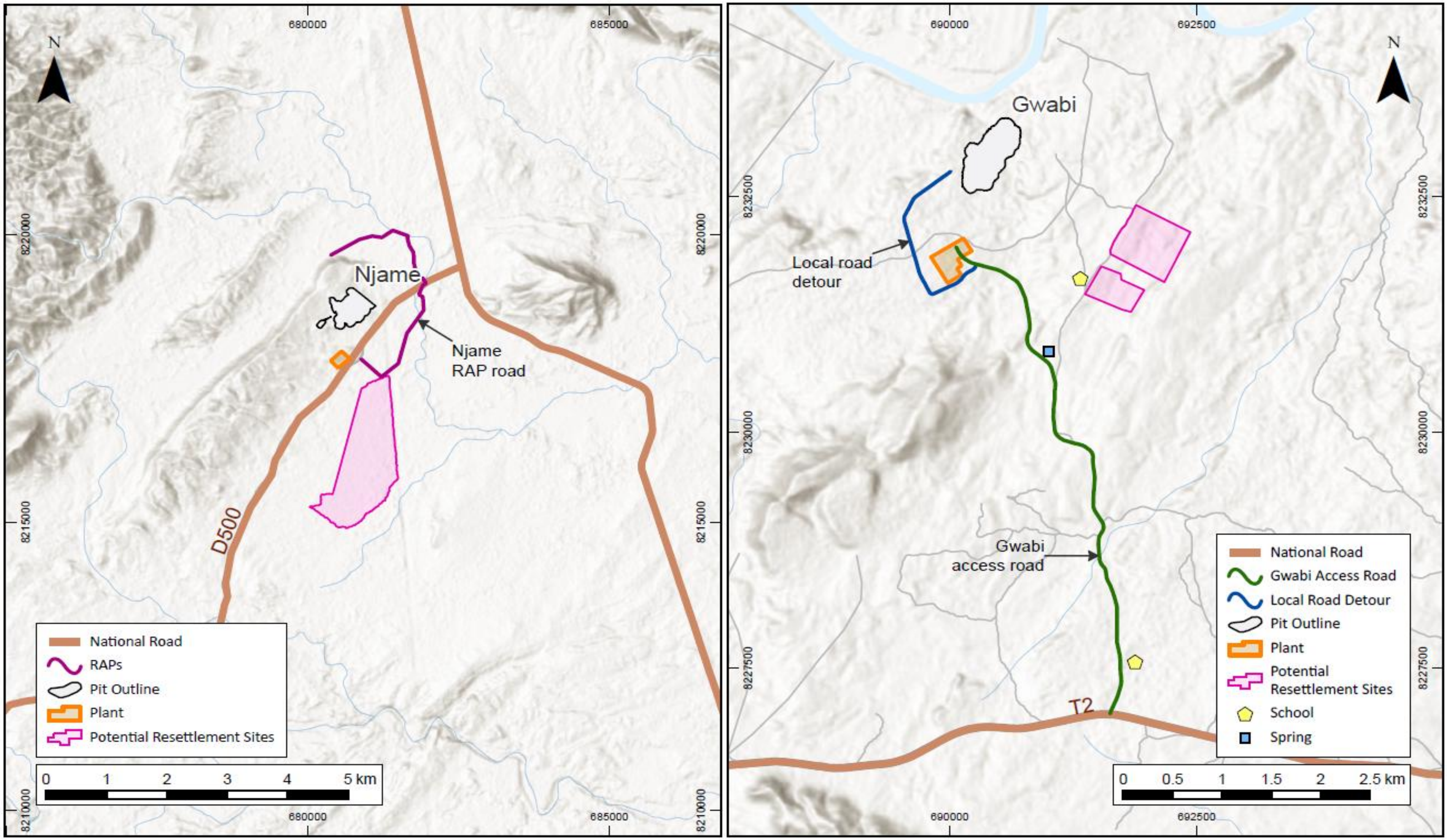


Figure 24-8: Access roads for Gwabi and Njame

Earthworks, culverts/ drainage, fencing and signage, and dust suppression will be implemented according to the same guidelines and design parameters and the other access roads as described in Section 18.1.

Table 24-6: Summary of RAP satellite roads

Item	Units	Value
Njame RAP road, Njame area		
Total length	**m**	**4 750**
Pavement surface area	m^2	14 250
Net elevation gain	m	18
Gwabi detour road, Gwabi area		
Total Length	**m**	**2 100**
Pavement surface area	m^2	6 300
Net elevation gain	m	5

24.1.6.3. Water management

The ore body of the Gwabi pit is located directly below a natural watercourse that discharges into the Kafue River. The Kafue River has a very large catchment area and therefore has a very large peak flow at the confluence with a nearby tributary, and the Kafue River can potentially back up into the proposed pit area during flood events resulting in flooding of the pit. In addition, the tributary may cause flooding of the pit due to the runoff in the river upstream of the proposed pit. Several flood remediation measures were investigated to prevent flooding of the Gwabi Mine pit. The flood remediation interventions aim to ensure that no runoff from either the Kafue River or the 'Kafue2T' tributary enters the pit footprint. This is to be mitigated by upstream attenuation dams, the construction of a bund on the downstream side of the pit and four attenuation dams to prevent the Kafue River from entering the pit.

Stormwater and groundwater systems were analysed and designed according to the same principles discussed in Section 18. Dewatering boreholes as described in Table 24-7 will be required to reduce the ingress of groundwater to the pits.

Table 24-7: Dewatering boreholes for the satellite pits

Project area	Total number of dewatering boreholes	Passive residual pit inflows [m^3/d]
Dibbwi	3	1 120
Njame	3	400
Gwabi	4	1300

24.1.6.4. Bulk power supply

The following bulk supply infrastructure is required at the satellite operations:

- For operations at Dibbwi, a 10km, 35 kV single circuit OHL will connect the satellite pit to the Muntanga 132/33/11 kV substation. This may be replaced by a rented diesel generator installation.
- For operations at Njame, a 43km, 35 kV single circuit OHL will connect the satellite pit to the Muntanga 132/35/11 kV substation
- For operations at Gwabi, a 9.8km, 11 kV single circuit OHL will connect the satellite pit to the Gota-Gota 88/11 kV substation

24.1.7.Satellite production

Table 24-8 shows a summary of the LOM production for the satellite pits. A total of 25.0 Mt of material is mined, of which 6.5 Mt is ore, at an average grade of 300 ppm. Radiometric sorting by the Rados system reduces this to 4.6 Mt at an increased grade of 408 ppm. After processing in the Central uranium processing and refining plant with an average recovery of 89.1%, the high-grade ore yields 3.4 Mt of saleable U_3O_8 product.

