Dasa Uranium Project

Feasibility Study

NI 43-101 Technical Report 0376-00-00-REP-0001

Prepared For

Global Atomic Corporation

Qualified Persons

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Amended Date – January 13, 2026

Report Prepared By – METC

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Certificate of Qualified Person – Dmitry Pertel

I, Dmitry Pertel, am employed as the Principal Geologist with AMC Consultants Pty Ltd with an office address at Level 1, 1100 Hay Street, West Perth, Western Australia 6005.

I am an author of this technical report entitled "Dasa Uranium Project, Feasibility Study, NI 43-101 Technical Report 0376-00-00-REP-0001", prepared for Global Atomic Corporation with the initial report dated March 27, 2024 amended as of January 13, 2026 for the Dasa uranium project ("Dasa Project") located in Niger, and with an Effective Date of February 28, 2024 (the "Technical Report"), and do hereby certify that:

- I am registered as a Professional Geoscientist, Certificate number 2248 (Australian Institute of Geoscientists).
- I am a graduate of the Saint Petersburg Mining University in 1986 with a Masters degree in Geology.
- I am a Member of the Australian Institute of Geoscientists (AIG). I have worked as a Geologist for a total of 36 years since my graduation. My relevant experience for the purposes of the Technical Report is:
- development and reporting of mineral resource models; and
- review and report QA/QC procedures and protocols, site visits and laboratory inspections.
- I have acted as a Principal Geologist on a number of mineral resource studies and development of block models for the uranium industry in Africa, Australia, Turkey, and Asia.
- I have read the definition of "Qualified Person" set out in National Instrument 43-101 (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.
- I have visited the Dasa Project between 25 March and 6 April 2017 with five days at the deposit site and exploration camp, and several days at the Global Atomic Corporation office in Niamey, Niger.
- I am responsible for preparation of Sections 7 to 9 (inclusive), 6, 10, 11, 12, 14 and Section 23, and I have made contributions in Sections 1, 2, 24, 25, 26, of the Technical Report.
- I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
- I have prior involvement in the Dasa Project. I have:
- Co-authored the report titled "Dasa Uranium Project, Phase 1 Feasibility Study, NI 43-101 Technical Report 0093-00-00-REP-0010" revised and amended as of 9 January 2023 with an effective date of 15 November 2021
- co-authored CSA Global's report titled "Dasa Uranium Project, Preliminary Economic Assessment, NI 43-101 Technical Report, report No. R203.2020", with an effective date of 15 April 2020.
- authored CSA Global's report titled "NI 43-101 Technical Report for the Dasa Uranium Project Mineral Resource Update, Central Niger", prepared for the Issuer and with an Effective Date of 1 June 2019.
- co-authored CSA Global's report titled "NI 43-101 Technical Report on the Preliminary Economic Assessment – Dasa Uranium Project, Central Niger", prepared for the Issuer and with an Effective Date of 16 November 2018.
- co-authored CSA Global's report titled "NI 43-101 Technical Report on the Dasa Uranium Project, Mineral Resource Update, Central Niger", prepared for Global Atomic Corporation and with an Effective Date of 1 June 2018.
- co-authored CSA Global's report titled "NI 43-101 Technical Report - Dasa Uranium Project, Central Niger", prepared for the Global Atomic Corporation and with an Effective Date of 30 April 2017.
- I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the Effective Date, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 13 day of January 2026 at Perth, Western Australia

s/"Dmitry Pertel"

Dmitry Pertel, M.Sc., MAIG

AMC Consultants Principal Geologist

Certificate of Qualified Person – John Edwards

I, John Edwards, am employed as the Chief Metallurgist with METC Engineering with an office address at 12 Autumn Road, Rivonia, Johannesburg, South Africa 2191.

I am an author of this technical report entitled "Dasa Uranium Project, Feasibility Study, NI 43-101 Technical Report 00376-00-00-REP-0001" (the "Technical Report"), prepared for Global Atomic Corporation with the initial report dated March 27, 2024 amended as of January 13, 2026 for the Dasa uranium project (the "Dasa Project") located in Niger and with an effective date (the "Effective Date") of February 28, 2024 (the "Technical Report"), and do hereby certify that:

- I am a Fellow of the South African Institute of Mining and Metallurgy, registration number 701196.
- I am a Professional Metallurgist having graduated with a BSc Hons in Mineral Processing Technology in 1985 from Camborne School of Mines, UK, and I have worked as a Metallurgist for a total of 37 years since my graduation from university.
- My uranium experience relates to operations of plant in South Africa along with South Africa, Namibia, Tanzania, and Spain.
- I have read the definition of "Qualified Person" set out in National Instrument 43-101 – 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.
- I visited the Dasa Project site between 3 and 12 December 2020 with 4 days at the deposit site and exploration camp and 5 days at the Issuer's office in Niamey, Niger.
- I am responsible for Sections 3, 4, 5, 13, 17, 19, 20 and 27 of the Technical Report and am responsible for my respective contributions to Sections 1, 2, 18, 21, 22, and 24 to 26 (inclusive).
- I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
- I have prior involvement in the Dasa Project. I have:
- Co-authored the report titled "Dasa Uranium Project, Phase 1 Feasibility Study, NI 43-101 Technical Report 093-00-00-REP-0010" revised and amended as of 9 January 2023 with an effective date of 15 November 2021.
- Co-authored CSA Global's report titled "Dasa Uranium Project, Preliminary Economic Assessment, NI 43-101 Technical Report, report No. R203.2020" with an effective date of 15 April 2020.
- I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the Effective Date, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 13 day of January 2026 in Johannesburg, South Africa.

s/"John Edwards"

John Edwards, B.Sc. (Hons)

Certificate of Qualified Person – Andrew Pooley

I, Andrew David Pooley, am employed as the Chairman of Bara Consulting with an office address at 1st Floor Cresta Corner, Judges Avenue, Cresta, Johannesburg, 2194, South Africa.

I am an author of this technical report entitled "Dasa Uranium Project, Feasibility Study, NI 43-101 Technical Report, 0093-00-00-REP-0010" (the "Technical Report"), prepared for Global Atomic Corporation with the initial report dated March 27, 2024 amended as of January 13, 2026 for the Dasa uranium project (the "Dasa Project") located in Niger and with an effective date (the "Effective Date") of February 28, 2024, and do hereby certify that:

- I am a Fellow of the Southern African Institute of Mining and Metallurgy, registration number 701458.
- I am a Professional Mining engineer having graduated with a B.Eng. (Hons) in Mining Engineering from Nottingham University, UK, and I have worked as a Mining Engineer for approximately 30 years since my graduation from university.
- My uranium experience relates working on gold/uranium operations in South Africa as well as involvement in uranium projects in South Africa, Namibia, Central Africa, and Spain.
- I have read the definition of "Qualified Person" set out in National Instrument 43-101 – 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.
- I have not visited the Dasa Project site.
- I am responsible for Sections 15 and 16 of the Technical Report and am responsible for my respective contributions to Sections 1, 2, 18, 21, 22, 24, 25 and 26.
- I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
- I have prior involvement in the Dasa Project having:
- Been the QP for the maiden Mineral Reserve estimate with effective date November 2021.
- Co-authored the report titled "Dasa Uranium Project, Phase 1 Feasibility Study, NI 43-101 Technical Report 0093-00-00-REP-0010" revised and amended as of 9 January 2023 with an effective date of 15 November 2021.
- I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the Effective Date, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 13 day of January 2026 in Johannesburg, South Africa.

s/" Andrew Pooley"

Andrew Pooley, B.Eng. (Hons)

Abbreviations and Units of Measurement

Abbreviation	Description
%	Percent
°	Degrees
°C	Degrees Celsius
3D	Three-Dimensional
AFFF	Aqueous Film-Forming Foam
AE3	Adrar Emoles 3
AE4	Adrar Emoles 4
AISC	All-In Sustaining Cost
AIC	All In Cost (Including Non-Sustaining Capital)
ARD	Absolute Relative Difference
ASL	Above Sea Level
BEEEI	Bureau d'Evaulation Environmental Et des Etudes d'Impact
BNEA	National Office of Environmental Assessment
Bq	Becquerel
c/s	Counts Per Second
CAD$	Canadian Dollars
CAPEX	Capital Expenditure
CEA	Commissariat A l'Energie Atomique (French Nuclear Energy Commission)
CEET	Comminution Economic Evaluation Tool
CL	Calliper Logging
cm	Centimetre(S)
CFM	Cubic Feet per Minute
CRM	Certified Reference Material
CSA Global	CSA Global Consultants Canada Limited
DCF	Discounted Cash Flow
DOL	Direct online
DS	Directional Survey
DTM	Digital Terrain Model
EBIT	Earnings Before Interest and Income Taxes
EBITDA	Earnings Before Interest, Taxes, Depreciation and Amortization
ESIA	Environmental And Social Impact Assessment
eU_3O_8	Equivalent Uranium Oxide
FS	Feasibility Study
g	Gram(S)
G&A	General And Administration
GAC	Global Atomic Corporation
GAFC	Global Atomic Fuel Corporation
GM	Geiger Muller (Probe)
GPS	Global Positioning System
GR	Gamma-Ray (Logging)
GT	Grade Thickness
h	Hour(S)
HARD	Half Absolute Relative Difference
HG	High Grade (Stockpile)

Abbreviation	Description
HPGR	High-Pressure Grinding Rolls
IBAT	Integrated Biodiversity Assessment Tool
ICP-AES	Inductively Coupled Plasma – Atomic Emission Spectroscopy
IDW	Inverse Distance Weighted
IMU	Inertial Measurement Unit
IRR	Internal Rate of Return
GAAP	Generally Accepted Accounting Practice
GAC	Global Atomic Corporation
K	Potassium
kg	Kilogram(S)
kg/t	Kilograms Per Tonne
km, km^2, km/h	Kilometre(S), Square Kilometres, Kilometres Per Hour
kt, ktpa	Kilo-Tonnes (Or Thousand Tonnes) Kilo-Tonnes (Or Thousands of Tonnes) Per Annum
kV	Kilovolt
KBA	Key Biodiversity Areas
L	Litre(S)
lb	Pound(S)
LCOE	Levelized Cost of Electricity (LCOE)
LG	Low Grade (Stockpile)
LiDAR	Light Detection and Ranging (Survey)
LOI	Loss On Ignition
LoM	Life of Mine
LSA	Landscape Study Area
LSL	Loaded Strip Liquor Solution (OK Liquor)
m, m^2, m^3	Metre(S), Square Metre(S), Cubic Metre(S)
M	Million(S)
MCCB	Moulded Case Circuit Breakers
ml	Millilitre(S)
mg, mg/L	Milligram(S), Milligrams Per Litre
MDU	Magnesium di-Uranate
MG	Medium Grade (Stockpile)
Mlb	Million Pounds
mm	Millimetre(S)
MRE	Mineral Resource Estimate
MSO	Mineable Shape Optimization
Mt, Mtpa	Million Tonnes, Million Tonnes Per Annum
MW	Megawatt
NCF	Net Cash Flow
NI 43-101	National Instrument 43-101
NPV	Net Present Value
NSR	Net Smelter Return
O:A	Organic: Aqueous
OK	Ordinary Kriging
ONAREM	Niger National Geological Survey
OPEX	Operating Expenditure
PEA	Preliminary Economic Assessment

Abbreviation	Description
PFN	Prompt Fission Neutron
PFS	Prefeasibility Study
PLS	Pregnant Leach Solution
PNC	Power Reactor and Nuclear Fuel Development Corporation
ppm	Parts Per Million
QA, QC, QA/QC	Quality Assurance, Quality Control, Quality Assurance/Quality Control
REF	Radioactive Equilibrium Factor
RL	Resistivity Logging
RoM	Run of Mine
SAG	Semi-Autogenous Grinding
SCSR	Self-Contained Self-Rescue
SD	Standard Deviation
SP	Spontaneous Polarization
t, tpa, tpd, tph	Tonne(S), Tonnes Per Annum, Tonnes Per Day, Tonnes Per Hour
t/m^3	Tonnes Per Cubic Metre
Th	Thorium
U_3O_8	Uranium
U	Uranium
UOC	Uranium Oxide Concentrate
US$	United States dollars
UxC	Ux Consulting Company LLC
VAT	Value Added Tax
VLG	Very Low Grade (Stockpile)
VSD	Variable Speed Drive
WDPA	World Database of Protected Areas
XLG	Extra Low Grade (Stockpile)
XRD	X-ray Diffraction
XRF	X-ray Fluorescence

1. SUMMARY

1.1. Principal Outcomes

The Dasa Uranium Project Feasibility Study is based on Global Atomic Corporation processing 8.047 million tonnes uranium-bearing ore grading 4,113 ppm U_3O_8 at the Dasa Deposit. Processing will take place over a 24-year mine plan to produce 68.1 Mlb U_3O_8 of recovered Yellowcake, with an average steady-state metallurgical recovery of 94.15% (overall average 93.4% due to contingency on ramp-up period).

All dollar amounts are stated in United States Dollars throughout this Report. This Report updates and replaces the previous Phase 1 Feasibility Study of the Dasa Project that had an effective date of November 15, 2021. It is anticipated that the mine life will be significantly extended as additional mineral resources are brought into the mineral reserve categories.

The Feasibility Study work summarized in this Report has concluded on a Mineral Reserve estimate of 8,047 million tonnes uranium-bearing ore grading 4113 ppm U_3O_8 to result in 73.0 Mlb U_3O_8 Probable Reserves.

The total capital cost carried in the economic model is $647 million, inclusive of $297 million in pre-production capital costs based on a Class 3 AACE standard and $350 million of sustaining capital costs.

Total operating costs, excluding royalties, are estimated to be $1,745 million, $25.62/ lb U_3O_8. The average all-in sustaining cost (AISC) is $35.47/ lb U_3O_8.

Based on an average selling price of $75 /lb U_3O_8, the after-tax net present value (NPV) at 8% is $917 million, the internal rate of return (IRR) is 38.4%, and the assumed payback period is 2.2 years once production begins.

1.2. Terms of Reference

This Report is based on information known to the authors and METC Engineering and includes: the outcomes of the exploration and evaluation programs completed by GAC at the Project, the 2023 MRE (AMC Consultants, 2023), and the feasibility study completed by METC up to and including February 28, 2024 (the "Effective Date").

The 2024 Feasibility Report Update replaces the previous Feasibility report issued in 2021 as reported in the National Instrument 43-101 (NI 43-101) Technical Report.

The Report is specific to the standards dictated by NI 43-101 (30 June 2011), companion policy NI 43-101CP, and Form 43-101F1 (Standards of Disclosure for Mineral Projects).

Mineral Reserves for the Dasa Uranium Project have been estimated based on the geology and Mineral Resource Estimate discussed in the relevant sections of this report. An engineering design and costing exercise has been undertaken to a Feasibility Study (FS) level of accuracy which will support the Mineral Reserve Estimate. The FS has addressed all required aspects of the project to enable the estimation of Mineral Reserves and is discussed in detail in this report.

METC Engineering Pty LTD (METC) prepared the previous Phase 1 Feasibility Study and NI 43-101 Technical Report (Feasibility Report) on the Dasa Project. In June 2023, Global Atomic Corporation (GAC) also appointed METC as the procurement and construction manager for the Dasa Project and in November 2023, GAC also engaged METC to prepare an updated Feasibility Study (the "Study). METC relied on specialist companies Bara Consulting and Epoch Resources to undertake the Mining and Tailing Storage Facility (TSF) respectively, who had been involved in the Phase 1 Feasibility Study. Insight R&D were independently engaged by GAC to undertake the test work to determine an appropriate recovery process and expected uranium recovery. The primary purpose of this document (the "Report") is to update the previous Phase 1 Feasibility Study to take into account additional Indicated Resources resulting from the 2021 – 2022 drilling program and to update the estimated reserves.

METC Engineering, Bara Consulting and Epoch Resources acted independently as consultants and were paid fees based on standard hourly rates for the services provided. The fees were commensurate with the work completed and was not contingent on the outcome of the work. Neither METC, Bara or Epoch nor any of their staff rendering the services in connection with this Report, had any material, financial or pecuniary interest in GAC or its subsidiaries, or in the Project.

1.3. Property Description, Location and Access

The Dasa Uranium Project (Dasa or the Project) is located in the central part of the Republic of Niger, West Africa and lies within the IN-BOUKATT Mining Permit area, a carve out of the Adrar Emoles 3 Exploration Permit area, which with the contiguous Adrar Emoles 4 Exploration Permit, were 100% owned by Global Atomic Fuels Corporation (GAFC), a wholly owned subsidiary of GAC, and form part of a larger package of properties in Niger in which GAFC maintained a 100% interest. The Exploration Permits expired in December 2023 and while GAFC has applied for an extension or renewal, there is no certainty that GAFC will be successful. The IN-BOUKATT Mining Permit is held by a Niger corporation, Societe Miniere de DASA SA (SOMIDA). The shares of SOMIDA are owned 80% by GAFC and 20% by the Republic of Niger.

The region is largely uninhabited and is characterized by nomadic villages and three small towns located on the highway west of the Project area that links the regional towns of Arlit 105 km to the north and Tchirozerine 60 km and Agadez 95 km respectively to the south.

The centre of the Dasa Project is positioned at longitude 7.8° East and latitude 17.8° North within the IN-BOUKATT Mining Permit, which has a total area of 25 km^2. Under NI 43-101 guidelines, the Adrar Emoles 3 (AE3) and Adrar Emoles 4 (AE4) Exploration Permits, subject to a successful extension or renewal application, are considered to be the same Property as they would reasonably share common infrastructure should a mineral deposit be developed on either concession. The AE3 Exploration Permit covers an area of 96.2 km^2 and the AE4 Exploration Permit which adjoins the southern border of the AE3 Exploration Permit has an area of 122.4 km$^{2.}$

The Project area is accessible by an all-weather road connecting Agadez, Niger's second largest city, located 95 km south of the Project with the mining town of Arlit some 105 km north of the area of interest, and the capital, Niamey, approximately 1,000 km to the southwest.

There are two airports serving the project general area: The Mano Dayak airport at Agadez, which was recently upgrade and has a 3,000-metre runway, and the country's international airport in the capital city of Niamey. There are regular charter flights and daily connections between Agadez and Niamey.

1.4. Mineral Tenure, Permitting and Royalties

Exploration Permits and Mining Permits were in the past granted within the provisions of Mining Conventions negotiated between the Ministry of Mines and the applicant. The AE3 and AE4 Mining Conventions cover a period of 20 years ending September 2027, being the exploration period (three years plus two three-year renewals) and the first 10-year validity period of a Mining Permit. Mining Permits are then renewable for 5-year periods until the resource is depleted. The Mining Convention is renegotiated at each renewal of the Mining Permit. The Mining Convention can only be amended upon the mutual consent of both parties. The agreement is approved by Decree of the Council of Ministers and then signed by the parties and stipulates rights and obligations of the parties during the validity period.

A new Niger Mining Code came into effect in July 2022 and Exploration Permits are now granted without a Mining Convention. Mining Conventions are entered into only in respect of Mining Permits granted. Existing Mining Conventions are grandfathered until their expiry.

GAC, through its 100% owned subsidiary GAFC also held the Tin Negoran Exploration Permits which are located approximately 100 kms south-west of the Dasa Project, cover an area of 486.2 km^2 and were also extended on January 23, 2021, for the period ending December 23, 2023. GAFC has also applied for an extension or renewal of the Tin Negoran Exploration Permits.

The Tin Negoran Exploration Permits have been the target of over 22,000 metres of drilling by Global Atomic. All six Exploration Permits which lie within the Tim Mersoï Basin which has produced uranium for the Republic of Niger for the last 50-years.

On December 23, 2020, the Republic of Niger, Ministry of Mines, granted a Mining Permit to GAFC on behalf of a Niger mining company to be incorporated by GAC for the Dasa Project. The Mining Permit has an initial term of 10 years and is renewable for successive 5-year terms, until the resource has been fully depleted. Subsequently, SOMIDA was incorporated, and the Mining Permit is now held by SOMIDA. Under the Niger Mining Code, the Republic of Niger has the right to a 10% carried interest in the common shares of the Niger mining company and may subscribe for up to an additional 30% in the common shares of the Niger mining company up to the date of the incorporation of the Niger mining company provided it commits to fund its proportionate share of capital costs and operating deficits for such additional interest.

On the incorporation of SOMIDA, the Republic of Niger elected to subscribe for 10% of the shares of SOMIDA in addition to its 10% free carried interest.

On January 28, 2021, GAFC received its Certificate of Environmental Conformity for the Dasa Project from Republic of Niger Ministry of Environment, Urban Health, and Sustainable Development.

GAFC has received all permits and approvals required for the development and commercial operation of the Dasa Project.

The Dasa Project is subject to royalties payable to the Niger Government. Current tenure provisions are adequate to conduct exploration, development, extraction, and processing activities on the Project.

1.5. Geology and Mineralization

The rocks present within the GAC property range in age from Cambrian to lower Cretaceous age. They are mostly clastic sediments (sandstone, siltstone, and shale) with some minor carbonates. They originated from the Aïr Massif which has been continuously eroded since at least the Mesozoic period. The sediments were laid down in a continental setting and are generally the result of fluvial and deltaic deposition. In this environment, large shallow rivers meander across flat topography and create complex flow patterns where the coarse-grained sands and gravel are concentrated in the channels with the highest flow energies, while low energy flow regimes on the floodplains and tidal areas create silt and mudstone-type sediments.

Carboniferous sedimentary formations are the major host rocks for uranium mineralization, particularly in the northern part of the basin.

Uranium mineralization in Niger is located exclusively in sediments of the Tim Mersoï Basin and occurs in almost every important sandstone formation, however not always in economic concentrations and tonnage.

The uranium in many of the deposits of the Tim Mersoï Basin is generally oxidized. Among the primary tetravalent minerals, coffinite is dominant and accompanied by pitchblende and silico titanates of uranium. Uranium hexavalent minerals such as uranophane and meta-tyuyamunite are present in the Imouraren and TGT-Geleli deposits.

1.6. Exploration Status

In February 2008, the government of the Republic of Niger granted GAFC the AE3 and AE4 Exploration Permits. Ongoing exploration work and metallurgical studies have confirmed that the most significant uranium mineralization is located around the Dasa area within the AE3 Exploration Permit. Other uranium occurrences also exist within the AE3 and AE4 Exploration Permits.

GAC has undertaken exploration activities on the Dasa Project since its discovery by GAC in 2010. The Dasa Project area covers an area measuring approximately 10 km along the strike of the Azouza graben by about 2 km. However, drilling has only focused on a small portion of this area.

GAC has undertaken multiple phases of exploration and evaluation programs. These programs have included the following:

- Exploration and resource evaluation drill programs.
- Mapping.
- Geophysical investigations.
- Downhole geophysical logging.
- Geotechnical analysis of drill core.
- Metallurgical sampling and analysis.
- Hydrological studies.
- Baseline environmental work.

In 2011, drilling efforts were realigned to achieve two goals: expand the mineral resource, particularly the deeper higher-grade uranium mineralization, and to understand the geological controls on the distribution of the uranium mineralization.

In June 2012, the Dajy exploration camp was opened, enabling easier access to the entire concession area and drilling sites.

The 2017–2018 drilling program (58-hole) successfully delineated higher-grade mineralization within 300 m of the surface. The drilling was focused in areas of faulting associated with a graben structure – known as the Flank Zone - and has improved the understanding of the distribution of mineralization within the deposit and confidence in the geological model. This has resulted in an improved classification of resources in the Flank Zone from Inferred to Indicated, and the development of a lithological and structural model of the deposit to support the mineralization model.

The 2021-2022 drilling program was focused on in-fill drilling and successfully converted substantial Inferred Resources to Indicated Resources, as well as expanding the overall mining area.

1.7. Mineral Resources

The Dasa Project Mineral Resources were first estimated and reported by CSA Global in April 2017, and then updated in June 2018, updated again in June 2019, and updated again by AMC Consultants in May 2023 (as reported in Section 14 of this Report). The Mineral Resources were estimated by Ordinary Kriging (OK) using a geological model and a 100 ppm eU_3O_8 cut-off grade on the mineralized envelope. All mineralized intervals were flagged and composited to 0.5 m and estimated into 10 m × 10 m × 4 m blocks approximating half the drill density in the central parts of the deposit. The estimate has been completed by AMC Consultants' Principal Resource Geologist, Dmitry Pertel (MAIG) who was an Author and Qualified Person for the Mineral Resource Report. Dmitry Pertel was also the author of the previous Resource Reports prepared by CSA Global.

Information from all main phases of exploration and evaluation and the results of quality assurance/quality control (QA/QC) analysis has been considered to develop this updated Mineral Resource.

Mr. Pertel visited the Dasa Project area in March–April 2017 at the request of GAC. The purpose of the visit was to examine resource definition drilling practices used at Dasa, collect QA/QC data, and to inspect the sample preparation laboratory in Niamey.

Review and analysis of both the historical and recent QA/QC data, procedures and protocols indicated that the quality of data is acceptable to allow Mineral Resources to be reported in accordance with the CIM guidelines. The risk associated with the quality of the data is believed to be low.

GAC provided AMC Consultants with all exploration results completed to date and a complete project database that included drillhole collar coordinates, lithological codes, and analytical information for uranium. Most uranium grades used for estimating resources were calculated from the gamma-logging results (eU_3O_8 values). In addition to the downhole logging results, mineralized intersections from the drill core were sampled and sent for assay analysis to ALS Laboratories in South Africa and Canada.

Geological interpretation and wireframing were completed by AMC Consultants. It included interpretation of the main mineralized bodies based on a nominal cut-off grade of 100 ppm U_3O_8. The interpretation was based on the current understanding of the deposit geology and a lithological model of the deposit, which included wireframe models for all main lithological units as well as the major recognized faults within the Project. Closed wireframe models were generated for each modelled mineralized body.

The OK method was chosen to interpolate uranium grades into a block model. A dry bulk density value of 2.36 tonnes per cubic metre (t/m^3) was calculated following exploration programs and directly assigned to the model.

The Mineral Resources have been classified and reported in accordance with the CIM guidelines. Mineral Resource classification is based on confidence in the adopted sampling methods, geological interpretation, drillhole spacing and geostatistical measures.

Mineral Resources were reported on based on underground mining with a cut-off grade of 1,480 ppm.

The Mineral Resource statement is shown in Table 1-1 below.

Table 1-1: Dasa Mineral Resources (1) with an Effective Date of May 12, 2023.

Category	Tonnes (Mt)	eU_3O_8 (ppm)	Contained eU_3O_8 (Mlb)
Indicated Resources	10.09	4,913	109.3
Inferred Resources	4.45	5,243	51.4

(1) Measured and Indicated Mineral Resources are Inclusive of those Mineral Resources Modified to Produce the Mineral Reserves.

The MRE includes Inferred Mineral Resources that are normally considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. It is reasonably expected that most of the Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued infill drilling.

The current 2023 Mineral Resources were interpreted and modelled within a geological and structural model that was developed in 2019. The structural model was not updated in 2023. All mineralized envelopes were interpreted and controlled by the developed lithological model of the deposit and clipped to the interpreted and modelled fault planes. Additionally, significant additional XRF chemical analyses (4,773 analyses) were completed on the mineralized intercepts to refine the reliability of the gamma logging results. This process has provided greater confidence in both the eU_3O_8 results and the geological confidence, which has enabled a higher classification in several areas of deposit, especially those within the Flank Zone.

1.8. Mineral Reserves

The objectives of the feasibility study (FS) were to:

- Undertake a laboratory test work program to optimise the recovery process for the maximum extraction of uranium from the ore.
- Interpret the laboratory test work into a commercially viable and practical process plant.
- Develop a mine design that could recover the ore body such that the mineral resource could be converted into a mineral reserve estimate.
- Determine a total project cost for the construction of the underground mine, process plant, tailing storage facility and associated infrastructure to support and sustain an operational mine at the Dasa Project site.
- Present an economic analysis of the project to determine a return on investment such that the project could be developed into an operationally profitable mine.
- Present a report that has been undertaken and completed in accordance with NI 43-101 and CIM standards of disclosure.

The laboratory test work was undertaken in three independent pilot plant campaigns with results from each campaign guiding and directing the subsequent campaign. Variations in quantity and type of process recovery consumables was used to determine the optimum recovery of uranium for the most practical equipment selection with the lowest reasonable consumable cost. The final selection of the process followed the principles established in uranium operations in the region which have proven to be successful over the past 50 years.

Mineral Reserves for the Dasa Uranium Project have been estimated based on the geology and Mineral Resource Estimate undertaken for Global Atomic Corporation and as reported in section 14 of this report. An engineering design and costing exercise has been undertaken to a feasibility study (FS) level of accuracy which supports the Mineral Reserve Estimate (section 15 of this report). The FS has addressed all required aspects of the project to enable the estimation of Mineral Reserves and is discussed in detail in section 15.

Detailed and preliminary engineering designs have been undertaken for the underground mine workings, mining surface infrastructure, process plant, tailings storage facility, and support services infrastructure. These designs enabled detailed pricing enquiries to be issued to the market in the development of a comprehensive capital cost and sustaining cost estimate. Manning and consumable material requirements were developed and costed in the open markets to establish an expected operating cost over the life of mine of the operation. Sourcing of electrical power and water was determined to meet the mines requirements, and these too, contributed to the operational cost estimate.

The capital cost estimate, sustaining cost estimate and operational cost estimates for the various elements of the mine and process plant were combined into an economic analysis of the project to determine a financial model for the mine.

The feasibility study has been completed at a detailed level of design and engineering to enable an appropriate level of confidence to be applied to the economic viability and outcomes of the project.

All Mineral Reserves Estimated are in the Probable category as reported in Table 1-2.

Table 1-2: Mineral Reserve Estimate for Dasa Uranium Project (28 February 2024).

Mineral Reserve Category	RoM (Mt)	U_3O_8 (ppm)	U_3O_8 (kt)	U_3O_8 (Million lbs)
Proven Mineral Reserve	-	-	-	
Probable Mineral Reserve	8.05	4,113	33.1	73.0
Total Mineral Reserve	**8.05**	**4,113**	**33.1**	**73.0**

Notes on Mineral Reserves

- Mineral Reserves are reported with an effective date of 28th February 2024.
- Mineral Reserves are reported using the 2014 CIM Definition Standards.
- Inferred Mineral Resources, although included in the mining inventory, are excluded from the Mineral Reserve Estimate.
- Measured and Indicated Mineral Resources are inclusive of those Mineral Resources modified to produce the Mineral Reserves.
- The Qualified Person responsible for Mineral Reserve Estimate is A. D. Pooley.
- Differences between tonnes, grade and contained metal content are due to rounding.

1.9. Mining Methods

The selected mining method for the Dasa orebody is a transverse long-hole open stoping (LHOS) method with a cemented hydraulic fill. The method is fully mechanised, and an appropriate fleet of mining equipment has been included in the design.

The identified mining areas will be accessed by a single decline developed from surface, with a gradient of 8 degrees, located in the footwall of the orebody. Access to the stoping blocks will be at 22.5 m vertical intervals with a footwall drive developed along strike 20 m from the stopes. Stope access crosscuts will be developed at 16.5 m intervals off the footwall drive. Key geotechnical design criteria and modifying factors used in the mine design are detailed in section 16 of this report.

The mine ventilation system is a key aspect of the mine design due to the presence of radioactive elements in the air. The ventilation system designed is a once through system (no recirculation of air) and will replace the volume of air in the mine on average every 15 minutes. Ventilation of excavations within the orebody where radiation risk is higher is by use of an exhaust system which removes contaminated air from the workings immediately into the return airway system, ensuring that risk relating to exposure to radiation is always minimised.

Standard mechanized underground mining equipment is proposed and will comprise electro-hydraulic long hole face drilling rigs and modern ground support drilling rigs. Proposed rock handling equipment will comprise diesel powered 14-tonne Load Haul Dump (LHD) units and 42-tonne articulated dump trucks.

Ancillary equipment will consist of diesel-powered charge-up vehicles, utility vehicles and other light vehicles such as Integrated Tool Carrier (ITC) units, man-carriers, Front End Loader (FEL), mobile rock-breaker, maintenance utility vehicle, and crane.

Stoping operations are envisaged to utilize an electro-hydraulic long-hole production drill unit capable of drilling accurate holes up to 35 metres in a fan ring pattern which will be fired on a retreat basis. Blasted mineralized material will be mucked using a tele-remote LHD rated at 14 tonnes, loading into either 42-tonne haul trucks or temporary storage bays placed along the access level such that tramming distance is optimal. Stope voids will be backfilled with a cemented backfill produced in the backfill plant situated in the process plant and will use tailing material from the process plant.

Mineralized broken material and excess broken waste will be transported via the ramp and main decline system to surface in 42-tonne haul trucks for dumping at surface stockpiles or waste dump storage facilities near the mine portal. The waste rock produced from the development of the mine will either be used in the construction of the dry stack tailing storage facility or disposed of underground by tipping into the mined out secondary stopes. All waste produced will be deposited at one or other of these places and no permanent waste rock dump will be required. RoM pad stockpiles will be blended to obtain the desired feed grade required by the process plant. Four distinctive stockpiles are envisioned, graded from very low to high grade.

To determine the optimum stope shapes, the design input parameters as defined in the mine design criteria were used as input into DeswikSO® software. This is a mineable shape optimizer which seeks to create stope shapes, based on pre-set input parameters, while optimising the extraction of ore from the resource. A primary input to the DeswikSO® process is specification of the cut-off grade which was determined to be 1,500 ppm as detailed in section 15.