Table 24-8: Satellite production summary

Production parameter	Units	Gwabi	Njame	Dibbwi	Total
ROM					
Annual steady-state ROM	Mtpa	0.5	0.5	0.5	**0.5**
Waste	Mt	6.2	11.2	1.0	**18.4**
Ore	**Mt**	**3.4**	**2.3**	**0.9**	**6.5**
Total mined	Mt	9.6	13.5	1.9	**25.0**
Stripping ratio	t:t	1.8	4.9	1.1	**2.8**
ROM ore grade	**ppm U_3O_8**	**322**	**300**	**220**	**300**
Contained U_3O_8	Mlb	2.4	1.5	0.4	**4.3**
Mining duration	years	7.3	5.1	1.8	**13.8**
Rados sorted ore					
Mass pull	%	70	70	70	**70**
U_3O_8 recovery	%	90	90	90	**90**
Annual high-grade ore	Mtpa	0.35	0.35	0.35	**0.35**
High-grade sorted ore volume	**Mt**	**2.4**	**1.6**	**0.6**	**4.6**
Sorted ore grade	**ppm U_3O_8**	**436**	**408**	**298**	**408**
Contained U_3O_8	Mlb	2.3	1.4	0.4	**4.1**
Rados reject					
Reject low-grade ore to WRD	Mt	1.0	0.7	0.3	**2.0**
Reject grade	ppm U_3O_8	54	50	37	**50**
Uranium processing and refining					
U_3O_8 recovery	%	73.1	93.0	92.2	**81.9**
Saleable U_3O_8 product	**Mlb**	**1.7**	**1.3**	**0.4**	**3.4**

*Allocated on ROM feed basis
2 – Based on the subsequent schedule where equipment from Central is pulled when it becomes available.

24.1.8.Satellite Capex, Opex and revenue based on the subsequent schedule

Table 24-9 provides a summary of the Capex required to develop the satellite pits. A total of USD 60.9 million is required, which translates to USD 9.31/ROM t or USD 18.07/ lb U_3O_8 saleable product.

Table 24-9: Satellite Capex

Satellite Capex	Units	Gwabi	Njame	Dibbwi	Total	USD/t ROM	USD/ lb U_3O_8 product
Mining fleet	USD million	11.6	8.9	10.6	**31.1**	4.75	9.22
Mining infrastructure	USD million	5.2	5.4	4.3	**14.9**	2.27	4.41
Rados system*	USD million	2.6	1.7	0.7	**5.0**	0.76	1.48
Roads	USD million	1.4	1.2	1.7	**4.3**	0.65	1.26
Power	USD million	3.0	1.9	0.9	**5.7**	0.88	1.70
Total	**USD million**	**23.7**	**19.2**	**18.0**	**60.9**	**9.31**	**18.07**

Table 24-10 provides a summary of the LOM costs of operating the satellite pits. A total of USD 159 million is required, which translates to USD 24.30/t ROM or USD 47.16/ lb U_3O_8 saleable product .

Table 24-10: Satellite Opex

Satellite LOM Opex	Units	Gwabi	Njame	Dibbwi	Total	USD/t ROM	USD/ lb U_3O_8 product
Mining	USD million	51.5	41.3	11.7	**104.5**	15.97	30.99
Mining infrastructure	USD million	0.5	0.4	0.3	**1.3**	0.20	0.39
Processing (incl Rados)	USD million	20.6	12.2	5.2	**38.1**	5.82	11.29
Royalties	USD million	7.5	6.0	1.7	**15.2**	2.32	4.50
Total Opex	USD million	**80.2**	**59.9**	**18.9**	**159.0**	**24.30**	**47.16**

At the FS base price of USD 90/ lb, the saleable production volumes from Table 24-8 result in the LOM revenue of USD 303.5 million. Applying the LOM costs to the revenue gives the EBIT shown in Table 24-11.

The average EBIT margin is 28 %, with both Gwabi and Njame exceeding 30 %. Dibbwi shows a negative margin due to the high capital expenditure relative to ore volume (only 14 % of satellite ROM production). Value engineering approaches such as reducing infrastructure by operating out of Central or trucking directly to Central (as the distance to Central is far shorter than Gwabi and Njame) without implementing Rados can be explored.

Table 24-11: Satellite EBIT

Satellite EBIT	Units	Gwabi	Njame	Dibbwi	Total	USD/t ROM	USD/ lb U_3O_8 product
U_3O_8 product	M lb	1.7	1.3	0.4	**3.4**		
U_3O_8 sales price	USD/lb	90.00	90.00	90.00	**90.00**		
Revenue	USD million	**150.1**	**119.5**	**33.9**	**303.5**	**46.38**	**90.00**
Capex	*USD million*	*-23.7*	*-19.2*	*-18.0*	***-60.9***	*-9.31*	*-18.07*
Opex	*USD million*	*-80.2*	*-59.9*	*-18.9*	***-159.0***	*-24.30*	*-47.16*
Total costs	USD million	**-103.9**	**-79.1**	**-36.9**	**-220.0**	**-33.62**	**-65.23**
EBIT	USD million	**46.2**	**40.4**	**-3.1**	**83.5**	**12.76**	**24.77**
EBIT margin	%	31 %	34 %	-9 %	**28 %**	28 %	28 %

24.1.9.Environmental, social and closure

As described in Section 20, the environmental, social and closure studies and permitting are being done for all operations, and the satellite pits are included in this process. The areas hold valid mining licences, and once ESG approval processes are complete and regulatory approval is received, mining operations can begin at any point after the implementation of the relevant RAPs. The conceptual closure plan does include the satellite pits but the closure cost will need review as and when a decision is taken to exploit these resources.

24.1.10. Conclusion

The satellite pits were studied with the same FS-level of detail as the Central pits and are developed to a point where they could be implemented with minimal additional technical work. The economic analysis results above show that the financial performance of the satellite pits is positive at the USD 90/ lb FS base price for U_3O_8., but at this stage the satellite pits would detract from the financial performance of the overall project. However, at higher prices, the performance will improve – for example, at USD 100/ lb U_3O_8, revenue increases to USD 337 million. GoviEx is in a position to wait for favourable market conditions at which time the satellite operations could be built and operated.

24.2. Inferred material in the mining schedule

The mining schedule presented in Section 16.2.8.4 includes mineralised material from Inferred Resources. As the schedule stands, this material is blasted, loaded and then hauled to the WRD and discarded as waste. At Dibbwi East this mineralised material comprises 5.4 Mt at an average grade of 217 ppm, and at Muntanga 465 kt at an average grade of 283 ppm at as shown in Table 24-12, giving a total for the Project of 5.8 Mt at a grade of 222 ppm.

Please note that this mineralised material contains Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as Mineral Reserves, and there is no certainty that any value will be realised from them.

Table 24-12: Inferred Mineral Resource mineralised material mined

Mined inferred	Total [Mt]	Grade [ppm]	U_3O_8 content [Mlb]
Muntanga	0.5	283	0.3
Dibbwi East	5.4	217	2.6
Total	**5.8**	**222**	**2.9**

Ukwazi recommends further drilling and study work to investigate upgrading the Inferred Mineral Resources.

24.3. Inferred material out of the Central mining schedule

In order to investigate the potential contribution to the Project of the Inferred Mineral Resources presented in Section 14.6, the Whittle pit shell optimisation described in Section 15 was run for Muntanga and Dibbwi East using the same parameters as the Measured- and Indicated Resources-only case. This produced a bigger pit shell (the “Inferred pit shell”) which completely encloses the selected shell to develop the Mineral Reserve mine schedule.