A summary of the Phase 1 planned output and expected plant throughput is presented in Table 1-3.

Table 1-3: 2024 FS Project Summary.

Project Overview	Unit	Value
Mining		
Total mined	,000's t	8,047
Metal mined	eU_3O_8 M lb	73.0
Average mining grade	eU_3O_8 ppm	4,113
Mine life		
Ramp-up	months	36
Years at steady state	years	23.75
Average production rate	k tpm	28.2
Metallurgical		
Total processed	,000's t	8,047
Metal processed	eU_3O_8 M lb	73.0
Average processed grade	eU_3O_8 ppm	4.113

Project Overview	Unit	Value
Metallurgical recovery	%	93.4
Payable metal	eU_3O_8 M lb	68.1
Payable metal per tonne processed	eU_3O_8 lbs/t	8.46

In accordance with recommendations of the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM"), if Inferred Mineral Resources are used in the development of mine plans and production schedules, they should be treated as waste materials. The mine plan includes 10,708 tonnes of Inferred Mineral Resources (0.13%) of the mineral inventory mined which is treated as zero grade waste material dilution.

1.10. Recovery Methods

Test work undertaken by Insight R&D included 3 pilot plant campaigns to determine the optimum process to recover the maximum uranium from the ore body using conventional process plant equipment and lowest consumable costs to produce a saleable product.

Ore from the underground mining operation is delivered by haul trucks to surface and deposited onto stockpiles of differing grades or the ore is directly tipped into the crusher feed bin if the grade is within a predetermined limit. A front-end loader will feed the crusher from the stockpiles such that the blend over a period is within the predetermined limits. Ore delivered to the crusher is reduced in size and delivered to a mill feed stockpile by conveyor belt – the stockpile providing a buffer capacity to ensure feed to the mill is constant.

Reclaim feeders under the stockpile feed ore via a conveyor belt to the SAG Mill, where the ore is dried and milled – the milling process includes screening and ore recirculating back to the SAG Mill until a final product size of 0.6 mm is attained. Ore discharging from the milling process reports to a revolving pug drum where sulphuric acid and nitric acid are introduced to start the uranium extraction process. The retention time in the pugging drum is approximately 15 minutes before the ore discharges and reports via a feed conveyor to the curing conveyor belt.

The curing conveyor belt is a slow-moving wide conveyor that transports the ore over a total distance of 300 m (150 m on the outgoing leg and 150 m on the return leg). The ore cures for approximately 3 hours on the curing belt where the uranium recovery is further enhanced. Ore discharges from the curing belt and is fed to the leach section where it discharges into re-pulping tanks and then pumped to the leach tanks where additional sulphuric acid is added to enhance the dissolution process.

The slurry solution in the leach circuits flows through a 5-tank gravity cascade system and is pumped to the horizontal belt filters. The slurry is distributed across 2 horizontal vacuum belt filters where the solution containing the uranium is separated from the solids (tailing). The belt filter solids (tailings) discharge on a conveyor belt that can distribute the tailing solids to either the back fill plant or the tailing storage stockpile in a "one or the other" process. The back fill process will consume 50% of the total tailing product in a batch process, and when the back fill plant does not require tailings, then the total tailing stream will report to the tailings stockpile.

The uranium containing solution from the horizontal vacuum belt filter is pumped via the pregnant leach solution tanks to the solvent extraction plant. At the solvent extraction plant, the solution will report to the 4-stage extraction section where kerosene and other reagents are used for the purification of the uranium. The next step in the solvent extraction plant is the scrubbing process (3-stage) that minimises the transfer of organic to the stripping stage where the remaining organic is removed using sodium carbonate in a 3-stage process.

The uranium bearing solution is then pumped to the precipitation section where the uranium is precipitated from the solution as sodium di-uranate. Precipitation of the uranium is achieved through a 5-tank cascade process at elevated temperatures with the addition of caustic soda. The separation of the solids from the liquid fraction is then undertaken in a further horizontal vacuum belt filter with the uranium reporting as the solids. Discharge from the belt filter is then dried and packed into drums in an automatic drum filling process.

Drums are loaded onto pallets and the pallets are loaded into containers for export to customers.

Reagents used in the recovery process are delivered in a range of packaging types (dry powder to liquids) and through several dedicated reagent make-up plants are made up into useable and easy to dose concentrations for application at specific points in the recovery process.

1.11. Project Infrastructure

Mining and process plant surface infrastructure will include:

- Access and site roads.
- Water storage, evaporation ponds, and water treatment facilities.
- Boreholes to provide raw water to the process plant, complete with electrical supply, pipes, and pumps.
- Administration offices, training facilities, medical facilities, change house, canteen, security guard house and access control.
- Sewage treatment plant and waste disposal site.
- Communications and IT infrastructure.
- Plant and mining vehicle workshops and repair workshops, vehicle wash bays, and tire changing facilities.
- Fuel and oil stores for both the process plant and the mining equipment.
- Consumable, reagent, and spares stores in appropriate open and closed facilities.
- A 700-man accommodation camp complete with canteen and cooking facilities, laundry and administration facilities and dry and cold storage facilities.
- Tailing storage facility construction.
- Compressor houses and infrastructure to accommodate the mine ventilation fans.

1.12. Environment and Community Impact

The Dasa Project is located within a sparsely uninhabited region characterized by the presence of small villages and three larger, permanent villages (Agatara, Teguef Nakh and Tagaza) are located along the highway which runs approximately 5 kms west of the Project site. The region is characterized by an arid intermediate climate of the Sahelian desert type with two distinct main seasons: the dry season between October and May and the wet season from June to September.

In its 2020 Environmental and Social Impact Assessment (“ESIA”), Niger environmental consultancy; Groupe Art & Genie identified a 7 km radius area of influence zone and a 15 km radius extended area of influence zone around the mine site. In its 2022 ESIA, Niger environmental consultancy; Firme d’Expertise en

Environnement et Développement ("FEED Consult") conducted additional biological baseline studies and consultation with area villages and regional authorities. Global Atomic completed an ESIA Addendum Report in early 2023 which was designed to incorporate information from the 2023 revision of the Feasibility Study, as well as to summarize both the government approved 2022 ESIA and the 2022 FEED Consult ESIA.

Villages proximal to the 7 km perimeter are inhabited on a seasonal basis in support of nomadic livestock herding and seasonal farming referred to as "market gardening" within and proximal to the koris (ephemeral watercourses) located outside the 7 km perimeter, where water is available on a seasonal basis.

A desktop study of the Project and surrounding areas was recently completed in accordance with the requirements of the International Finance Corporation's Performance Standard 6 ("IFC PS6"). The purpose of the desktop study was to identify biodiversity features of interest and conduct critical habitat ("CH") and Species of Conservation Concern (SoCC) screening and inform on-going biodiversity field work. Several Protected Areas were identified, but none of them overlap the Project Area.

Biodiversity surveys undertaken pursuant to the 2020 and 2022 ESIA's did not indicate the presence of threatened flora or fauna species or areas of Natural or Modified Habitat within the landscape study area. A Critical Habitat Desktop Study was conducted by Treweek Environmental Consultants Ltd. and Abel Geospatial Consulting Ltd. in 2023.cmapping. Based on GIS spatial assessment, desktop critical habitat screening, historical and recent fieldwork, there is no critical habitat in the Project's area of influence (defined as a 50 km radius around the mine site). However, to align with IFC PS6 guidelines, a local environmental consultancy group has been commissioned to conduct further field assessment and habitat mapping focused on the biodiversity features identified in the Critical Habitat Desktop Study.

Global Atomic has been engaging with local communities since its arrival in the Dasa Project area in 2008. Engagement has included informal engagement with village elders and the development of community support programs which cover the following areas:

- Food security.
- Medical support.
- Infrastructure.
- Local business support and procurement.
- Regional and national procurement.

Additional future development support programs will be delivered in collaboration with NGO's currently active in Niger and provide targeted benefits to women including enhanced irrigation, training and support of existing market gardening initiatives, support for development of goods and services related to workers apparel and PPE and associated education, training, and mentoring programs. Corporate Social Responsibility contributions will be reviewed with reference to the success of projects to date and priorities identified in consultation with communities.

In 2020, as part of the ESIA undertaken for the national permitting process, formal consultations took place in the communities around the Project area, including Tagaza, Agatara, Issakanan, Sikiret/Tadant, Oufound, Mizeine, Ghalab, the Kelezeret Tribe and Inolamane.

FEED Consult carried out additional engagement in the local villages as part of the ESIA conducted in 2022. The 2022 engagement also included the Governorate, the Regional Council, the Regional Director of Mines, the Regional Directorate for the Environment and the Fight against Desertification, the Regional Directorate for the Advancement of Women and Child Protection, the Regional Directorate of Hydraulics and Sanitation, the Regional Labour Inspectorate, and the Regional Directorate of Livestock. At the Departmental level, the Town Hall, and the Prefecture as well as the villages listed above were consulted.

During the 2022 and earlier consultations, participants raised various environmental and social concerns regarding the Project, which the ESIAs have aimed to address.

Over the course of 2023, SOMIDA continued consultations with the parties listed above and expanded the geographical scope of consultations to include communities within a 30 km circle of the Dasa Project. SOMIDA also shares the results of its local consultation program and its wider social programs with government authorities in the urban centers of Agadez, Tchirozérine, Danet and Arlit and, regional and national Government Ministries.

The principal aim of SOMIDA's consultation program is to keep local people informed as to the progress of the Project, encourage community involvement in the Project through local employment and subcontracting and provide a forum for concerns to be expressed.

GAC and now SOMIDA has been supporting local communities through various Community Social Relations ("CSR") programs since 2008, as summarized in the table below which also shows anticipated increased levels of support and new programs through the construction phase and mining operations. Support programs will be evaluated on an on-going basis through the operations and closure phases of the Project.

1.13. Capital and Operating Costs

METC Engineering together with Bara Consulting and Epoch Resources have undertaken a combination of detailed and preliminary designs and after obtaining quotations and pricing from the open market and from data base sources a capital and operating cost estimate was determined. Manpower and operating costs in line with the anticipated consumable costs and manpower histograms were developed and incorporated into the financial model, the outputs of which are included in the financial analysis.

The capital costs are summarised in Table 1-4 below.

Table 1-4: Feasibility Study CAPEX Summary.

Capital Costs	Initial ($ M)	Sustaining Capital ($ M)	Total ($ M)	$/lb U_3O_8	$/feed tonne
Mining	58.8	218.7	277.5	4.07	34.48
Processing	83.2	38.9	122.1	1.79	15.17
Infrastructure	68.2	5.2	73.4	1.08	9.12
Total Direct Capital Costs	210.2	262.8	473	6.94	58.77
Indirect and Owner's Costs	60.9	30.0	90.9	1.33	11.30
Total (including Indirect Costs)	271.1	292.8	563.9	8.27	70.07
Contingency	37.2	29.9	67.1	0.99	8.34
Reclamation	0	15.9	15.9	0.23	1.97
TOTAL CAPITAL	**308.3**	**338.6**	**646.9**	**9.49**	**80.38**

Due to rounding, some columns and rows may not total exactly as shown.

Capital costs in Table 1-4 exclude $67.2 million already incurred to December 31, 2023

The total capital cost for the Dasa mine is $308 million (including $37 million for contingencies) with a further $339 million for sustaining costs over the life of mine (including $30 million for contingencies). The sustaining costs include provision of mine development cost, major equipment replacement, tailings storage facility and reclamation. These items include mechanized mining equipment and major processing plant equipment components.

The detailed capital costs are presented in section 21.6.10 for the mining and in Table 21-13 Process Plant Capital Cost Summary.

Operating Costs

The mining operating costs for the feasibility study project are $9.10/lb U_3O_8 based on an owner-operated model. Ramp access development and mining costs are capitalized prior to the start of the processing plant and expensed as a component of operating costs thereafter. A contract miner was engaged to assist in training and supervision of mining activities. The SOMIDA personnel have now been trained to be self sufficient on the new mining equipment and a new group of miners is being trained on this and maintenance. The contract miner will also be assisting in training of grade control personnel and stope drilling and mining. It is expected that SOMIDA will be fully owner operated in 2026. The mine will have a total compliment of 280 people arranged on a day shift, night shift and "off" shift basis. Shift duration will be 10 hours from start to finish.

Average process plant operating costs are calculated to be $10.00/lb U_3O_8 with the largest contributors to operating costs being sulphuric acid and electricity costs. Electricity costs are based on grid power rates supplemented by a full complement of diesel power generators that will provide power when grid power is not available or sufficient. The processing facility will be operated and maintained by a staff of 213 people on a 3-shift rotation basis. The shift pattern is 2 × 12 hr shifts with a relief shift. The process plant will operate 24 hours per day, 365 days per year The Operating Cost Estimate Summary is presented in Table 1-5.

Table 1-5: Operating Cost Estimate Summary.

Description	Total Cost ($ million)	Average Annual ($ million)	Unit Cost ($/t processed)	Unit Cost ($/lb U_3O_8)
Mining Cost	620.2	26.1	77.08	9.10
Processing Cost	681.5	28.7	84.69	10.00
General and Administration	443.7	18.7	55.15	6.51
Total Cash Costs Before Royalties	**1,745.4**	**73.5**	**216.92**	**25.62**

Notes:

1 – Total may not sum due to rounding.

2 – Average annual costs based on 23.75 years.

General and Administration (G&A) costs include a 700-person camp and facilities, camp staffing and offsite costs. Site support staff number 131 persons plus 194 outside contractors for camp services and security. An additional 24 persons are forecast for the Niamey office support. General and Administration (G&A) and offsite cash operating cost totals $6.51/lb U_3O_8.

Including sustaining capital and royalty costs, the all-in sustaining costs (AISC) is $35.47/lb U_3O_8 ($300/t processed). All-in sustaining cost is a non-GAAP measure. There are no historic operations for comparison and the cost has been built up from cost estimates as shown in Table 1-11.

Ore grades are highest in the early years when the Graben area is being mined, as shown in Figure 1-1 ore tonnes and grade mined.

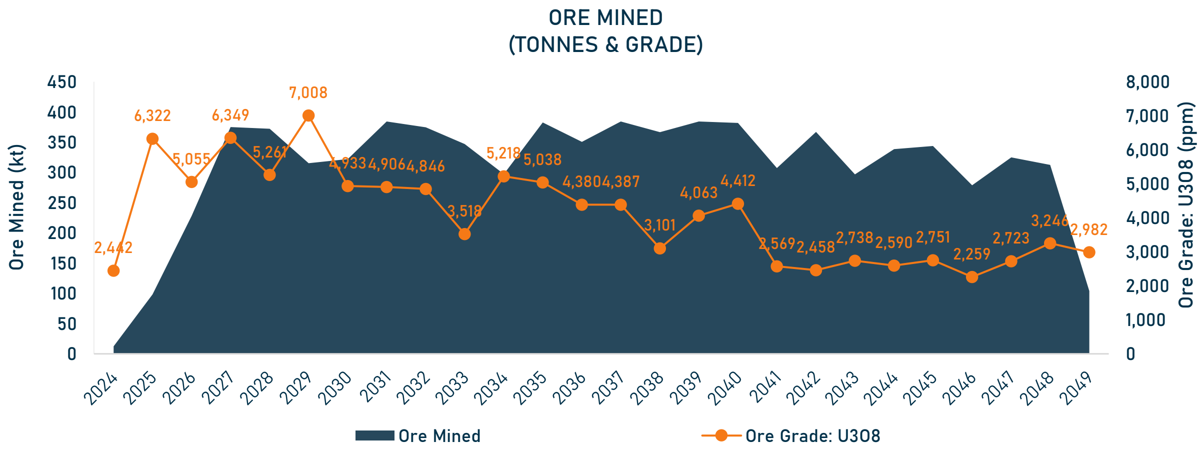

Figure 1-1: Ore Mined (Tonnes and Grade) Over Life of Mine.

Maintaining a processing plant throughput of 1,000 tonnes per day, the processed ore grades follow a similar pattern.

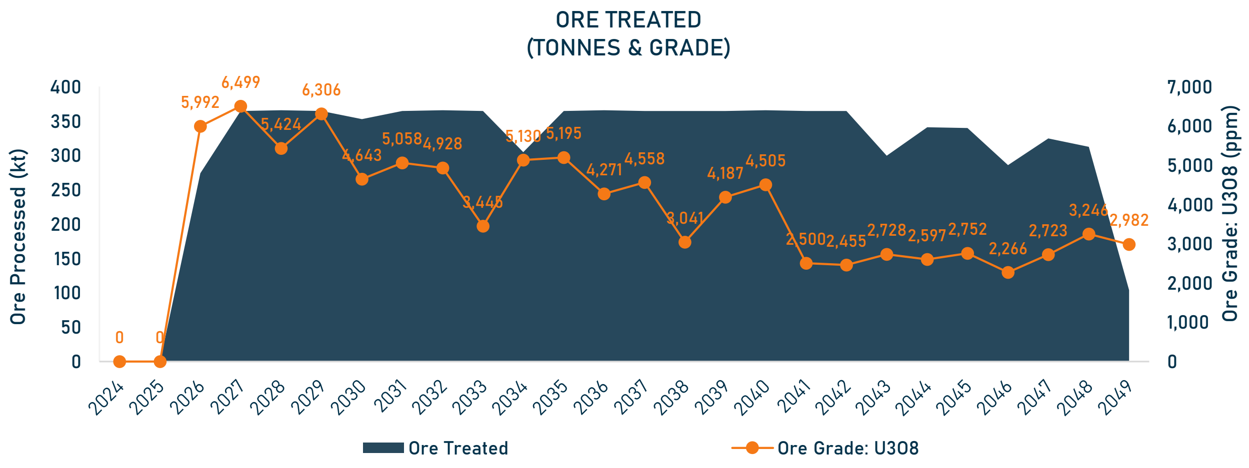

Figure 1-2: Ore Treated (Tonnes and Grade) Over Life of Mine.

Following on the grade profile of ore processed, U_3O_8 recovered exceeds 4 million pounds per annum in the early years and drops to about 2 million pounds per annum in the later years.

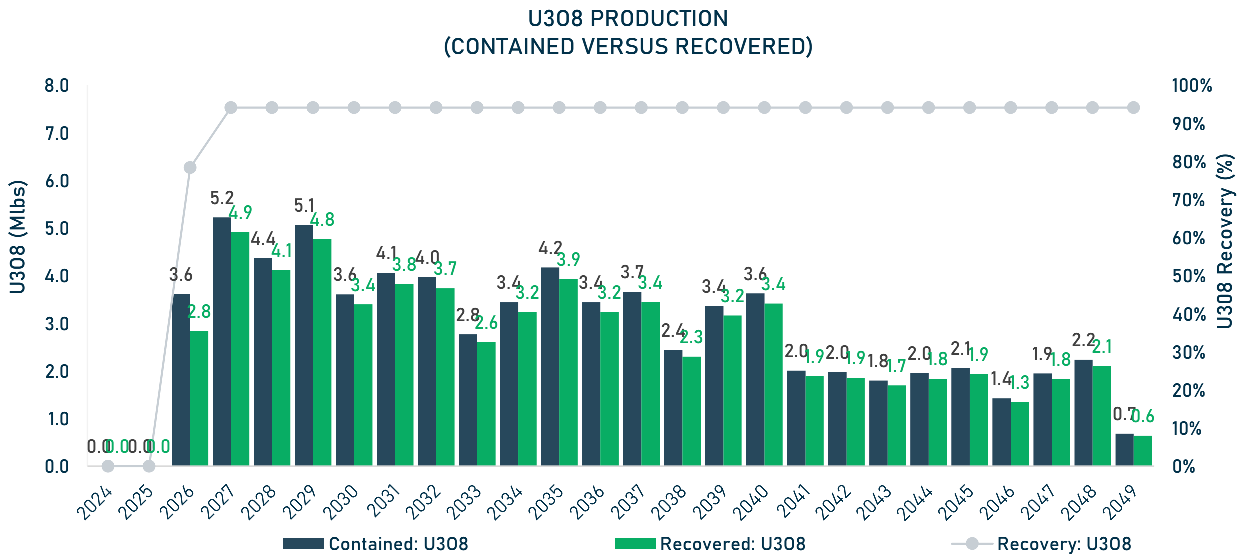

Figure 1-3: U_3O_8 Production Over Life of Mine.

The unit production costs increase over time as, a result of these trends.

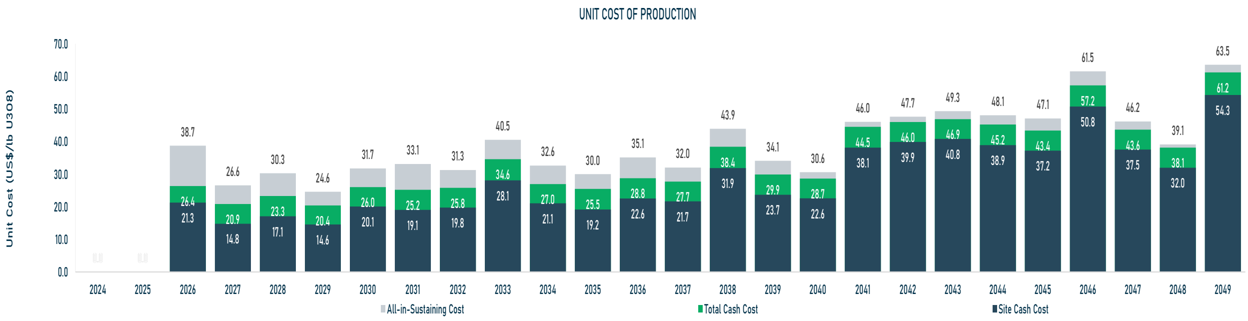

Figure 1-4: Unit Cost of Production Over Life of Mine.

Enhancement of throughput and possible mill expansions will be investigated to improve and maintain the processing plant output. In view of substantial fixed operating costs for the mill and site infrastructure, achieving such targets will significantly lower the unit operating costs over time. Additional infill drilling is expected to also move a large portion of Inferred Resources to the Indicated Resource category so these can be included in the mine plan.

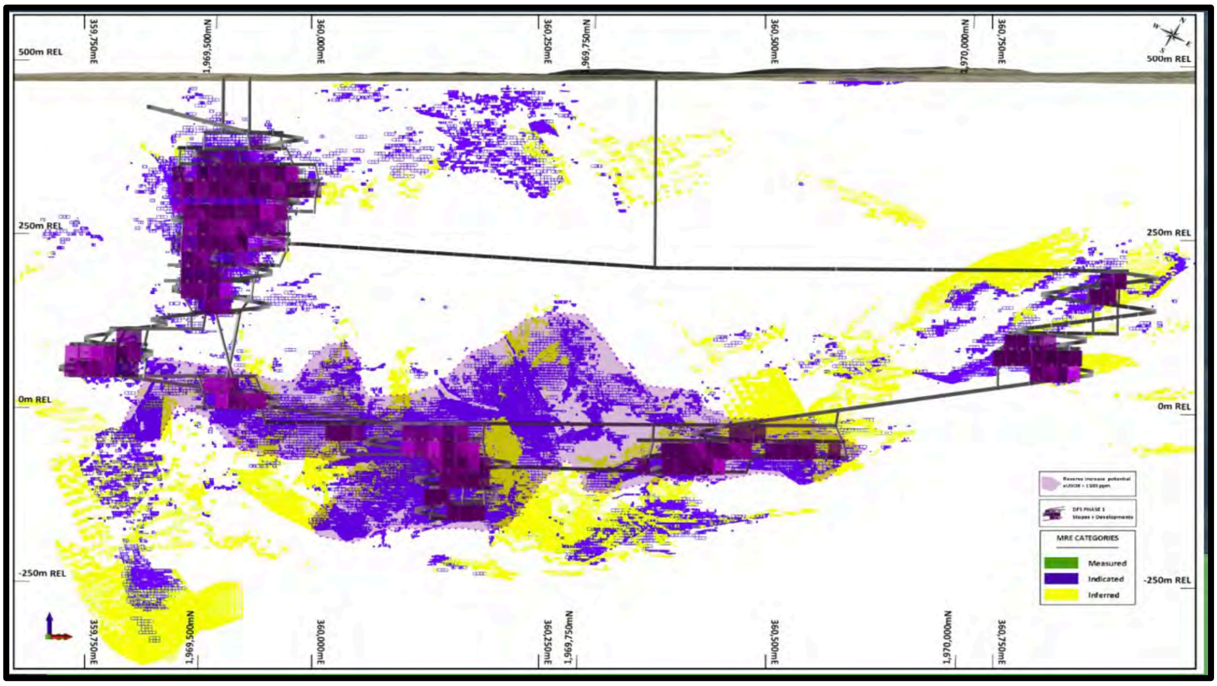

Figure 1-5: Indicated, Inferred and Measured Resources in Mining Lease Area.

The Inferred Resources have been estimated at 4.45 million tonnes grading 5,243 ppm for an estimated 51.4 million pounds U_3O_8. Comparing the first 12-years of the current Study to the Phase 1 Study provides a better comparison of unit cost trends during periods of similar high grades.

Table 1-6: Comparison of Cost Trends.

	Phase 1 Study 2026-2037	2024 Study 2026-2037	2024 Study 2026-2049
Mine plan (years)	12	12	23
Uranium price ($/lb)	$35	$75	$75
Average mill feed grade (ppm)	5,267	5,103	4,113
Uranium recovered (Mlb)	44.1	44.1	68.1
Average cash cost before royalty ($/lb)	$15.72	$20.35	$25.62
Average cash cost[1] ($/lb)	$19.02	$25.38	$30.73
Average AISC[2] ($/lb)	$22.13	$31.49	$35.47
Internal After-tax Rate of Return ("IRR")	22.3%		57.0%
After-tax Net Present Value ("NPV$_8$") ($ million)	$147		$917
Pay-back (years)[3]	3.25		2.2

- [1] Cash cost per pound represents mining, processing, on-site and offsite general and administrative costs, selling expenses and royalties, divided by recovered U_3O_8.
- [2] All-in sustaining cost per pound of uranium represents mining, processing, site and offsite general and administrative costs, royalties and sustaining capital expenditures including rehabilitation provision, divided by recovered U_3O_8.
- Pay-back is based on total cost, including amounts already paid.

1.14. Economic Analysis

A forward-looking discounted cash-flow model was developed in MS Excel to assess the economic potential of the project. The detailed model is presented in Table 22-6.

The model accounts for the following line items:

- Revenue.
- Mining royalties.
- Operating costs.
- Taxes, and
- Capital costs.

The output from the model is the free cash flow, which is the revenue less the sum of the other items. Other metrics, such as the cash cost, the all-in sustaining cost, the NPV and the IRR are calculated.

The model is based on monthly mine and process plant production as forecast in Chapter 16 for the underground mining plan.

Assumptions are as follows:

- The sale price for U_3O_8 is $75/lb U_3O_8.
- Measures of units for mine production are metric tonnes, and for uranium are pounds U_3O_8.
- All values are in 2024 values, no inflation has been included.
- All monetary values refer to US dollars.

Net present value figures in this Report are calculated using an 8% discount rate and cash flows are discounted to January 1, 2026, the start of the processing plant, with undiscounted capital costs deducted therefrom.

Revenue is estimated from the production plan and the $75/lb U_3O_8 except for those quantities already contracted, which are priced at contract prices. The mine production has been forecast so that surface ore stockpiles represent approximately 3 to 4 months inventory once operations stabilize in late 2026. The processing plant begins operations at the start of 2026 with an 11 month ramp up to steady state, at which time, processing plant recovery is estimated to be 94.15%. The plant is assumed to have an operating capacity of 1 000 t/d of run-of-mine feed based on approximately 86% availability from the processing plant that has been designed at 1,200 tonnes per day throughput. Excess mined material is stockpiled for later processing.

The Government of Niger collects a mining royalty that is proportional to operating profitability. The royalty rate so determined is applied to revenues. Where the royalty rate was previously based on a sliding scale, the royalty rate under the new Mining Code is now a fixed 7%.

The corporate income tax rate in Niger is 30%. A three-year grace period is provided from start of commercial production.

Exploration and development expenses are depreciated at a rate of 20%, plant at 10% and infrastructure at 5%. An initial balance of depreciable assets of $129.4 million is included in the calculation.

Niger has a value-added tax system (VAT) with a rate of 19%. The project is exempt from VAT until commencement of production. Under the previous Mining Code, the VAT rate for uranium was set at 0%. Once regulations under the new Mining Code have been finalized, it is expected there will be a similar exemption for VAT or at least a recovery upon export of U_3O_8.

The mining and processing capital estimates have been prepared according to a Class 3 estimate as defined by the AACE. The direct and indirect costs contributing to the initial and sustaining capital costs are given below.

Table 1-7: Initial and Sustaining Capital Expenditures.

Item	Amount, 000 $
Mining	58,836
Processing	80,248
Owners' costs & infrastructure	132,087
Contingency	37,196
Total Initial Capital Expenditure	308,367
Sustaining and closure costs	338,550
Total Initial plus Sustaining Capital expenditure	**955,284**

Initial capital expenditures include contingencies ranging from 5% to 25%, for an overall average contingency of 14.1%. Sustaining capital expenditures include a contingency of 15% on mine development and mine capital expenditures.

1.15. Economic Results at $75/lb U_3O_8

The forecast of the economics of the Dasa Project are shown in Table 1-8 as follows.

Table 1-8: Forecast Project Economics.

	Total	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Ore mined (000 Tonnes)	8,046.6	12.3	97.9	228.0	375.0	372.2	315.0	322.1	384.0	374.5	346.7	297.3	382.4	350.3
Ore Processed (000 Tonnes)	8,046.6			273.9	364.6	365.6	364.6	352.7	364.6	365.6	364.6	304.5	364.6	365.6
Grade (ppm)	*4,113*			*5,992*	*6,499*	*5,424*	*6,306*	*4,643*	*5,058*	*4,928*	*3,445*	*5,130*	*5,195*	*4,271*
Contained U3O8 (million lbs)	73.0			3.6	5.2	4.4	5.1	3.6	4.1	4.0	2.8	3.4	4.2	3.4
Recovery	*93.4%*			*78.4%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*
Production U3O8 (million lbs)	**68.1**			**2.8**	**4.9**	**4.1**	**4.8**	**3.4**	**3.8**	**3.7**	**2.6**	**3.2**	**3.9**	**3.2**
Revenues	5,041.4			169.6	359.0	311.2	335.8	243.3	284.3	272.5	207.7	230.4	303.6	244.5
Mining costs	620.2			17.9	23.8	22.3	20.9	21.8	25.8	26.8	27.9	23.2	28.2	26.9
Processing costs	681.5			25.0	32.2	31.1	32.0	29.9	30.7	30.6	29.0	28.8	30.8	29.9
Site & Niamey costs	375.6			17.4	16.9	16.9	16.6	16.6	16.6	16.5	16.3	16.3	16.3	16.3
Selling expenses	68.1			2.8	4.9	4.1	4.8	3.4	3.8	3.7	2.6	3.2	3.9	3.2
Royalties	348.1			11.7	24.8	21.5	23.2	16.8	19.6	18.8	14.4	15.9	21.0	16.9
EBITDA	**2,947.8**			**94.7**	**256.4**	**215.3**	**238.3**	**154.8**	**187.8**	**175.9**	**117.4**	**143.0**	**203.4**	**151.2**
Initial capital expenditures	297.6	167.1	130.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Sustaining capital expenditures	333.4			45.0	27.1	27.7	18.8	18.6	29.4	19.6	14.8	17.7	16.8	19.9
Closure fund payments	15.9			0.6	1.2	1.0	1.1	0.8	0.9	0.9	0.6	0.7	0.9	0.8
Working capital	-30.7			16.3	1.9	-11.3	9.5	-1.9	2.3	6.3	-17.0	8.6	-5.0	1.2
Pre-tax cash flow	**2,331.6**	**-167.1**	**-130.5**	**32.9**	**226.3**	**197.9**	**208.9**	**137.3**	**155.2**	**149.1**	**119.0**	**116.0**	**190.6**	**129.4**
Income taxes	492.1			0.0	0.0	0.0	0.0	0.0	31.6	43.7	26.3	35.3	54.6	39.7
Post-tax cash flow	**1,839.5**	**-167.1**	**-130.5**	**32.9**	**226.3**	**197.9**	**208.9**	**137.3**	**123.7**	**105.5**	**92.7**	**80.6**	**136.0**	**89.7**

	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049
Ore mined (000 Tonnes)	384.0	366.6	384.0	381.8	307.1	366.8	296.8	338.6	343.6	278.6	324.6	312.5	103.7
Ore Processed (000 Tonnes)	364.6	364.6	364.6	365.6	364.6	364.6	299.5	341.0	339.6	285.8	324.6	312.5	103.7
Grade (ppm)	*4,558*	*3,041*	*4,187*	*4,505*	*2,500*	*2,455*	*2,728*	*2,597*	*2,752*	*2,266*	*2,723*	*3,246*	*2,982*
Contained U3O8 (million lbs)	3.7	2.4	3.4	3.6	2.0	2.0	1.8	2.0	2.1	1.4	1.9	2.2	0.7
Recovery	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*
Production U3O8 (million lbs)	**3.4**	**2.3**	**3.2**	**3.4**	**1.9**	**1.9**	**1.7**	**1.8**	**1.9**	**1.3**	**1.8**	**2.1**	**0.6**
Revenues	253.1	184.5	238.2	251.5	147.5	138.7	124.7	141.1	144.7	105.9	135.8	156.2	57.8
Mining costs	28.2	29.0	29.6	31.3	28.4	30.4	27.0	28.3	28.9	26.9	28.0	27.4	11.3
Processing costs	30.2	28.6	29.8	30.2	28.0	27.9	26.5	27.5	27.6	25.8	27.2	27.4	14.9
Site & Niamey costs	16.3	15.8	15.8	15.7	15.7	15.7	15.7	15.7	15.7	15.6	13.7	12.6	8.7
Selling expenses	3.4	2.3	3.2	3.4	1.9	1.9	1.7	1.8	1.9	1.3	1.8	2.1	0.6
Royalties	17.5	12.8	16.4	17.4	10.2	9.6	8.6	9.7	10.0	7.3	9.4	10.8	4.0
EBITDA	**157.4**	**96.1**	**143.4**	**153.5**	**63.3**	**53.2**	**45.1**	**57.9**	**60.6**	**29.0**	**55.7**	**75.9**	**18.3**
Initial capital expenditures	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
Sustaining capital expenditures	14.1	12.2	12.5	5.9	2.4	2.6	3.8	4.8	6.7	5.5	4.3	1.6	1.5
Closure fund payments	0.8	0.6	0.7	0.8	0.5	0.4	0.4	0.4	0.4	0.3	0.4	0.5	0.2
Working capital	4.4	-8.3	0.7	2.7	-6.8	0.7	1.8	-2.4	0.3	-3.6	1.2	2.3	-34.4
Pre-tax cash flow	**138.1**	**91.6**	**129.5**	**144.2**	**67.2**	**49.5**	**39.2**	**55.0**	**53.1**	**26.7**	**49.9**	**71.5**	**51.0**
Income taxes	41.3	23.2	37.3	41.3	14.6	12.8	11.0	15.6	16.7	7.5	14.3	21.0	4.3
Post-tax cash flow	**96.8**	**68.4**	**92.1**	**102.9**	**52.5**	**36.7**	**28.2**	**39.5**	**36.4**	**19.2**	**35.6**	**50.5**	**46.7**

Table 1-9: Forecast of the Project Economics Over the Dasa Project.