Table 24-13 shows the resulting additional mineralised material, obtained by subtracting material from the selected pit shell (designed pit) from the Inferred pit shell. This has the potential to bring another 6.6 Mt of mineralised material at an average grade of 278 ppm into the schedule. Note that this material is in addition to the Inferred material discussed in Section 24.2. 40 % of the additional material in Table 24-13 comprises mineralised material from Indicated Mineral Resources, representing blocks that were not included in the selected pit shell.

Table 24-13: Incremental contribution of resources outside of the selected pit shell

Incremental contribution	Mineralised material [Mt]	Grade [ppm]	U_3O_8 [Mlb]	Waste [t]	Total Scheduled [Mt]	Stripping ratio [t:t]
Muntanga	2.9	254	1.6	6.5	9.4	2.2
Dibbwi East	3.6	297	2.4	17.6	21.2	4.9
Incremental mineralised material from Indicated and Inferred Mineral Resources	**6.6**	**278**	**4.0**	**24.1**	**30.6**	**3.7**

Please note that this mineralised material falls outside of the FS pit design and mining schedule, and contains Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as Mineral Reserves, and there is no certainty that any value will be realised from them.

Figure 24-9 and Figure 24-10 show the additional material scheduled (grey-blue and purple blocks) outside of the FS pit shells (brown). Mining the additional material would not require additional pushbacks, and the pit access and ramps in the existing mine design would not need to be changed.

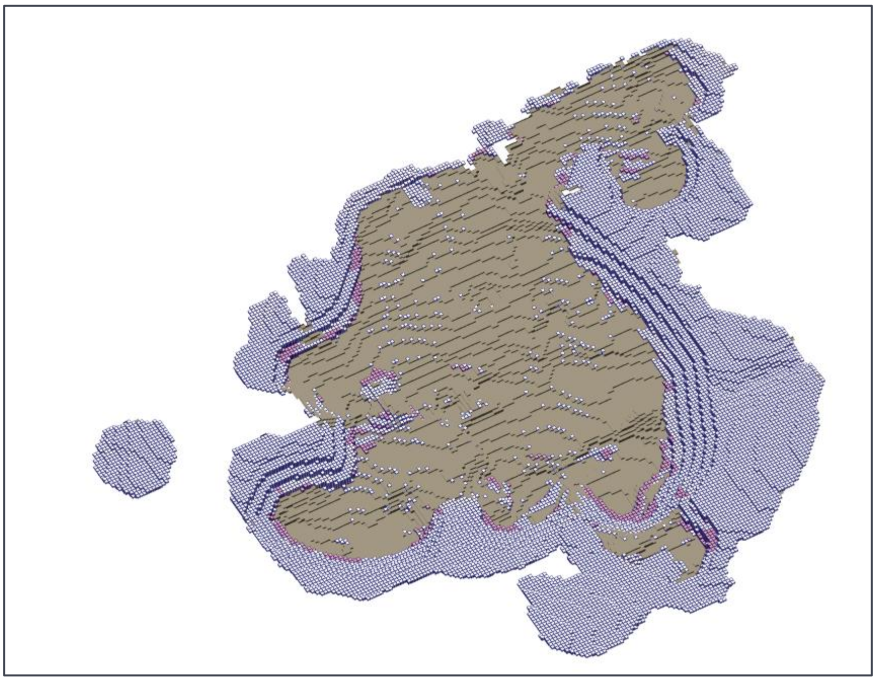


Figure 24-9: Additional material mined - Muntanga

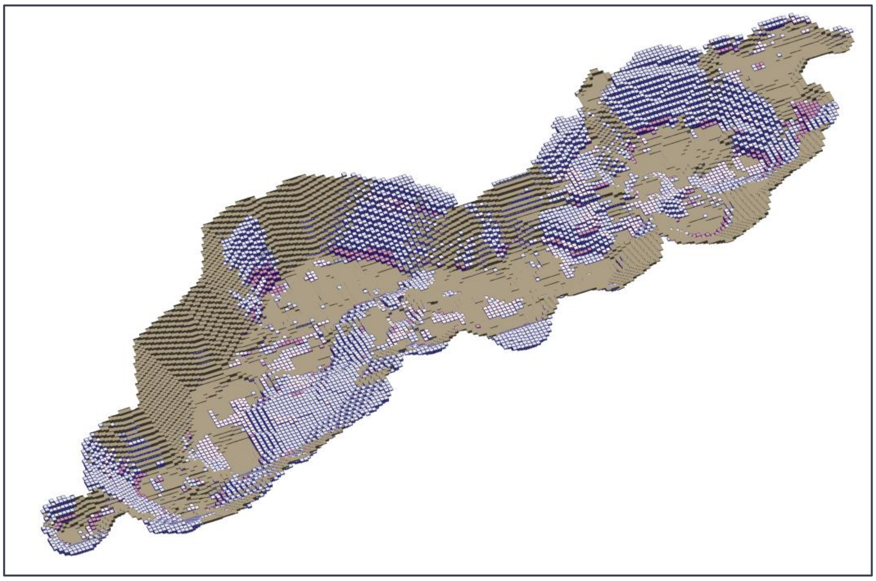


Figure 24-10: Additional material mined - Dibbwi East

Ukwazi recommends further drilling and study work to investigate upgrading the Inferred Resources.

24.4. Open pit and waste rock dump geotechnical study

SRK SA was engaged to complete the geotechnical pit slope design. This section of the report serves to summarise the work undertaken to provide appropriate slope design recommendations for the FS. The work was carried out in accordance with SRK Proposal 31372: Scope and Workplan for the Muntanga FS. The study commenced in early 2022, involving geotechnical drilling and sampling across all five mineralised deposits. The aim was to develop a geotechnical model for each site to guide and define suitable slope design parameters. This section summarises the work performed, including data collection, interpretation, and analysis, which resulted in the slope design recommendations presented.

24.4.1. Summary of work programme

The work programme of the study is summarised as follows:

- **Review of data:** Assessed geological, structural, and geotechnical data provided by GoviEx, including the resource drillhole database, geotechnical drillhole logs, previous reports and existing geological model
- **Develop geotechnical model:** Integrated geological, structural, rock mass, and hydrogeological models in collaboration with other disciplines
- **Slope stability analysis:** Conducted kinematic analyses, limit equilibrium and finite element analysis for bench, inter-ramp faces and overall slopes for open pits and WRDs
- **Design recommendations:** Provided slope design recommendations based on slope stability analysis results.

24.4.2. Geological setting

24.4.2.1. Regional geology

The Project area is located within the Karoo Supergroup, a significant sedimentary formation spanning much of Southern Africa. This geological group, which dates from the Carboniferous to Triassic periods, formed during the breakup of Gondwanaland. It includes the Lower Karoo Group, consisting of basal conglomerates, siltstones, sandstones, and mudstones, and the Upper Karoo Group, which features the Escarpment Grit along with interbedded mudstones and sandstones.