Item	LoM Total, $M
Revenue	5,041
Mining royalties	348
Operating costs	1,745
Operating income	2,948
Taxation	492
Capital costs, closure fund and working capital	616
Free cash flow	1,839

Note: values may not add up due to rounding.

Table 1-10: Financial Indicators Over the Dasa Project.

Item	Unit	Value
NPV_8 after tax	$M	$917
IRR after tax	%	57.03%
Payback period from start of production	years	2.2
Cash flow (before capex)	$M	$2,948
Free cash flow	$M	$1,840
Operating costs		
Cash cost	$/lb U_3O_8	30.73
All-in sustaining cost (1)	$/lb U_3O_8	35.47
Capital costs		
Initial capital costs	$M	$308
Sustaining capital costs	$M	$339

(1) All-in sustaining cost is a non-GAAP measure. There are no historic operations for comparison and the cost has been built up from cost estimates as shown on Table 1-11.

The cash cost per pound of U_3O_8 represents the sum of the costs of mining, processing, mining royalties and site and offsite general and administrative costs, divided by the pounds of recovered U_3O_8. The all-in sustaining cost per pound of U_3O_8 represents the sum of the costs of mining, processing, mining royalties, site and offsite general and administrative costs and the sustaining capital expenditures, divided by the pounds of recovered U_3O_8.

The sensitivity of the NPV_8 and the IRR to the changes in the input estimates to the financial model is discussed in section 22 Economic Analysis of this report – the model is most sensitive to the price of U_3O_8. as shown in the following table:

Table 1-11: Economic Sensitivity with Varying Price of U3O8.

U_3O_8 price (per pound)	**$60**	**$75**	**$90**	**$105**
Before-tax NPV_8	$656 M	$1,122 M	$1,572 M	$2,022 M
After-tax NPV_8	$551 M	$917 M	$1,269 M	$1,621 M
After-tax IRR	38.2%	57.0%	74.8%	92.9%

On February 28, 2024, the effective date of this report, the U_3O_8 price was $95/lb. The lower end of the sensitivities has been used throughout this report.

1.16. Interpretation and Conclusions

The Project's exploration data and work completed to date is of an appropriate standard, allowing the estimation of a reliable Mineral Resource Estimate (MRE) for the Project's Dasa uranium deposit based on the full lithological model of the deposit intersected to date.

Additional mineralization exists along strike beyond the current resource.

Based on the MRE and the geological interpretation provided by GAC, geotechnical and mining assessments have been undertaken which demonstrate that it is technically feasible to extract the identified Mineral Resources using well known and understood extractive techniques. The outcome of these assessments was an initial, Phase 1 mine with an approximate 12-year life, producing approximately 33,000 tonnes per month with a high percentage extraction.

The test work pilot plant campaigns show that the use of pugging and curing as a metallurgical recovery process can produce a high recovery rate with acceptable operational costs.

The economic analysis for the Project indicates that the project is economically feasible for extraction of the mineralized material entirely by underground means, even within the low uranium price regime of \$35/lb U_3O_8.

The Feasibility Study has determined that the Dasa Project is viable and should proceed to production.

Risks

A review of the main Project risks identified that radiation exposure, potential ground water inflow, and geotechnical variances are potential risks at the underground mine.

Opportunities

The main Project opportunities are the following:

- The Project presents a low probability of a negative NPV in the current uranium market.
- There is good potential to expand resources at Dasa, along strike and at depth.
- Additional drilling should result in conversion of substantial Inferred Resources to Indicated Resources and subsequent inclusion in the mine plan; U_3O_8 pounds in the Inferred Resource category equate to 47% of the U_3O_8 pounds in the Indicated Resource category, although they bear much greater realization risk.
- The cut-off grade for the mine plan was based on a U_3O_8 price of \$70/lb; use of a higher price would likely increase the economically mineable tonnes and the overall reserve quantum.
- The mining dilution rate was set at 10% in the current Study compared to 5% in the Phase 1 Study; reducing dilution would have a material effect on the project economics.
- Process plant availability was set at 86% based on estimated power interruptions; the Dasa Project will have a full back-up power generation and should achieve availability close to the 92% realized by the other uranium processing plants in Niger.
- The Dasa deposit is in a very arid desert area with limited flora and fauna and with very limited population. These conditions are favourable for mining development.
- Further optimization work should translate into lower costs.
- Most mill components have been over-sized by 20%, which could enable throughput to be increased to 1,325 tpd with limited additional capital investment.
- Reopening of the Benin transport route would reduce operating costs included in the forecasts.

Recommendations

The Dasa Project has demonstrated its economic viability through the extensive technical work completed to date. The Qualified Persons and METC Engineering support the Project's construction and uranium production.

2. INTRODUCTION

2.1. Introduction

Mr. Dmitry Pertel, M.Sc., MAIG, Mr. Andrew Pooley, BEng (Hons) Mining, and Mr. John Edwards, BSc (Hons) Mineral Processing Technology have prepared an NI 43-101 Technical Report (the Report) on the Dasa Uranium Project (Dasa or the Project) for Global Atomic Corporation (GAC). The Dasa project is located in the north central part of the Republic of Niger, West Africa, and approximately 95 km north of the city of Agadez. The country is bordered by Algeria and Libya to the north, Chad to the east, Nigeria and Benin to the south, and Burkina Faso and Mali to the west.

2.2. Issuer

GAC is a Toronto Stock Exchange (TSX) listed mineral exploration and development company based in Toronto, Ontario, Canada. GAC, through its wholly owned subsidiary, Global Atomic Fuels Corporation (GAFC, founded in 2005), has been successfully investigating the uranium potential of six permits currently covering approximately 750 km^2 in the Agadez region of central Niger.

GAC's mineral assets in Niger occur in two project areas – Adrar Emoles and Tin Negoran. Uranium mineralization has been identified on each of the permit areas, with the most significant discovery being the Dasa deposit, discovered in 2010 by GAC geologists through grassroots field exploration on the Adrar Emoles 3 Exploration Permit.

Exploration and evaluation programs completed to date are sufficient to estimate Mineral Resources and the Dasa Project was the subject of a now historical Preliminary Economic Assessment (PEA) reported in 2018 that was subsequently updated in 2020, and a Mineral Resource estimate (MRE) update in 2019 that was subsequently updated in 2023, and a Phase 1 Feasibility Study (FS) reported in 2021 and amended in 2023. Other Exploration Permit areas have also been explored and have demonstrated potential for uranium mineralization which may result in additional Mineral Resources for the Project with additional work.

2.3. Terms of Reference

The Report has been prepared in support of disclosures in Global Atomic Corporation's news release dated March 5, 2024, entitled "Updated Dasa Project Feasibility Study".

In August 2020, GAC appointed METC Engineering Pty LTD (METC) to undertake a Feasibility Study and NI 43-101 Technical Report on the Dasa Project with the objective of moving the project into development in 2022. At the end of 2021, based on the results of the Phase 1 Feasibility Study, GAC determined that the technical feasibility and commercial viability of the Dasa Project had been demonstrated and the GAC Board of Directors approved proceeding with development of the Dasa Project. In September 2022, GAC engaged

Development Consultants Private Limited (DCPL) to complete engineering for the Dasa Project development and Lycopodium Minerals Canada Ltd. (Lycopodium) to prepare a project execution plan, project schedule and initiate the procurement process. In June 2023, GAC engaged METC Engineering (METC) to provide project management, procurement, project controls and construction management services for the completion of the Dasa Project. In November 2023, GAC engaged METC to complete an updated feasibility study based on the May 2023 updated Mineral Resource Estimate and the updated mine plan, capital, and operating costs for the Dasa Project.

METC is an engineering house specialising in the design of mineral processing plants for the mining industry. METC was established in 2017 and is headed by a team of process plant and engineering specialists with a combined experience in excess of 200 years in the mining industry. Headquartered in Johannesburg, South Africa, the METC team currently has several projects and studies in various countries (predominantly Africa, but also Europe and the Caribbean).

METC appointed specialist companies Bara Consulting and Epoch Resources to undertake the Mining and Tailing Storage Facility (TSF) respectively. Insight R&D were independently engaged by GAC to undertake the test work to determine an appropriate recovery process and expected uranium recovery, as well as being the owners' representative for the engineering, procurement, and construction management (EPCM) of the Dasa Project. The primary purpose of this document (the "Report") is to update the previous feasibility study to a feasibility study level at a AACE Class 3 accuracy based on the EPCM process, including an update to the Mineral Reserve estimates based on the updated May MRE.

This Report is based on information known to the authors and METC Engineering and includes: the outcomes of the exploration and evaluation programs completed by GAC at the Project, the 2023 MRE (AMC, 2023), the EPCM work to date by DCPL and METC, and the feasibility study completed by METC up to and including February 28, 2024 (the "Effective Date").

The 2024 Feasibility Report replaces the previous 2021 Feasibility Report reported in the National Instrument 43-101 (NI 43-101) Technical Report.

The Report is specific to the standards dictated by NI 43-101 (30 June 2011), companion policy NI 43-101CP, and Form 43-101F1 (Standards of Disclosure for Mineral Projects). The MRE used in this Feasibility Study was previously reported in a May 23, 2023, press release with an effective date of 12 May 2023 (AMC 2023) and has been prepared in accordance with CIM Definition Standards for Mineral Resources and Mineral Reserves (10 May 2014).

METC Engineering acted independently as GAC's consultant and was paid fees based on standard hourly rates for the services provided. The fee was commensurate with the work completed and was not contingent on the outcome of the work. Neither METC, Bara or Epoch nor any of their staff rendering the services in connection with this Report, had any material, financial or pecuniary interest in GAC or its subsidiaries, or in the Project.

Units used in the report are metric units unless otherwise noted. Monetary units are in United States dollars (US$) unless otherwise stated.

2.4. Qualified Person Section Responsibility

The sections of this Report were prepared by or under the supervision of the Qualified Persons identified in Table 2-1.

Table 2-1: Qualified Person Section Responsibility.

Section	Section title	Qualified Person(s)
1	Summary	All
2	Introduction	All
3	Reliance on Other Experts	John Edwards
4	Property Description and Location	John Edwards
5	Accessibility, Climate, Local Resources, Infrastructure and Physiography	John Edwards
6	History	Dmitry Pertel
7	Geological Setting and Mineralization	Dmitry Pertel
8	Deposit Types	Dmitry Pertel
9	Exploration	Dmitry Pertel
10	Drilling	Dmitry Pertel
11	Sample Preparation, Analyses and Security	Dmitry Pertel
12	Data Verification	Dmitry Pertel
13	Mineral Processing and Metallurgical Testing	John Edwards
14	Mineral Resource Estimates	Dmitry Pertel
15	Mineral Reserve Estimates	Andrew Pooley
16	Mining Methods	Andrew Pooley
17	Recovery Methods	John Edwards
18	Project Infrastructure	John Edwards and Andrew Pooley
19	Market Studies and Contracts	John Edwards
20	Environmental Studies, Permitting, and Social or Community Impact	John Edwards
21	Capital and Operating Costs	John Edwards and Andrew Pooley
22	Economic Analysis	John Edwards and Andrew Pooley
23	Adjacent Properties	Dmitry Pertel
24	Other Relevant Data and Information	All

25	Interpretation and Conclusions	All
26	Recommendations	All
27	References	John Edwards

2.5. Qualified Person Property Inspection

The METC Engineering, Qualified Person, John Edwards, undertook a site visit to the Dasa exploration camp and the deposit between 5 December and 12 December 2020, spending five days at the deposit site and the exploration camp, and 1 day in the office in Niamey. Mr. Edwards inspected core logging and storage facilities, quality assurance/quality control (QA/QC) protocols and procedures, local geology of the deposit, reviewed sample preparation techniques and visited the laboratory in Niamey.

The authors and METC Engineering consider Mr. Edwards' 2020 site visit to be current under Section 6.2 of NI 43-101. Another visit was not warranted as the same procedures were in place and there was no material changes or new areas under investigation at the time of the feasibility study.

2.6. Sources of Information

Sources of information, data and reports reviewed as part of this Technical Report can be found in Section 27 (References). The authors take responsibility for the content of this Technical Report and believe the data reviewed to be accurate and complete in all material respects.

Author John Edwards acquired and relied upon mineral titles information on the Mining Permit, Niger Mining Code, Niger Royalties, Niger Income Taxes, Environmental Studies, Social or Community Impacts, Market Studies and Contracts from GAC. This Report partly relies on information provided by GAC and others, including documents, data and reports compiled by GAC management, consultants, contractors, and their own technical staff. METC was supplied the results of previous work completed by GAC and others. The previous work included geological reports covering the results of the drilling programs compiled in a digital database including both probe results and chemical assays, geophysical surveys (surface and downhole) and the results of previous MREs.

Author Dmitry Perterl acquired the primary dataset used to inform the Mineral Resource from the digital drillhole database provided by GAC and subsequently reviewed the data, including completion of relevant QA/QC checks, and is satisfied the data is adequate for estimation of Mineral Resources.

Author John Edwards acquired the process test work prepared by Insight R&D as summarized in various test Reports.

The authors have reviewed the information provided and believe it is accurate.

3. RELIANCE ON OTHER EXPERTS

Section Figure 4-1: Location of Dasa Mining Permit, AE3 and AE4 Exploration Permits of GAC Source: Global Atomic GIS (2021) – ownership information has been provided by Global Atomic Corporation - the information has not been independently verified.

Section 4.3. Permitting Considerations – the permitting status has been provided by GAC – the information has not been independently verified.

4. PROPERTY DESCRIPTION AND LOCATION

4.1. Location of Property

The Dasa Uranium Project (Dasa or the Project) is located in the central part of the Republic of Niger, West Africa and lies within the IN-BOUKATT Mining Permit area, a carve out of the Adrar Emoles 3 Exploration Permit area, which with the contiguous Adrar Emoles 4 Exploration Permit, were 100% owned by Global Atomic Fuels Corporation (GAFC), a wholly owned subsidiary of GAC, and form part of a larger package of properties in Niger in which GAFC has held a 100% interest. The Exploration Permits expired in December 2023 and while GAFC has applied for an extension or renewal, there is no certainty that GAFC will be successful. The IN-BOUKATT Mining Permit is held by a Niger corporation, Societe Miniere de DASA SA (SOMIDA). The shares of SOMIDA are owned 80% by GAFC and 20% by the Republic of Niger.

The region is largely uninhabited and is characterized by nomadic villages and three small towns located on the highway west of the Project area that links the regional towns of Arlit 105 km to the north and Tchirozerine 60 km and Agadez 95 km respectively to the south.