The Escarpment Grit, a 400 m thick sequence of arenaceous silici-clastic sediments, is the primary host for mineralisation at Muntanga. This unit is overlain by sandstone and mudstone formations, while the Madumabisa Mudstone below acts as an impermeable barrier controlling the base of mineralisation. The Escarpment Grit is divided into two informal facies: the "Braided Facies" and the more extensive "Meandering Facies," with the Dibbwi East located within the Meandering Facies. The geology at Gwabi and Njame consists entirely of thick coarse conglomerate beds to thinly bedded fine to medium-grained sandstones of the Escarpment Grit formation.

24.4.2.2. Local geology

- **Muntanga:** This deposit is characterised by three stratigraphic packages within the Escarpment Grit
  1. **Package A:** Comprises grey mudstones transitioning to coarse-grained sandstones and conglomerates, capped by a 5 m thick conglomerate unit
  2. **Package B:** Features fining-upward cycles from coarse conglomerates to siltstones and mudstones, with sulfides present around 50 m depth. Above this depth, oxidation and weathering are evident
  3. **Package C:** Consists of coarse-grained sandstones with fewer mudstones compared to Package B
- **Dibbwi and Dibbwi East:** These areas show mineralisation hosted by un-faulted meandering facies of the Escarpment Grit. The strata dip 8° to 15° SE and strike NE-SW, with sandstone layers ranging from 10 m to 50 m in thickness, alternating with 2 m to 5 m thick mudstones and siltstones
- **Njame:** This area features Escarpment Grit on a gently dipping slope, intersected by secondary and strike-parallel faults. It includes a variety of clastic sediments and is divided into five facies packages, displaying a fining-upward trend
- **Gwabi:** Similar to Njame, Gwabi contains Escarpment Grit on a gentle southeast slope, characterised by coarse conglomerates and fine to medium-grained sandstones, intercalated with shale and siltstone layers.

Geological logging across these areas reveals a dominance of fine to coarse-grained sandstones, with interbedded mudstones and siltstones present throughout the Muntanga, Dibbwi, Dibbwi East, Njame, and Gwabi areas. Litho-structural models were developed for all study areas using all available geological data, in Leapfrog GeoTM. To develop the appropriate lithology models, drillhole lithology intercepts were grouped into argillaceous units (siltstone, mudstone) and arenaceous units (conglomerate, grit, sandstone) and these are presented in Figure 24-11.

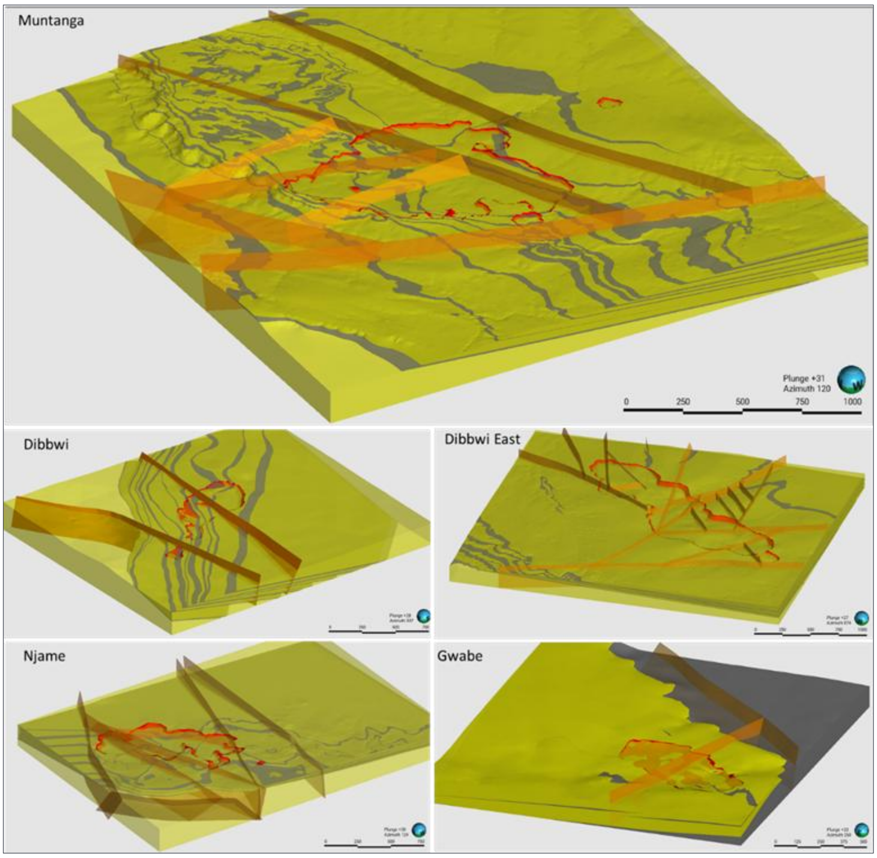


Figure 24-11: Lithology and fault models for the sites (arenite- yellow, argillite- grey), pit shells shown in red

24.4.2.3. Structural model

The Project is located in the Zambezi Rift Valley Basin, a NE-trending half-graben rift system. Faulting began in the Ordovician period and was reactivated during the Jurassic and with the development of the African Rift system. The area features NE to ENE trending fault-bounded blocks with steep SE dips. Key fault systems include the Lusitu, Dibbwi, and Bungua mountain faults. Regional seismic data indicates NW-SE crustal extension, supporting normal fault kinematics. Fault modelling used historical maps, geophysical data, and interpreted lineaments, supplemented by topographic and geotechnical drillhole information where available and is presented in Figure 24-12. Due to limited data, these models have low confidence, with most faults modelled as steep structures with unknown dip angles. Regional outcrop mapping is recommended to enhance the understanding of fault orientations and their impact on slope stability.

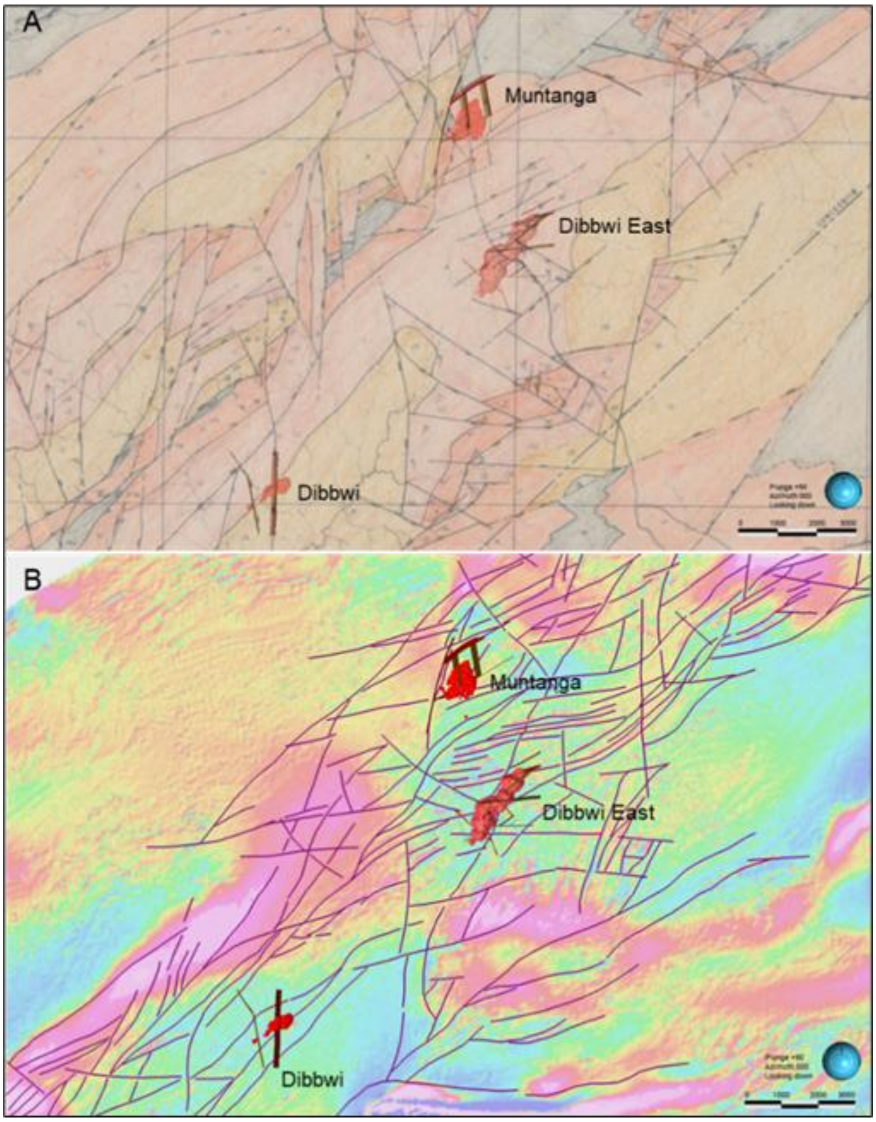


Figure 24-12: A) Historical geology map and B) regional magnetic data with SRK lineament analysis to identify major fault systems in the southern Muntanga-Dibbwi East and Dibbwi area

24.4.2.4. Structural defect analysis

In 2023, a structural defect analysis was conducted across the Project area using only geotechnical logging data from 13 out of 14 drillholes, due to joint orientation logging issues and low-confidence data from earlier resource holes. The analysis revealed the following key findings:

- **Dibbwi Pit:** Two geotechnical boreholes identified two primary joint sets, a dominant sub-horizontal bedding set and a moderately SE dipping set. A steep NW dipping set was also present but not consistently detected in both boreholes due to orientation bias
- **Dibbwi East Pit:** Six boreholes detected several joint sets, including the flat bedding set, a steep NW/SE dipping set, and moderately dipping sets. Although orientations varied, all identified joint sets were assumed to be present throughout the pit
- **Muntanga Pit:** Two boreholes showed the dominant flat dipping joint set and additional joint sets. The joint sets appeared to have a more moderate dip compared to other pits, with combined data from both boreholes used for a representative analysis
- **Njame Pit:** Two boreholes identified the flat dipping and moderately SE dipping joint set, along with a prevalent steep NW/SE dipping joint set. The moderately NE dipping joint set was not detected in some boreholes but was assumed to be present across the pit
- **Gwabi Pit:** Only one of two boreholes was oriented, confirming the presence of the flat dipping joint set, moderately SE dipping joint set, and steep NW dipping joint set. Due to limited data, structural data from the adjacent Njame pit was used to supplement the Gwabi analysis.

Although some variations in joint sets were identified per drillhole, the absence of a joint set in one drillhole, where it was present in others, was considered unreliable due to potential drillhole orientation biases. Joint sets identified in one pit but not in others were considered to be ubiquitous across all sites and applied to all pits. The joint data presented in this section was used as the basis to determine the risk of structurally controlled failures across the study area.

24.4.3.Rock mass model

Rock mass strength is determined by both the strength of the intact rock and the properties of its discontinuities. This relationship is described using the Generalised Hoek-Brown Criterion, with parameters obtained from geotechnical logging and laboratory tests. It comprises four inputs which are summarised as follows.

1. Intact rock strength (“IRS”) was determined through uniaxial compressive strength testing on 82 samples from various sites. Sandstone, the dominant rock type, exhibited an average strength ranging from 20MPa to 30 MPa in unweathered zones. Conglomerate and siltstone displayed higher strengths, with values of 56 MPa and 38 MPa, respectively. Mudstone strengths ranged from 20 MPa to 43 MPa, although these results are biased as only competent samples could be selected whereas a lot of the mudstone was weak and disintegrated. Due to a limited number of samples and their limited exposure interbedded with the sandstone, conglomerates and siltstones were grouped with sandstones for analysis.
2. The geological strength index (“GSI”) was calculated using an internal SRK method based on Laubscher’s rock mass rating. The results from the rock mass model provide a detailed outline of the strength parameters and geological characteristics necessary for assessing rock mass behaviour and stability in the Project.
3. The material constant mi describes the relationship between the principal stresses at failure. Several methodologies can be used to calculate mi, and two different methods were applied, where sufficient data was available for the geotechnical domain considered. Where data was limited, published values for similar materials were applied.
4. The effect of blast damage on the slope is accounted for by the implementation of a disturbance factor (D), which is a rating of 0 (no disturbance) to 1 (high disturbance). For this study, a blast damage zone was applied at the face with the factor of D=0.7, which then decreased incrementally away from the face to a factor of 0. A summary of the rock mass parameters applied in the study is presented in Table 24-14.

Table 24-14: Summary of rock mass strength parameters applied to the analyses

Parameter	Rock type	Soil	Mudstone	Sandstone
GSI	HW		15	25
	MW		35	45
	SW/UW		35	55
UCS, MPa	HW			5
	MW		28.5	20
	SW/UW			30
m_i	HW			
	MW		5	18
	SW/UW			
D	HW			
	MW		0.7 - 0	
	SW/UW			
Unit weight, kN/m^3		17.9	22.9	24.2
Cohesion (Kpa)		10		
Friction angle (°)		30		
Poisson's ratio	HW		0.2	0.4
	MW	0.25	0.3	0.4
	SW/UW		0.4	0.5
Young's modulus (GPa)	HW		2.6	7.0
	MW	0.01	6.2	7.0
	SW/UW		12.9	7.0

24.4.4.Hydrogeological model

The hydrogeological study was conducted to assess groundwater inflows, quality, and dewatering requirements for the Project. Utilising data from pump tests, hydraulic test boreholes, pit exploration holes, and water supply wells. The study aimed to develop a comprehensive understanding of groundwater dynamics to support pit slope stability analyses. The static water level observed in the drilled boreholes exhibits considerable variability across the project site. The Dibbwi East area has static water levels between 53 m and 62 m below ground, while the Gwabi site experiences artesian conditions with water at the surface.

Two numerical groundwater models were created:

1. Model 1 for the Dibbwi, Dibbwi East, and Muntanga areas, and
2. Model 2 for the Gwabi and Njame areas.

These models incorporate geological data, water levels, precipitation, evaporation, pumping needs, aquifer characteristics, and mining schedules.

Five scenarios were analysed to estimate pit inflows and dewatering requirements.

For slope stability, Scenario 1, the baseline case with no water management measures, indicated that inflows would cause seepage faces and wet pit floors. While significant seepage is not expected in the Muntanga, Dibbwi East, and Dibbwi pits, transient pore pressure from mudstones and high recharge events could occur. Effective dewatering is crucial for maintaining dry pit floors and ensuring safe operations.