The centre of the Dasa Project is positioned at longitude 7.8° East and latitude 17.8° North within the IN-BOUKATT Mining Permit, which has a total area of 25 km^2. Under NI 43-101 guidelines, the Adrar Emoles 3 (AE3) and Adrar Emoles 4 (AE4) Exploration Permits, subject to a successful extension or renewal application, are considered to be the same Property as they would reasonably share common infrastructure should a mineral deposit be developed on either concession. The AE3 Exploration Permit covers an area of 96.2 km^2 and the AE4 Exploration Permit which adjoins the southern border of the AE3 Exploration Permit has an area of 122.4 km^2.

The Project area is accessible by an all-weather road connecting Agadez, Niger's second largest city, located 95 km south of the Project with the mining town of Arlit some 105 km north of the area of interest, and the capital, Niamey, approximately 1,000 km to the southwest.

There are two airports serving the project general area: The Mano Dayak airport at Agadez, which was recently upgraded and has a 3,000-metre runway, and the country's international airport in the capital city of Niamey. There are regular charter flights and daily connections between Agadez and Niamey.

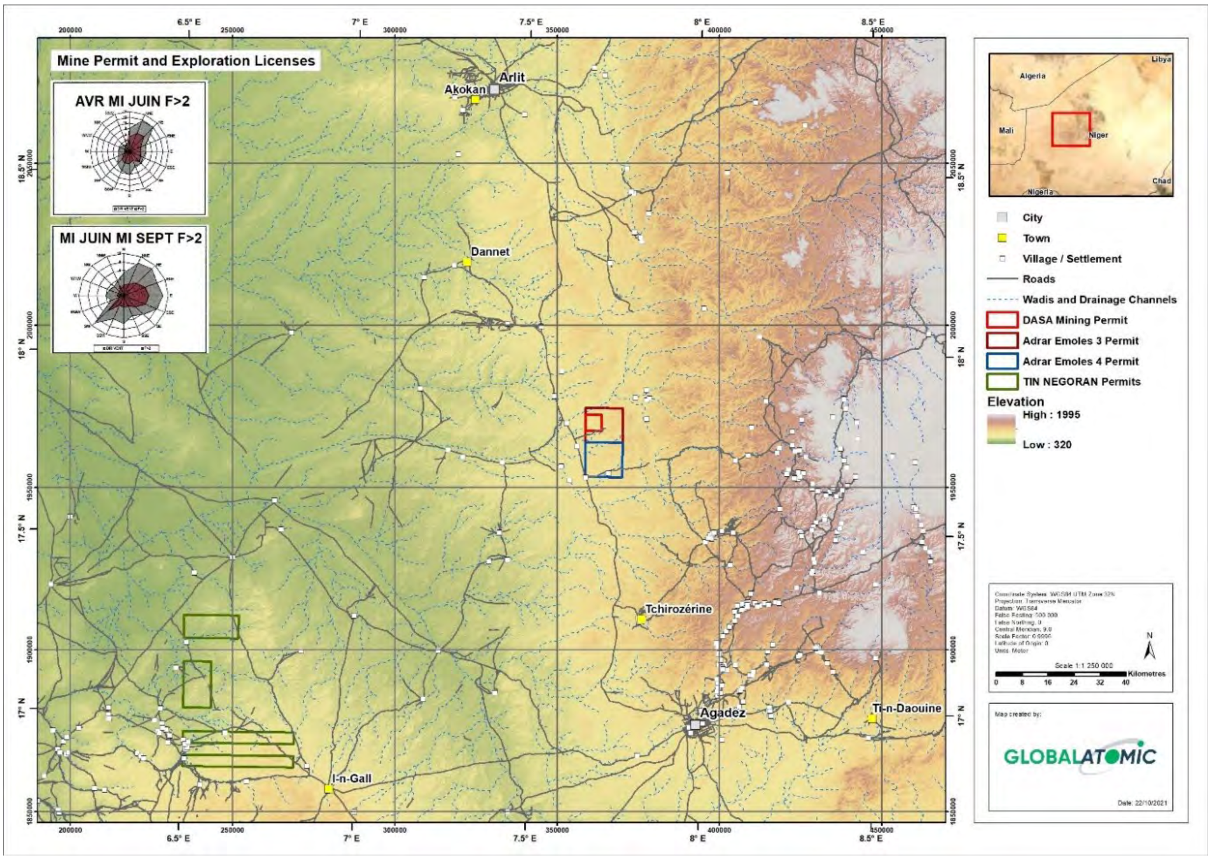

Figure 4-1: Location of Dasa Mining Permit, AE3 and AE4 Exploration Permits of GAC Source: Global Atomic GIS (2021).

4.2. Mineral Tenure

Under the former Mining Code, Exploration Permits and Mining Permits were granted within the provisions of a Mining Convention negotiated between the Ministry of Mines and the applicant. Such a Mining Convention covered a period of up to 20-years, being the exploration period (three years plus two 3-year renewals) and the first 10-year validity period of a Mining Permit. The Mining Convention is then renegotiated at each renewal of the Mining Permit. The Mining Convention can only be amended upon the mutual consent of both parties. The Convention is approved by Decree of the Council of Ministers and then signed by the parties and stipulates rights and obligations of the parties during the validity period.

GAC, through its 100% owned subsidiary GAFC, entered, into and holds a 100% interest in two Mining Conventions named AE3 and AE4 as of 25 September 2007. Each Agreement covered an initial area of approximately 500 km^2.

The Exploration Permits for AE3 and AE4 were granted on 8 February 2008 for the first three-year period on perimeters defined to include approximately 488.7 km^2 and 492.5 km^2, respectively. On 16 November 2010, the Exploration Permits for AE3 and AE4 were extended by the Minister of Mines. The first three-year renewal of the AE3 and AE4 Exploration Permits were received on 17 January 2013, concurrent with the required 50% reduction in area to approximately 243.7 km^2 and 246 km^2, respectively. The second renewal was granted on

29 January 2016, reducing the AE3 and AE4 areas to approximately 121.2 km^2 and 122.4 km^2, respectively. The AE3 Exploration Permit area was further reduced to 96.2 km^2 with the carve out of 25 km^2 for the Mining Permit on its granting.

Both the AE3 and AE4 Exploration Permits were extended on 17 December 2018 for an additional two years, extending to 29 January 2021 and extended again on January 21, 2021, for the period ending December 17, 2023. Application has been made to further extend or renew the AE3 and AE4 Exploration Permits. There can be no assurance that GAFC will be successful in this regard.

GAC, through its 100% owned subsidiary GAFC also holds the Tin Negoran Exploration Permits which are located approximately 100 kms south-west of the Dasa Project, cover an area of 486.2 km^2 and were extended on January 21, 2021, for the period ending December 17, 2023. Application has also been made to further extend or renew the Tin Negoran AE3 and AE4 Exploration Permits. There can be no assurance that GAFC will be successful in this regard.

The Tin Negoran Exploration Permits have been the target of over 22,000 metres of drilling by Global Atomic. All six Exploration Permit Areas lie within the Tim Mersoï Basin which has produced uranium for the Republic of Niger for the last 50 years.

On July 5, 2022, the Republic of Niger enacted a new Mining Code. Under the new Mining Code, the Mining Convention is specific to the Mining Permit and no longer covers the Exploration Permits. Exploration Permits are granted as Specifications (Cahier de Charges) for a period of 4-years plus two 3-year renewal periods. On discovery of a mineral resource, the holder of an Exploration Permit must complete a feasibility study and may apply for a Mining Permit. Mining Conventions are now specific to the Mining Permit and specify the terms and conditions under which exploitation activities must operate. The Mining Convention is initially granted for 10 years and is renewable every 5 years until the resource is depleted.

Under the new Mining Code, existing Mining Conventions are grandfathered. Accordingly, GAC will operate under the existing Mining Convention until its expiry on September 25, 2027, at which time, it will be renewed under the new Mining Code.

4.3. Permitting Considerations

On December 23, 2020, the Republic of Niger Ministry of Mines granted a Mining Permit to GAFC on behalf of a Niger mining company to be incorporated by GAC for the Dasa Project. The Mining Permit has an initial term of 10 years and is renewable for successive 5-year terms, until the resource has been fully depleted.

On August 10, 2022, a Niger mining company, Societe Miniere de DASA S.A. (SOMIDA), was incorporated and on November 3, 2022, GAFC completed the transfer of the Mining Permit to SOMIDA. Under the Nigerien Mining Code, the Republic of Niger has the right to a 10% carried interest in the common shares of the Niger mining company and may subscribe for up to an additional 30% in the common shares of the Niger mining company up to the date of the incorporation of the Nigerien mining company, provided it commits to fund its proportionate share of capital costs and operating deficits for such additional interest. On the incorporation of SOMIDA, the Republic of Niger Ministry of Mines elected to subscribe to a 10% interest in SOMIDA in addition to the free carried interest, for a total 20% interest in the share capital of SOMIDA.

On January 28, 2021, GAFC received its Certificate of Environmental Conformity for the Dasa Project from the Republic of Niger Ministry of Environment, Urban Health, and Sustainable Development.

GAFC has received all permits and approvals required for the development and commercial production of the Dasa Project.

With the granting of the Mining Permit, the area of the Mining Permit was removed from the perimeter of the AE3 Exploration Permit. The Dasa Mining Permit has an initial 10-year period to December 23, 2030, and is then renewable every 5-years until the resource is depleted.

Exploration expenditures and corporate overhead costs incurred in Niger related to the Dasa Deposit by GAFC to the date of granting of the Mining Permit in the amount of US$ 54.9 million was transferred to SOMIDA on its incorporation and will be reimbursed to GAC from Dasa Mine operations.

The current status including area and geographic coordinates for the Dasa Mining Permit and the AE3 and AE4 exploration permits is summarized in Table 4-1 below.

Table 4-1:Mining Permit and AE3 and AE4 Exploration Permits.

Dasa Mine Permit			**Adrar Emoles 3 Exploration Permit ***			**Adrar Emoles 4 Exploration Permit**		
Tenement Type: Mining Permit Company: SOMIDA Date granted: 23/12/2020. Valid: 10 years subject to 5-year renewals until Deposit is depleted. Area: 25 km^2			Tenement type: Exploration Company: GAFC Date granted: 21/01/2021. Valid: to 17/12/2023 Area: 96.2 km^2			Tenement type: Exploration Company: GAFC Date granted: 21/01/2021. Valid: to 17/12/2023 Area: 122.4 km^2		
Point	Longitude east	Latitude north	Point	Longitude east	Latitude north	Point	Longitude east	Latitude north
A	7°39'59.82''	17°50'08.14''	A	7°40'00''	17°51'14''	A	7°40'00''	17°45'30''
B	7°42'49.81''	17°50'08.14''	B	7°46'28''	17°51'14''	B	7°46'28''	17°45'30''
C	7°42'49.81''	17°47'26.15''	C	7°46'28''	17°45'30''	C	7°46'28''	17°39'43''
D	7°39'59.82''	17°47'26.15''	D	7°40'00''	17°45'30''	D	7°40'00''	17°39'43''

**The coordinates shown for the Adrar Emoles 3 Exploration Permit represent the boundaries within which the Mining Permit has been granted.*

The Dasa Project is subject to royalties payable to the Niger Government. Current tenure provisions are adequate to conduct exploration, development, extraction, and processing activities on the Project. At the time of writing, the QP is not aware of any environmental liabilities on the Project or any other factors that would affect access or the rights to work on the Project.

4.4. Other Significant Factors and Risks

Environmental, permitting, legal, title, taxation, socio-economic, marketing, and political or other relevant issues could potentially materially affect access, title or the right or ability to perform work on the Property. However, as of the Effective Date of this Report, the QP is unaware of any such potential issues affecting the Property.

4.5. Environmental and Social Impact Assessment

In 2011, GAC commissioned Niger engineering and environmental consultancy Groupe Art & Genie to complete an Environmental Characterization Study ("ECS") and establish environmental and socio-economic baseline information for the Dasa Project. GAC also retained Groupe Art & Genie to conduct hydrology and hydrogeology studies during the period 2012 - 2016.

In 2020, GAC retained Groupe Art & Genie to conduct an Environmental and Social Impact Assessment ("ESIA") for the Dasa Project Phase I Mine Plan. The ESIA updated previous environmental and socio-economic baseline information in support of the Company's Mining Permit application for the Dasa Project. GAC engaged with the Ministry of Environment and Sustainable Development ("MESUDD") to confirm the Terms of Reference and scope of both the ESIA and the Environmental and Social Management Plan ("ESMP") at the start of the ESIA process.

The 2020 ESIA served to update the ECS completed by Groupe Art & Genie in 2011 and included hydrological and air quality baseline data compiled during the interim period, as well as addressing environmental, social, and economic impacts associated with construction of the mine, mining operations and reclamation and closure.

The ESIA included an analysis of alternative design and operating scenarios to reduce environmental and social impacts associated with the Mine. A key element of the ESIA was public consultation to inform stakeholders of the proposed Project, and technical studies related thereto. Community meetings were held in area villages and small towns as part of the ESIA process and were organised by MESUDD and attended by Government and GAC personnel. GAC has maintained regular engagement with surrounding communities since the start of exploration activities in 2008 and provides on-going support to these communities in the areas of water supply, food security, healthcare, education and training, local business support and procurement.

A key component of the ESIA is the ESMP. The ESMP governs the Company's activities from construction through operations and mine closure and establishes the protocols for Project monitoring and reporting to government ministries and agencies. The ESMP also outlines the management procedures, mitigation, monitoring, and reporting protocols that will be used to avoid, reduce, and manage environmental and social impacts during the construction, operations, and closure phases of the Dasa Project.

The ESIA was submitted to the MESUDD for review in October 2020 in accordance with Law No. 98-56 of 29 December 1998 on the framework law on environmental management and Law No. 2018-28 of 14 May 2018 setting out the fundamental principles of environmental review. The MESUDD approved the ESIA and ESMP in November 2020, and officially advised the Ministry of Environment and Ministry of Mines that GAC had successfully completed the ESIA process and was eligible to receive a Mining Permit for the Dasa Project. The Mining Permit was issued by Presidential Decree effective December 23, 2020. The Mining Permit was issued for an initial period of ten years, subject to automatic five - year renewals until the deposit is depleted.

GAC also entered into two agreements with the MESUDD in December 2020: the “Partnership Agreement”, which establishes the framework for the implementation and monitoring of the ESMP, and capacity building and budgets for the relevant ministries and agencies and, the Cahier des Charges ("CCES") (Environment and Social Charges Book), which establishes implementation, monitoring and reporting protocols related to the ESMP. The BNEA issued its final authorization, the Certificate of Environmental Conformity, on January 23, 2021. The Dasa Project has received all Permits required to commence construction and operations.

GAC is committed to undertake its operations in line with the Equator Principles (“EP4”), an international financial industry benchmark for determining, assessing, and managing environmental and social risks. Pursuant to the above, in 2022 GAC commissioned a new ESIA by Firme d'Expertise en Environnement et Development (“FEED Consult”), a Nigerien environmental consultancy and subject-area specialist.

In 2023, GAC compiled an ESIA Addendum report (“Addendum”), which was designed to incorporate information from the 2023 revision of the Feasibility Study, as well as to summarize both the government - approved 2020 ESIA and the FEED Consult ESIA.

The Addendum summarizes environmental and social management measures put in place to ensure the Project is undertaken in accordance with both government requirements and good international industry practice (“GIIP”) to include the International Finance Corporation Performance Standards on Environmental and Social Sustainability ("IFC PS"), IFC Environmental, Health and Safety ("EHS") Guidelines, and the guidance of the International Atomic Energy Agency ("IAEA"), of which Niger is a member state.

4.6. Environmental Considerations

The Dasa Project is located within a largely uninhabited region characterized by nomadic villages and three small towns located on the highway east of the Project area that links the regional towns of Arlit 105 km to the north and Tchirozerine 60 km and Agadez 95 km respectively to the south.

Groupe Art & Genie identified a 7 km radius area of influence zone and a 15 km radius extended area of influence zone around the mine site. The 7 km zone is the area within which the main mine infrastructure is located and therefore the area that will be most affected by environmental and social changes during the construction, operating and reclamation phases of the Project. The 7 km perimeter is considered the Project safety perimeter. The 15 km zone is defined as an extended area of influence and includes two towns located on the Agadez – Arlit Highway.

Villages proximal to the 7 km perimeter are inhabited on a seasonal basis in support of nomadic livestock herding and seasonal farming referred to as “market gardening” within and proximal to the koris (ephemeral watercourses) where water is available on a seasonal basis. The towns of Tagaza and Egatrak are on or proximal to the 7 km perimeter and located on the Agadez - Arlit Highway.

A desktop study of the Project and surrounding areas was recently completed in accordance with the requirements of the International Finance Corporation's Performance Standard 6 (IFC PS6). The purpose of the desktop study was to identify biodiversity features of interest and conduct critical habitat (CH) and Species of Conservation Concern (SoCC) screening and inform on-going biodiversity field work.

The previous ECS and ESIA identified a limited number of biodiversity features of interest including Protected Areas in the region, species which may be present in the Project area and habitats / vegetation known to occur.

The desktop study employed the Integrated Biodiversity Assessment Tool (IBAT) to procure both an IBAT report for the Project Area and to download IBAT GIS data for Protected Areas. GIS data included the World Database of Protected Areas (WDPA), Key Biodiversity Areas (KBA), IUCN Species Ranges (for Critically Endangered (CR), Endangered (EN) and Vulnerable (VU) species), a review of UNESCO World Heritage Sites, Man and Biosphere (MAB) Reserves, Ramsar Wetland sites, the UNEP Global Critical Habitat screening layer and a review of IUCN Red list species based on IBAT results. Several Protected Areas were identified through reference to the above data bases however none of them overlap the Project Area.

A Critical Habitat screening was carried out in accordance with PS6 to identify biodiversity features that might potentially meet the CH criteria and thresholds. A Landscape Study Area (LSA) was identified to include the Dasa mining project location and a 50 km buffer around it. More detailed screening was then undertaken within a more localised study area (the Dasa mining project location with a 15 km buffer).

Twenty-four species of conservation concern have been identified as potential CH qualifying features based on the limited habitat and site-specific information currently available for the LSA. Of these, 13 species have been conservatively classified as "may occur" and 11 species are considered "unlikely to occur" with a higher degree of confidence. Three wide ranging/nomadic mammals: North-west African Cheetah (*Acinonyx jubatus ssp. hecki*), Addax (*Addax nasomaculatus*) and Dama Gazelle (*Gazella dama/ Nanger dama*), have the potential to qualify the landscape study area as Critical Habitat under Criterion 1 (Critically Endangered and Endangered species).

Previous project reports did not confirm the presence of threatened flora or fauna species or areas of Natural or Modified Habitat within the landscape study area. However, to align with IFC PS6 guidelines, a local environmental consultancy group has been commissioned to conduct further field assessment and habitat mapping focused on the biodiversity features identified in the Critical Habitat Desktop Study.

4.7. Social Considerations

GAC and now SOMIDA has been engaging with local communities since its arrival in the area in 2008. In 2020, as part of the ESIA undertaken for the national permitting process, formal consultations took place in the communities around the Project area, including Tagaza, Agatara, Issakanan, Sikiret/Tadant, Oufound, Mizeine, Ghalab, the Kelezeret Tribe and Inolamane.

FEED Consult carried out additional engagement in the local villages as part of the ESIA conducted in 2022. The 2022 engagement also included the Governorate, the Regional Council, the Regional Director of Mines, the Regional Directorate for the Environment and the Fight against Desertification, the Regional Directorate for the Advancement of Women and Child Protection, the Regional Directorate of Hydraulics and Sanitation, the Regional Labour Inspectorate, and the Regional Directorate of Livestock. At Departmental level, the Town Hall, and the Prefecture as well as the villages listed above were consulted.

During the 2022 and earlier consultations, participants raised various environmental and social concerns regarding the Project, which the ESIAs have aimed to address.

Over the course of 2023, SOMIDA continued consultations with the parties listed above and expanded the geographical scope of consultations to include communities within a 30 km circle of the Dasa Project. SOMIDA also shares the results of its local consultation program and its wider social programs with government authorities in the urban centers of Agadez, Tchirozérine, Danet and Arlit and, regional and national Government Ministries.

The principal aim of SOMIDA's consultation program is to keep local people informed as to the progress of the Project, encourage community involvement in the Project through local employment and subcontracting and provide a forum for concerns to be expressed.

GAC and now SOMIDA has been supporting local communities through various Community Social Relations ("CSR") programs since 2008, as summarized in Table 4-2 below which also shows anticipated increased levels of support and new programs through the construction phase and mining operations. Support programs will be evaluated on an on-going basis through the operations and closure phases of the Project.

Table 4-2: Summary of Community Support Initiatives Undertaken and Planned.

Global Atomic Corp - CSR / ESG	Exploration														Construction			Ops
	2008	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025
Food																		
millet			x	x	x	x	x	x	x	x	x	x	x	x	x		x	x
sugar			x	x	x	x	x	x	x	x	x	x	x	x	x	x	x	x
rice			x	x	x	x	x	x	x	x	x	x	x	x	x		x	x
Medical																		
ambulance										x								
supplies										x				x	x	x	x	x
food										x				x	x		x	x
Covid													x					
Infrastructure																		
roads								x	x	x	x	x	x	x	x	x	x	x
water well - local / herding								x		x		x		x		x	x	x
water well - Camp / community use												x			x	x	x	x
water well - agricultural supply												x			x		x	x
Environment																		
EISA and baseline studies / inventory		x	x										x	x	x	x	x	x
project area inventory													x	x	x	x	x	x
re-vegetation initiatives														x	x	x	x	x
mitigation programs																x	x	x
Education / Training																		
education - exploration			x	x	x	x	x	x	x	x	x	x	x	x	x	x	x	x
training - exploration			x	x	x	x	x	x	x	x	x	x	x	x	x	x	x	x
training - mining apprenticeship															x	x	x	x
traininig - environment														x	x	x	x	x
agriculture - training / support																x	x	x
Local Business Support / Procurement																		
agriculture														x	x	x	x	x
food services														x	x	x	x	x
micro business - community			x	x	x	x	x	x	x	x	x	x	x	x	x	x	x	x
camp supply				x	x	x	x	x	x	x	x	x	x	x	x	x	x	x
Regional / National procurement																		
exploration drilling	x	x	x	x	x	x	x	x			x	x	x	x	x		x	x
road work					x									x	x	x	x	x
camp site development / maintenance					x	x	x	x	x	x	x	x	x	x	x	x	x	x
food services			x	x	x	x	x	x	x	x	x	x	x	x	x	x	x	x
water wells install / maintain					x	x	x	x	x	x	x	x	x	x	x	x	x	x
camp security - regional / federal					x	x	x	x	x	x	x	x	x	x	x	x	x	x

Future development support will be delivered in partnership with non-governmental organisations currently active in-country and provide targeted benefits to women including enhanced irrigation, training and support of existing market gardening initiatives, support for development of goods and services related to workers apparel and PPE, and associated education, training, and mentoring programs.

One of the key achievements of 2023 is the success of training programs aimed at untrained youth and collaboration with Universities and Technical Colleges in Agadez and Niamey, the national capital. This initiative resulted in the hiring of 19 individuals in 2023 and will be expanded in 2024 to include collaboration with the Mining School of Agadez University.

Employment opportunities range from camp services, facilities management to equipment operator positions. The Project currently employs approximately 290 people. SOMIDA prioritizes local and regional hiring whenever practical. The workforce is expected to peak at 900 during construction and level out at 825 during mining operations and therefore represents a significant opportunity for local and regional economic development. The SOMIDA workforce is currently 98% Nigerien and expected to remain so during the full life of the Project.

As the Project ramps up into commercial operations, corporate social responsibility contributions will be reviewed with reference to the success of projects to date, and priorities will be identified in consultation with communities via implementation of the stakeholder engagement plan.

5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1. Accessibility

The Project area is accessible by an all-weather road (N25) approximately 95 km north from Agadez, Niger's second largest city and the final 10 km by unsealed sand piste (track) east from highway N25. The mining town of Arlit is some 105 km north of the Project and Niamey (the capital of Niger) is some 1,000 km to the west. The main sealed road N25 (Figure 5-1) is also known as the Route d'Uranium as it is on this road, that all the yellow cake from the Orano uranium mine near Arlit, is transported by truck to the port of Cotonou in Benin, West Africa.

The road continues north from Arlit as a sand piste to the Algerian border and from there as a bitumen road via Tamanrasset, all the way to Algiers and the Mediterranean coast.

Figure 5-1: Road N25, Just North of Agadez - Source: GAC (2019).

The SOMIDA camp (Figure 5-2), is located some 95 km north of Agadez and 10 km east of the N25 highway, accessible via a sand piste. Its coordinates are 17°47'54" north and 7°43'33" east. The mine and processing plant site is located to the west of the camp.

With a few exceptions of rough, rocky terrain the whole Project area is traversable by all-terrain vehicles or 4-WD vehicles.

Figure 5-2: GAC's Dajy Exploration Camp. Source: Pertel (2017).

There are two airports serving the area: Agadez has an airport, Mano Dayak, with a paved 3,000 m runway and recently upgraded infrastructure. It is connected to the airport in Niamey 720 km to the southwest, via charter flights or commercial scheduled connections and at one time also handled international tourist flights from Europe.

Arlit also has an airport with a 2,000 m unpaved runway; however, nearly all flights operating from this airport are charters for Orano’s mining operations.

5.2. Climate

The region is characterized by an arid intermediate climate of the Sahelian desert type with two distinct main seasons: the dry season between October and May and the wet season from June to September.

The temperatures can vary between 0 °C at night in January and more than 55 °C in May or June during the day.

The mean annual precipitation is less than 200 mm and up to 90% of it occurs during the wet season. The rainy season provides sufficient precipitation to allow local basic agricultural activities. Flash floods are frequent inside alluvial dry riverbeds originating in the Aïr Mountains and can quickly turn into torrential streams, making local roads temporarily impassable. Much of the sparse vegetation grows around the riverbeds.

Year-round activities at the Project are possible; however, off-road accessibility within the Project area may be hampered at times during the rainy season. Mine operations in the region operate year-round with supporting infrastructure. METC Engineering believes the climate of the Project area presents no risk to the development of the Project.

5.3. Physiography

The Project is located between the western foreland of the Aïr Mountains and the N25 highway connecting Agadez to Arlit on the eastern edge of the Tim Mersoï Basin. The terrain is generally flat (Figure 5-3), monotonous sandy peneplain with an average elevation of some 500 m above sea level (ASL) with elevations decreasing to the west. The highest elevation is in the Azouza hills, 553 m ASL, whereas the Aïr Mountains located some 30 km to the east may reach over 1,800 m ASL.

Figure 5-3: Typical Terrain at Dasa Project Area. - Source: Pertel (2017).

5.4. Local Resources and Infrastructure

The Project is located in the Department of Agadez which comprises 52% of the surface area of Niger but has only 322,000 inhabitants with a population density of 0.2/km^2.

The Project area is traversed by a 132 kV powerline connecting the Sonichar power plant – located some 60 km south of the Project near the small city of Tchirozerine – with the two uranium mines near Arlit 105 km to the north. The power plant runs two 16 MW generators and is fed by coal which was discovered during the uranium exploration phase in the early 1970's.

Sonichar also supplies electricity to the city of Agadez and has considerable excess capacity for any industrial development in the area. With the closure of the Cominak Mine in March 2021, Sonichar will be a significant source of power for the Dasa site.

There are no permanent surface water sources available, but several underground aquifers exist at depths between 300 m and 500 m.

A large pool of mostly unskilled labour is available on short notice within the immediate Project area or from Agadez and Arlit. The Orano uranium operations have trained a local labour force over the years and able workers are available. This includes technical personnel from supervisory levels upwards.

The labour code and the organization of labour are very much based upon the French system.

Mining equipment and most supplies need to be imported from outside Niger. Warehousing facilities exist to some extent in Agadez or Arlit.

Niger has a long history of resource extraction, and mining and exploration services are available on a local level from drilling companies to environmental consultants and support services.

6. HISTORY

6.1. Introduction

Uranium exploration commenced in Niger in the early 1950’s following up on indications from spotty surface mineralization. The exploration for uranium has occurred over time in three phases dictated by the economics of the mineral at various times.

The following section is based on information sourced from the following reports:

- Périmètre In Adrar (Cogema, 1977a).
- Rapport des activités de la campagne de prospection d’uranium (Association Onarem PNC, 1983).
- Projet Sekiret, Programme des Travaux de la 3eme Campagne 1983-1984 (Association Onarem PNC, 1984).
- Projet Sekiret, Programme des Travaux de la 4eme Campagne 1984-1985 (Association Onarem PNC, 1985).
- Projet Sekiret, Programme des Travaux de la 5eme Campagne 1985-1986 (Association Onarem PNC, 1986).

6.2. Regional Exploration by the French Nuclear Energy Commission (1957 to 1981)

Systematic regional uranium exploration in the area started in 1959 after the first uranium mineralization was noted during geological reconnaissance missions in the Aïr Mountains in 1956 (J.R. Leconte mission) and in 1957–1958 near Azelik just west of the Dasa Project area during an exploration program for copper in the Teguida n’Adrar-Assaouas region.

The French Nuclear Energy Commission (Commissariat a l’Energie Atomique – CEA) was responsible for all the work. From 1957 to 1967, an intensive geological exploration program was implemented, which resulted in the discovery of the uranium deposits of Azelik (1960), Madaouela (1964), and finally Arlit-Akouta (1966–1967).

Airborne radiometric and magnetic surveys located many surface anomalies which were quickly followed up on the ground. The CEA later merged into Cogema which became AREVA and is now called Orano.

In the late 1960’s, Cogema completed wide-spaced drilling (several kilometres apart) to test the stratigraphy of the area and to investigate how closely the geology resembled that of the Arlit area further north where uranium mineralization had been known since the mid-1960’s.

In addition to the drilling, other exploration techniques such as geological mapping, rock, and water well sampling, ground radiometric surveys and airborne surveys such as magnetic, electromagnetic, and radiometric were employed.

An airborne radiometric survey with 250 m flight-line spacing delineated a large number of anomalies which were confirmed on the ground and consequently drilled. Much of this drilling was rotary, “wild cat” spaced at several kilometres and stratigraphic in nature. The spacing was reduced to 800 m and 400 m in more encouraging areas. Core drilling was used to confirm the geology and lithology as needed.

The first holes were completed in 1960 and continued until 1972 within the “In Adrar” concession including the Dajy area of the current AE3 concession. A total of 652 holes were completed all over the “In Adrar” concession, of which 12 were in close range of Dajy. No holes were drilled within the actual Dasa area.

The drilling confirmed that the area was underlain by stratigraphy that closely resembled that of the Arlit region.

All holes were probed by radiometric and electric methods using Cogema’s own logging systems.

Significant radiometric anomalies were discovered within the AE3 Exploration Permit in strata older than the Upper Jurassic host of Orano’s Imouraren uranium deposit, located only some 40 km northwest of the AE3 Exploration Permit and the Dasa deposit.

6.3. Regional Exploration by PNC and ONAREM (1981 to 1990)

In 1981, Cogema dropped major parts of their landholdings due to the depressed uranium market at that time. A joint venture between Power Reactor and Nuclear Fuel Development Corporation (PNC) based in Japan and ONAREM (Niger National Geological Survey) acquired a large exploration permit called Sekiret which covered an area of some 4,200 km^2, including the current AE3 and AE4 concession areas. PNC conducted stratigraphic drilling on 800 m × 800 m and 400 m ×400 m centres.

In 1982, 4,686 m were drilled on several kilometre-wide spaced grids exploring several ground anomalies. A much larger program was completed in 1983 comprising 36 holes totalling 11,000 m as a combination of rotary and cored drilling. Drillhole spacing was 2.5 km × 2.5 km over western and eastern sections of the Property. All drillholes were probed for natural gamma, resistivity, sonic, and calliper using Japanese-made equipment.

In 1984, encouraging results were noted in 13 drillholes (6,266 m) in the Dajy area, 13 core holes (1,848 m) in the Sekiret area and five drillholes (2,672 m) near the Arlit fault in the west.

In 1985–1986, 27 drillholes (10,702 m) were completed, of which 7,808 m were core and 2,894 m were rotary. Some of the holes were over the northern sector while others were collared over Dajy and Isakanan. Additional drilling was done in 1987 (7,672 m), seven holes totalling 2,139 m in 1988, 11 holes in 1989 totalling 3,505 m and finally 12 drillholes totalling 3,466 m in 1990.

PNC’s work confirmed that uranium was present in the Tarat, Madaouela and Guezouman formations and in a surface anomaly at Dasa in the sandstones of the Tchirezrine 2 Formation.

The drilling was successful in expanding the Dajy prospect within the current AE3 concession and discovering the Isakanan prospect within the current AE4 concession. The joint venture terminated in 1988.

From 1990 to 2007, the AE3 and AE4 concession areas remained unexplored, and no known exploration activity was reported.

6.4. Exploration Activity from 2007 Onwards

In September 2007, the AE3 and AE4 blocks were granted to GAFC, totalling about 1,000 km^2 and located some 40 km southeast of Orano's proposed large Imouraren open pit.

Within the AE3 and AE4 concessions, mineralization was known to exist in the lower Carboniferous Guezouman and Tarat sediments and the lower Cretaceous Tchirezrine 2 sandstone. The AE3 block includes the historic Dajy prospect where uranium mineralization was known to occur within a 10 km-long × 2 km-wide zone. Dajy is situated along a northwest-southeast trending major lineament, the Azouza fault.