The study underscores the need for robust groundwater management and dewatering strategies to maintain slope stability and operational safety, recommending additional measures to manage increased inflows and control pore water pressures.

24.4.5.Stability analysis

24.4.5.1. Kinematic analysis

Kinematic analysis was performed to evaluate potential failure mechanisms in pit walls, focusing on major discontinuities and slope geometry. The analysis includes the pits at Dibbwi, Dibbwi East, Muntanga, Njame, and Gwabi. The results (an example of which is presented in Figure 24-13), indicate that failure probabilities are within acceptable criteria for both batter and inter-ramp slopes. Recommendations include maintaining batter face angles at or below 80° and addressing small-scale failures with operational measures, such as catchment provisions.

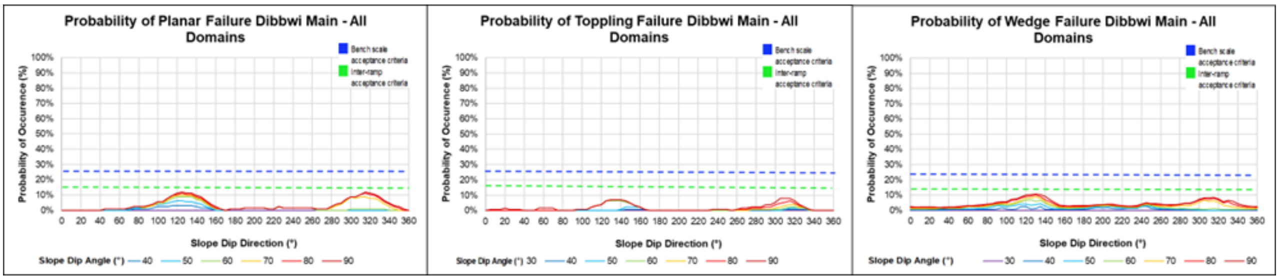


Figure 24-13: Graphs showing PoO of planar, toppling and wedge failure for the Dibbwi East Pit

Additionally, kinematic analysis was used to assess the impact of faults by identifying where modelled faults intersect pit walls and evaluating the risk of localised structural instabilities. The key findings are:

- **Dibbwi Pit:** Only one fault intersects the pit walls. The risk of planar or toppling failures is minimal, and the probability of wedge failures is low (less than 10 %) for slopes flatter than 70°
- **Dibbwi East Pit:** Fifteen out of twenty modelled faults intersect the pit. Wedge failure probabilities exceed acceptable limits at steeper slope angles (greater than 80°). It is recommended to limit batter face angles to 80° to mitigate these risks
- **Muntanga Pit:** Of the seven modelled faults, three intersect the pit. The probability of wedge failures remains below 10 %, even at 90° slope angles
- No analysis was carried out for the Njame and Gwabi pits due to the very limited data and low confidence in fault models for these areas.

The analysis suggests that structurally controlled failures are likely to be small and localised and can be managed with operational controls. Continued outcrop mapping and data collection are recommended to enhance confidence in structural stability and failure predictions.

24.4.5.2. Slope stability analysis

2D limit equilibrium stability analyses using Slide 2 and finite element analysis with RS2 were conducted for the Dibbwi East pit, the largest and deepest pit in the Project area. The phased modelling approach was considered to provide early geotechnical feedback essential for mine planning. Initial analyses assumed horizontal strata and used estimated rock mass strength parameters from borehole logs and field observations, and typical values for similar rock types, in the absence of rock testing results and a litho-structural model. The results showed that all slopes have a FoS greater than 2.5, exceeding the acceptance criterion of 1.5.

Detailed mudstone analysis, based on drillhole logs and core photographs, highlights the importance of mudstone in slope stability. Mudstones can disintegrate and lead to slope failures or bench loss. Significant discrepancies were found between geotechnical and Mineral Resource datasets regarding mudstone content, a summary of the percentage of mudstone per data source is presented in Table 24-15. For Muntanga, these differences required conservative modelling. For other pits, data comparisons and client feedback refined mudstone percentage estimates. An analysis was carried out to determine the expected thickness of the mudstone units, and the data suggests that mudstone layers are generally less than 5 m thick, an example of this is shown in Figure 24-14. Runout analysis for a potential bench failure estimated a runout of 12.3 m, resulting a recommendation for conservative bench design at Muntanga due to its higher mudstone content.

Table 24-15: Summary of logged mudstone and siltstone from different datasets

Pit Area	Litho-structural Model [%MST and SLT]	Resource Holes [% MST and SLT]	Resource Holes [%MST]	2023 Geotech Holes [% MST and SLT]	2024 Geotech Holes [% MST]
Dibbwi East	20	3	**1**	23	**6**
Dibbwi	16	18	**4**	18	**7**
Muntanga	25	40	**28**	3	**3**
Njame	21	14	**0**	27	**14**
Gwabi	16	15	**0**	34	**20**

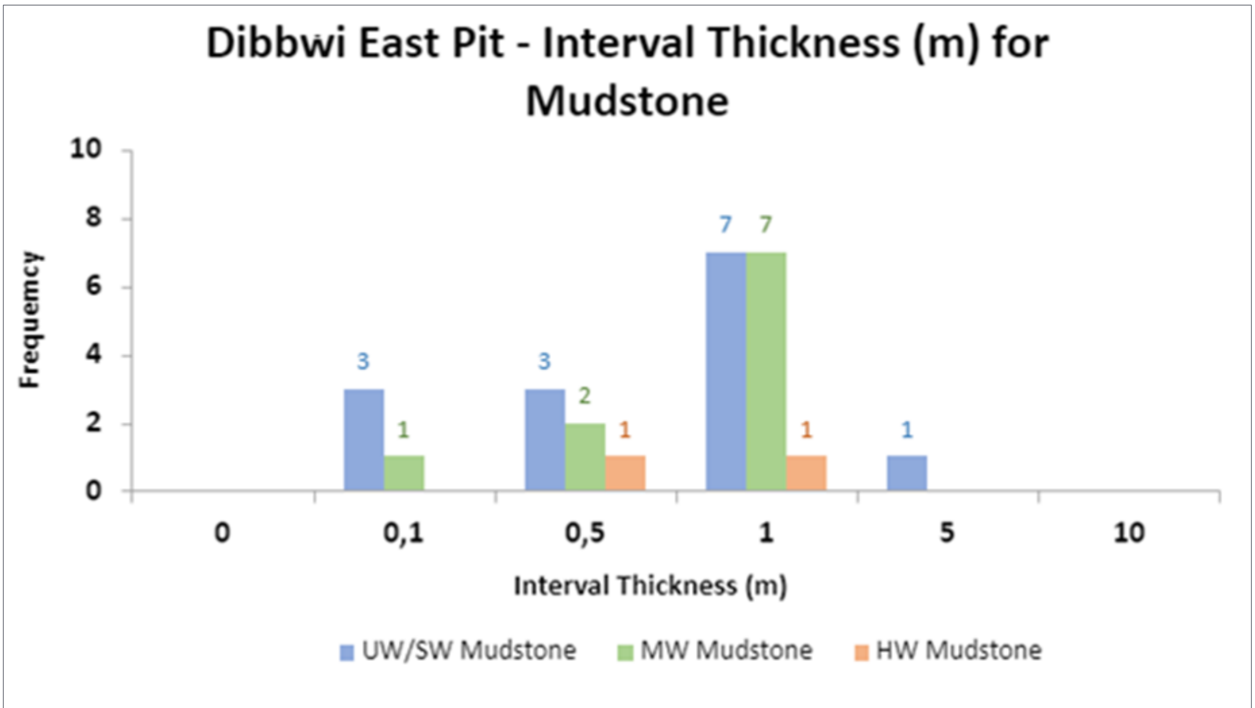


Figure 24-14: Mudstone interval thickness distribution for Dibbwi East

Once laboratory data was received, and the litho-structural model developed, updated analysis was carried out using limit equilibrium and finite element analysis and new rock mass parameters confirmed that initial stability assessments remain valid, with no expected overall or inter-ramp failures. While the finite element analysis supported the limit equilibrium results, it highlighted potential risks associated with weak, weathered mudstone at the pit bottom, especially on the NW wall where bedding dips approximately 15° to the SE. The high FoS suggest that slope optimisation may be possible. However, a better understanding of mudstone behaviour, including its presence, thickness, and characteristics, is necessary before any optimisation of the slope design.

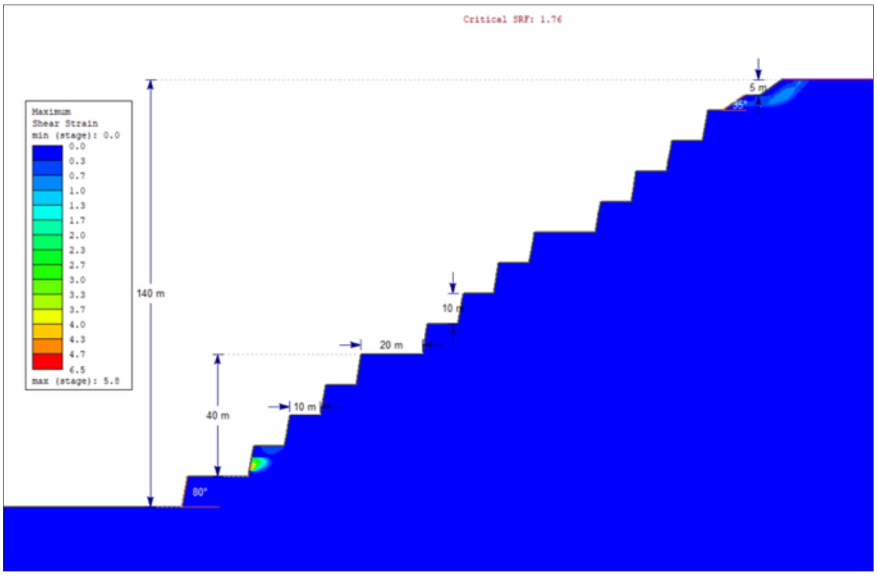


Figure 24-15: Finite element analysis results for the Dibbwi East pit, Section CS3 SE wall, SRF=1.76

Based on the analysis, slope design geometries are recommended and provided in Table 24-16.

Table 24-16: Slope design recommendations for the LOM pit plan

Description	Maximum slope height [m]	Bench height [m]	Batter angle [˚]	Berm width [m]	Geotechnical berm width [between stacks]	Ramp width [m]	Overall slope angle [˚]
Overburden	10	5	35	5			
Dibbwi and Dibbwi East	140	10	80	10	20	25	39
Muntanga	60	10	80	13	No berms required for ~60 m pits	25	34
Njame	60	10	80	10	No berms required for ~60 m pits	25	38
Gwabi	50	10	80	10	No berms required for ~50 m pits	25	38

24.4.6. Waste rock dump analysis

An analysis of WRD stability at the Muntanga Mine was conducted, focusing on both in-pit and external dumps. The stability analyses utilised the designed WRDs based on the following specifications:

- Lift height: 15 m
- Bench slope angle: 37˚
- Berm width: 40 m for Muntanga dump and 15 m for other dumps
- Ramp width: 25 m
- Ramp gradient: 10 %.

The Barton-Kjaernsli methodology was used to determine the shear strength parameters essential for evaluating WRD stability. The analysis primarily concentrated on the Njame WRD, the steepest and highest external dump in the area. The design of the Njame WRD met the minimum stability criteria, with FoS ranging from 1.7 to 1.8 for both coarse and fine materials at various porosity levels. This indicates that the external dump is generally stable.

For the Dibbwi East pit, which contains the largest and highest in-pit dump, the analysis revealed the following:

- **Worst-case scenario:** Under baseline worst-case hydrogeological conditions, the in-pit dump did not meet the minimum design criteria for any combination of waste rock consistency and porosity. Dewatering was recommended to maintain stability; however, depending on risk tolerance, alternatives such as allowing flooding might be considered
- **Favourable scenario:** In a more favourable hydrogeological scenario, where water levels are maintained below the pit bottom, the overall slope of dump design meets the stability criteria. Stability in this scenario is generally acceptable, with factors of safety exceeding 1.2 for most conditions.

The analysis indicates that while external dumps generally meet stability criteria, in-pit dumps require careful management, particularly regarding dewatering and groundwater control. The findings emphasise the importance of implementing conservative design practices and proactive stability management to mitigate potential risks associated with WRD construction and operation.

24.4.7. Conclusion and recommendations

The geotechnical study for the Project evaluated the stability of open-pit slopes and WRDS and established geotechnical criteria for open-pit design. The analysis, which included kinematic assessments, led to the following recommendations:

- **Bench face angles:** Recommended as 80° for all pits and 35° for overburden to ensure stability
- **Berm widths:** Specified as 5 m for overburden, 10m for most pits, and 13 m for the Muntanga pit. A geotechnical berm width of 20 m is suggested for the Dibbwi and Dibbwi East pits, which will have multi-stack slopes
- **Bench heights:** Set at 5 m for overburden and 10 m for all pits.

The study concluded that overall slope failures are unlikely, although some bench-scale failures are expected but manageable. Key risks include potential bedding parallel failures in weak mudstone, which do not necessitate immediate design changes.

Recommendations for future work once in operation include:

- Comprehensive assessment of geological structures to refine pit design
- Detailed bench mapping and ongoing geotechnical data collection to optimise design
- Effective surface water management and seismic risk assessment
- Regular updates to geotechnical models based on operational data.

By following these recommendations, the Project aims to enhance stability and safety. Include any additional information or explanation necessary to make the TR understandable and not misleading.

25. Interpretation and conclusions

Ukwazi and SRK's interpretations of the geology, Mineral Resources and feasibility level studies of mining, infrastructure and processing options for the Project are as follows:

Ukwazi Transaction Advisory and SRK (UK) Limited, with the assistance of SGS-Bateman, have completed technical studies to a feasibility level of confidence for the Muntanga Uranium Mine open pit project, process plant and associated infrastructure. The Project development plan projects mining a total of 39.6 Mt of ore from the central Muntanga and Dibbwi East pits, at an average grade of 320 ppm U_3O_8. The average process recovery of 90.5 % will yield a total of 25.3 Mlb of saleable yellowcake product over the 12-year life of mine.

Initial capital costs are estimated at USD 282 USD million and sustaining capital at USD 101 million, giving LOM total capital costs of USD 383 million. Total operating costs (excluding royalties) are USD 827 million, translating to USD 20.9 /ROM t or USD 32.2 /lb saleable U_3O_8. A long-term uranium price of USD 90 /lb U_3O_8 was applied in the base case financial analysis. The DCF model for the project shows a total LOM net free cash of USD 673 million, which at a discount rate of 8 % gives an after-tax NPV of USD 243 million, with an IRR of 20.8 %.

In the schedule presented, some of waste rock which is discarded consists of mineralised material from Inferred Mineral Resources, which could potentially be upgraded if drilling is conducted to better characterise these resources. There are also Indicated and Inferred Mineral Resources lying outside of the designed pit shell. A pit shell was developed including these, which indicates the potential to increase production volumes and mine life. We also recommend drilling and further study work to better characterise these Inferred Mineral Resources, and examine whether they could be included in LOM mine plan. Please note that this mineralised material contains Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as Mineral Reserves, and there is no certainty that any value will be realised from them.

The satellite pits, Dibbwi, Gwabi and Njame, were studied with the same FS-level of detail as the Central pits, and are developed to a point where they could be implemented. Economic analysis shows that the financial performance of the satellite pits is positive at the USD90 / lb FS base price for U_3O_8, but at this stage the satellite pits would detract from the financial performance of the overall project. However, at higher prices, the performance will improve and GoviEx is in a position to wait for favourable market conditions at which time the satellite operations could be built and operated.

Ukwazi and SRK conclude that the Muntanga Uranium Mine project demonstrates technical and financial feasibility and is in a position to advance to the next stage of project development.

As with every mining project, risk is inherent in the development of the Muntanga Uranium project. During the course of the feasibility study, risks were identified which would threaten the ability of the Project to be developed according to schedule and within budget, and to operate at the targeted production levels at the expected operating costs. Risk identification was done by individual technical teams during the course of their work, in two multidisciplinary workshops attended by representatives of all technical teams and GoviEx, and a specific risk workshop. As a result, several improvements and changes were made to aspects of the project to reduce the likelihood of risks occurring, and/or reduce the impacts of these risks should they occur.

The more significant risks identified include:

- **Resettlement action plan:** Community dissent and dissatisfaction may cause difficulties in implementing the RAP, resulting in delays in implementation of the entire project and/or increased costs. To address this risk, the RAP is being developed by a Zambian company, AMC, with specific experience in that country, and the process of negotiation with affected people will be structured so that communication is clear. The RAP has been designed so that is resettlement aligned with the Project implementation plan
- **Availability of labour:** GoviEx has a policy of staffing the Muntanga project with Zambian people, and drawing as much labour as possible from local communities. If sufficient numbers of appropriately qualified employees are not available, this could lead to delays in implementation, low productivity, the displacement of local people by "imported" skilled labour and dissatisfaction among local community. To mitigate this risk, GoviEx is already running education and training programmes in the local community to increase the level of education and hence potential employees, and will be able to allocate unskilled positions to community members. If enough people cannot be found locally, the large pool of qualified mining skills in Zambia can be drawn on, with expatriate labour as a last resort.
- **Availability of power:** Power may not be available at times during construction, commissioning and operation due to outages on the ZESCO supply, and in the early stages of project development, late completion of the power supply infrastructure. This could lead to delays in project completion and loss of production during operations. This risk can be mitigated in various ways: contractors could be required to provide power as part of their scope of work, diesel generator sets cold be installed, close management of

the progress of powerline construction, and favourable contract terms in the power purchase agreement with ZESCO. A solar system was included in the design scope (see Section 18.4.5.4) which could be installed should power supply problems warrant it. Existing diesel generator backup is only sufficient to run critical loads. It has been confirmed by Utilink from prior historical knowledge, that the availability of both substations used to supply the Project is understood to be >96 % at the primary voltage level, but the Zambian power sector is currently experiencing a "challenging period" with drought causing low water levels in reservoirs and as a consequence, power rationing (eight-hour daily load shedding) across the country.

- **Logistics:** Most equipment for the Project will be sourced from outside of Zambia, and logistical complications in transport, customs and border controls could hinder the arrival of equipment on site, leading to a delay in implementation and higher costs. This can be addressed by timeous planning and ordering of equipment and the use of experienced logistics companies with a track record in the delivery of equipment to Zambia.
- **Market risk:** Every mineral project is subject to market risk, driven by uncertainty in the price of the commodity/ies that it produces. The uranium market research presented in this FS indicates that future demand/supply imbalance will result in favourable prices, but there is no assurance that these will be achieved. However, this is also an upside opportunity, and higher prices may prevail. The Project breaks even at price of USD 63.32 /lb U_3O_8, and the risk may be further mitigated by a mix of spot and term contracts as discussed in Section 19.8.
- **Mining:** specific risks mainly relate to the potential variation in the estimated tonnage an associated ROM grades. The Muntanga and Dibbwi East deposits consists of multiple overlaying ore bodies with internal waste requiring highly selective mining methods. Defined ore and waste is not visually distinguishable with identical RDs of 2.1 t/m³. Appropriate grade control practices must be developed, implemented and maintained during the operational phase to achieved planned production outputs. Not implementing appropriate grade control procedures may have a material impact on the ROM grades resulting in reduced process recoveries and increased unit costs. Reduced process recoveries and increased unit costs may have a material impact on the estimated Mineral Reserve. Appropriate schedule delays (15-hour delay per mining block) were incorporated in the production schedule to cater for grade control requirements.

26. Recommendations

The following recommendations are provided to advance the understanding of the geology, mineralisation controls, Mineral Resources (and possibly the Mineral Reserves) for the Project:

- Continue development of litho-structural models for the Project deposits, incorporating major fault interpretations within the vicinity of active mine areas or proposed future project infrastructure
- Continue infill drilling to support the conversion of Inferred to Indicated Mineral Resources
- Continue further exploration of other potential orebodies in the GoviEx licence areas
- Additional assay sampling to support further refinement of the Ra-grade correlation used to convert down-hole probe data into equivalent uranium grades
- Continue to assess for radon contamination within future drilling programmes and correct down-hole gamma signatures accordingly to mitigate the potential for over-estimation of grade due to radon
- Additional density analysis should be conducted on future drill programmes to refine tonnage estimates.

There are several components of the process design that can be optimised by future testwork.

- The control of iron leaching in the heap, and hence peroxide consumption, can be optimised by recirculating solutions continuously through a number of cycles using small lab columns
- The final product precipitation process can be optimised with respect to impurity deportment, particularly iron. These tests can be done using PLS produced by the small column tests described above, using lime sourced from Zambia
- Finally, rheology work can be done using the gypsum and uranium slurries produced above, to finalise parameters for sizing the various thickening and filtration equipment.