In 2011, GAC announced new uranium discoveries at the AE3 concession, in the area currently known as Dasa (Dajy Area Surface Anomaly), named to differentiate from the historic Dajy prospect. The discoveries are located along the Azouza Fault and hosted in the Tchirezrine 2 lower Cretaceous sandstones. The mineralization is contained in a graben environment with down-faulted blocks. Intersections were:

- Dasa 1 – 2,600 ppm U_3O_8 over 8.8 m.
- Dasa 2 – 1,100 ppm U_3O_8 over 8.6 m.
- Dasa 3 – 1,100 ppm U_3O_8 over 76 m.

Additional exploration work located uranium grades from blowouts on surface as high as 300,000 ppm U_3O_8 within the Tchirezrine 2 sandstone.

Later drilling confirmed that high-grade mineralization exists below potential open pit depths with reported grades in hole ASDH 307 of 0.35% eU_3O_8 over 30 m and hole ASDH 248 at 0.21% eU_3O_8 over 25 m.

In June 2012, the Dajy exploration camp was opened which allowed easier access to the whole concession area and the drill sites.

In 2017 to April 2018, GAC drilled an additional 36 holes which targeted the southern Flank Zone of the graben.

From September 2021 through May 2022, GAC drilled a further 43 holes which targeted inferred resource areas to upgrade these to the indicated resource category.

6.5. Historical Mineral Resource Estimates

Although completed in accordance with CIM definition standards and reported in accordance with NI 43-101, the following MREs are considered historical estimates. These historical estimates of Mineral Resources (described below) differed in several ways including the numbers of drillholes available, the level of confidence in the geological model at the deposits and the prevailing pricing environment for the uranium sector.

2009 GEOEX Isakanan and Dajy Historical Mineral Resource Estimate

An MRE by GEOEX in 2009 yielded a total of 27.9 million tonnes (Mt) of resources at a grade of 821 ppm eU_3O_8 or 50.5 million pounds (Mlb) eU_3O_8 for the Isakanan and Dajy deposits within the AE3 and AE4 concessions. No further information on the 2009 MRE is available to the Qualified Person.

A Qualified Person has not done sufficient work to classify the historical estimates as current Mineral Resources, and GAC is not treating the historical estimate as current Mineral Resources.

2013 SRK Consulting (Canada) Dasa Project Historical Mineral Resource Estimate

An MRE for the Dasa Project was previously completed by SRK Consulting (Canada) in September 2013 (Table 6-1).

Table 6-1: Mineral Resource statement, Dasa Uranium Project, Republic of Niger (SRK Consulting (Canada) Inc., 20 September 2013).*

Category	'000 tonnes	eU_3O_8 ppm	eU_3O_8 Mlb
Inferred (open pit) **	4,713	579	6.01
Inferred (underground) ***	19,396	1,797	76.84
Inferred – Total	**24,109**	**1,559**	**82.86**

* *All figures rounded to reflect the relative accuracy of the estimates. Mineral Resources are not Mineral Reserves and have not demonstrated economic viability.*

** *Open pit Mineral Resources reported at a cut-off grade of 250 ppm of eU_3O_8 per tonne assuming: metal price of \$70/lb of U_3O_8, mining cost of \$5/t, G&A cost of \$5/t, processing cost of \$24/t, process recovery of 90%, exchange rate of C\$ 1.00 equal US\$ 1.00, a mining rate of 10,000 tonnes per day and a pit slope angle of 45°.*

*** *Underground Mineral Resources reported at a cut-off grade of 600 ppm of eU_3O_8 per tonne assuming: metal price of \$70 per pound of U_3O_8, mining cost of \$71/t, G&A cost of \$5/t, processing cost of \$24/t, process recovery of 95%, exchange rate of C\$ 1.00 equal US\$ 1.00 and a mining rate of 5,000 tonnes per day.*

A Qualified Person has not done sufficient work to classify the historical estimates as current Mineral Resources, and GAC is not treating the historical estimate as current Mineral Resources.

The 2013 SRK MRE was superseded by CSA Global's 2017 MRE (CSA Global, 2017).

2017 CSA Global Dasa Project Historical Mineral Resource Estimate (CSA Global, 2017)

CSA Global completed a Mineral Resource estimation for the Dasa Project in February 2017 (Table 6-2: Dasa Mineral Resources as at 1 January 2017 (CSA Global). Mineral Resources were reported using cut-off grade of 250 ppm U_3O_8.

Table 6-2: Dasa Mineral Resources as at 1 January 2017 (CSA Global).

Category	Tonnes (Mt)	eU_3O_8 (ppm)	Contained eU_3O_8 (Mlb)
Indicated	18.9	931	39
Inferred	42.0	940	87

Notes:

- A cut-off grade of 250 ppm eU_3O_8 has been applied.
- A bulk density of 2.36 t/m^3 has been applied for all model cells.
- Rows and columns may not add up exactly due to rounding.
- Mineral Resources are classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (10 May 2014).
- No Measured Resources or Mineral Reserves of any category were identified.
- Mineral Resources are not Mineral Reserves and by definition do not demonstrate economic viability.
- This MRE includes Inferred Mineral Resources that are normally considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. It is

reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

A Qualified Person has not done sufficient work to classify the historical estimates as current Mineral Resources, and GAC is not treating the historical estimate as current Mineral Resources.

The 2017 CSA Global MRE was superseded by CSA Global's 2018 MRE (CSA Global, 2018).

2018 CSA Global Dasa Project Historical Mineral Resource Estimate (CSA Global, 2018)

CSA Global updated a Mineral Resource estimation for the Dasa Project in June 2018 (Table 6-3: Dasa Mineral Resources as at 1 June 2018 (CSA Global). Mineral Resources were reported in two parts; those that have potential for extraction by open cut mining methods, and the deeper higher-grade material outside of the open pit that may be amenable to underground mining.

The open pit Mineral Resources are the parts of the deposit above a cut-off grade of 320 ppm eU_3O_8 that fall within a conceptual optimized pit shell. Higher-grade material above a cut-off grade of 1,200 ppm outside of the optimized pit shell was considered for underground mining.

Table 6-3: Dasa Mineral Resources as at 1 June 2018 (CSA Global)

Category	Tonnes (Mt)	eU_3O_8 (ppm)	Contained eU_3O_8 (Mlb)
Indicated open pit	7.08	3,251	50.8
Indicated underground	2.50	2,553	14.1
Indicated – Total	**9.59**	**3,068**	**64.8**
Inferred open pit	0.26	1,135	0.7
Inferred underground	8.18	2,647	47.7
Inferred – Total	**8.44**	**2,600**	**48.4**

Notes:

- Mineral Resources for open pit mining are estimated within the limits of an ultimate pit shell.
- Mineral Resources for underground mining are estimated outside the limits of ultimate pit shell.
- A cut-off grade of 320 ppm eU_3O_8 has been applied for open pit resources.
- A cut-off grade of 1,200 ppm eU_3O_8 has been applied for underground resources.
- A bulk density of 2.36 t/m^3 has been applied for all model cells.
- Rows and columns may not add up exactly due to rounding.
- Mineral Resources are classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (10 May 2014).
- No Measured Resources or Mineral Reserves of any category were identified.
- Mineral Resources are not Mineral Reserves and by definition do not demonstrate economic viability.
- This MRE includes Inferred Mineral Resources that are normally considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

A Qualified Person has not done sufficient work to classify the historical estimates as current Mineral Resources, and GAC is not treating the historical estimate as current Mineral Resources.

The 2018 CSA Global MRE is superseded by CSA Global's 2019 MRE (CSA Global 2019).

2019 CSA Global Dasa Project Historical Mineral Resource Estimate (CSA Global, 2019)

CSA Global updated a Mineral Resource estimation for the Dasa Project in June 2019 (Table 6-4).

Mineral Resources were reported in two parts; those that have potential for extraction by open cut mining methods, and the deeper higher-grade material outside of the open pit that may be amenable to underground mining.

The open pit Mineral Resources are the parts of the deposit above a cut-off grade of 320 ppm eU_3O_8 that fall within a conceptual optimized pit shell. Higher-grade material above a cut-off grade of 1,200 ppm outside of the optimized pit shell was considered for underground mining.

Table 6-4: Dasa Mineral Resources as at 1 June 2019 (CSA Global).

Category	Tonnes (Mt)	eU_3O_8 (ppm)	Contained eU_3O_8 (Mlb)
Indicated open pit	25.59	1,711	96.5
Indicated underground	0.71	3,250	5.1
Indicated – Total	**26.30**	**1,752**	**101.6**
Inferred open pit	18.93	1,357	56.6
Inferred underground	3.38	4,151	31.0
Inferred – Total	**22.31**	**1,781**	**87.6**

Notes:

Mineral Resources for open pit mining are estimated within the limits of an ultimate pit shell.

Mineral Resources for underground mining are estimated outside the limits of ultimate pit shell.

A cut-off grade of 320 ppm eU_3O_8 has been applied for open pit resources.

A cut-off grade of 1,200 ppm eU_3O_8 has been applied for underground resources.

A bulk density of 2.36 t/m^3 has been applied for all model cells.

Rows and columns may not add up exactly due to rounding.

Mineral Resources are classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (10 May 2014).

No Measured Resources or Mineral Reserves of any category were identified.

Mineral Resources are not Mineral Reserves and by definition do not demonstrate economic viability.

This MRE includes Inferred Mineral Resources that are normally considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

A Qualified Person has not done sufficient work to classify the historical estimates as current Mineral Resources, and GAC is not treating the historical estimate as current Mineral Resources.

The 2019 CSA Global MRE is superseded by AMC's 2023 MRE (section 14 of this Report).

6.6. 2018 Preliminary Economic Assessment

CSA Global completed a PEA of the Dasa Project in 2018 (CSA Global, 2018). The 2018 Dasa PEA was based on CSA Global's 2018 Dasa MRE.

The 2018 PEA involved several iterations of mining study design, which included the investigation of open pit and underground mining methods to exploit the resources. Based on that investigation the most attractive returns were generated from a stand-alone, underground, high-grade mining scenario which would potentially operate for a period of 15 years and potentially produce between 4 Mlb and 7 Mlb of U_3O_8 annually.

The 2018 PEA was superseded by METC's Phase 1 Feasibility Study as published in 2021.

Phase 1 Feasibility Study

METC completed the Phase 1 Feasibility Study of the Dasa Project in 2021, subsequently revised and amended on January 9, 2023, with an effective date of November 15, 2021. The study was based on CSA Global's 2019 MRE but included only those indicated resources that achieved the cut-off grade required for underground mining. In developing the Feasibility Study, it was concluded that open pit mining was not economic and instead, the whole mine should be developed as an underground mine. The Phase 1 Feasibility Study included all economically mined indicated resources, which were largely limited to the Graben area of the Dasa deposit. The Feasibility Study concluded on a mine plan of approximately 12-years producing close to 4 million pounds U_3O_8 annually.

The Phase 1 Feasibility Study is no longer considered current and is replaced by the updated Feasibility Study presented in this Report.

6.7. Production from the Property

There is no known production from the Property.

7. GEOLOGICAL SETTING AND MINERALIZATION

7.1. Introduction

The geology and mineralisation in the region have been reported on since the 1950's with several reports published (a listing of these reports can be found at the end of this Section (7.7. References) and is also listed in Section 27 (References)

7.2. Regional Geology

The Dasa Project is located in north-eastern Niger inside the Tim Mersoï sedimentary basin (Figure 7-1). The basin covers an area of some 114,000 km^2 and is part of the much larger Iullemeden Basin (Palaeozoic-Tertiary) that stretches into Mali, Algeria, Benin, and Nigeria.

In the north and east, the Iullemeden Basin (including Tim Mersoï Basin) is bounded by the Hoggar Massif in Algeria and the Aïr Massif in Niger forming part of the Central Saharan Massif (Figure 7-2). The basin gets deeper to the south and the west. During the early Palaeozoic, continental sediments were deposited into an open gulf to the south of the Central Saharan Massif. In the Mesozoic and Tertiary, marine transgressions invaded from time to time diminishing in thickness to the south and passing laterally into continental series. Uplifts commenced in the mid Eocene, filling the basin with fluvial and lacustrine sediments.

All currently known uranium deposits in Niger are located within the Tim Mersoï Basin in several areas (Figure 7-1 and Figure 7-3):

- Near the city of Arlit, two Orano mines, Somaïr – open pit (discovered in 1967) and Cominak – underground mine (discovered in 1974), with historical production of over 145,000 tonnes of uranium, Orano (2022) reported production of 2,020 tonnes of uranium from Somaïr during 2022. The Cominak mine ceased operations in 2021 as a result of depletion of its resources.
- In the Teguida area, SOMINA/CNNC's Azelik – open pit (producing since 2011, but presently closed).
- At Imouraren (Imouraren SA/Orano), construction started in 2009 and production was originally planned to commence in 2015); projected to be the largest open pit uranium mine in the world. This project is currently on standby.

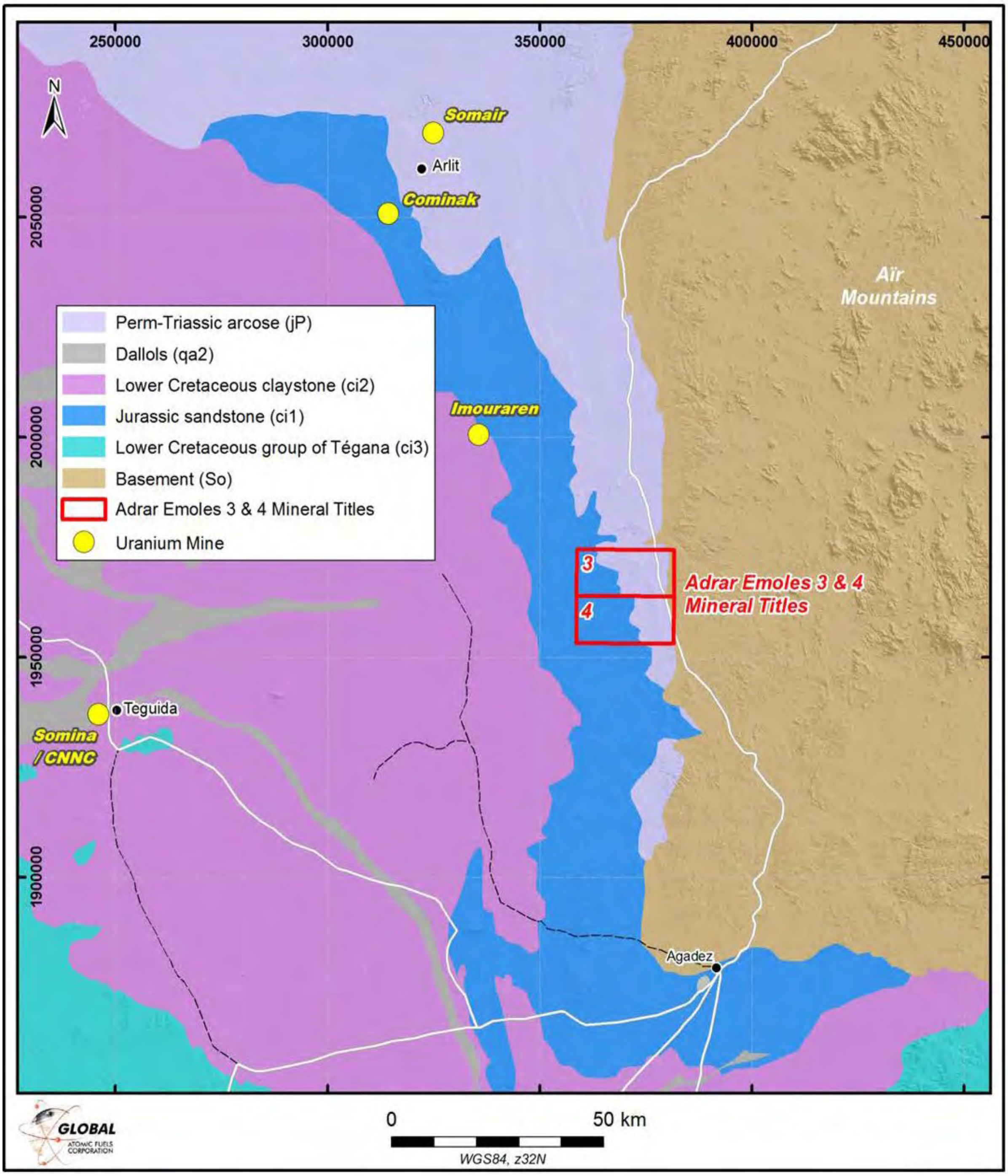

Figure 7-1: Regional Geology Map.

Source: After F. Julia (BRGM, 1963 at 1:500,000)

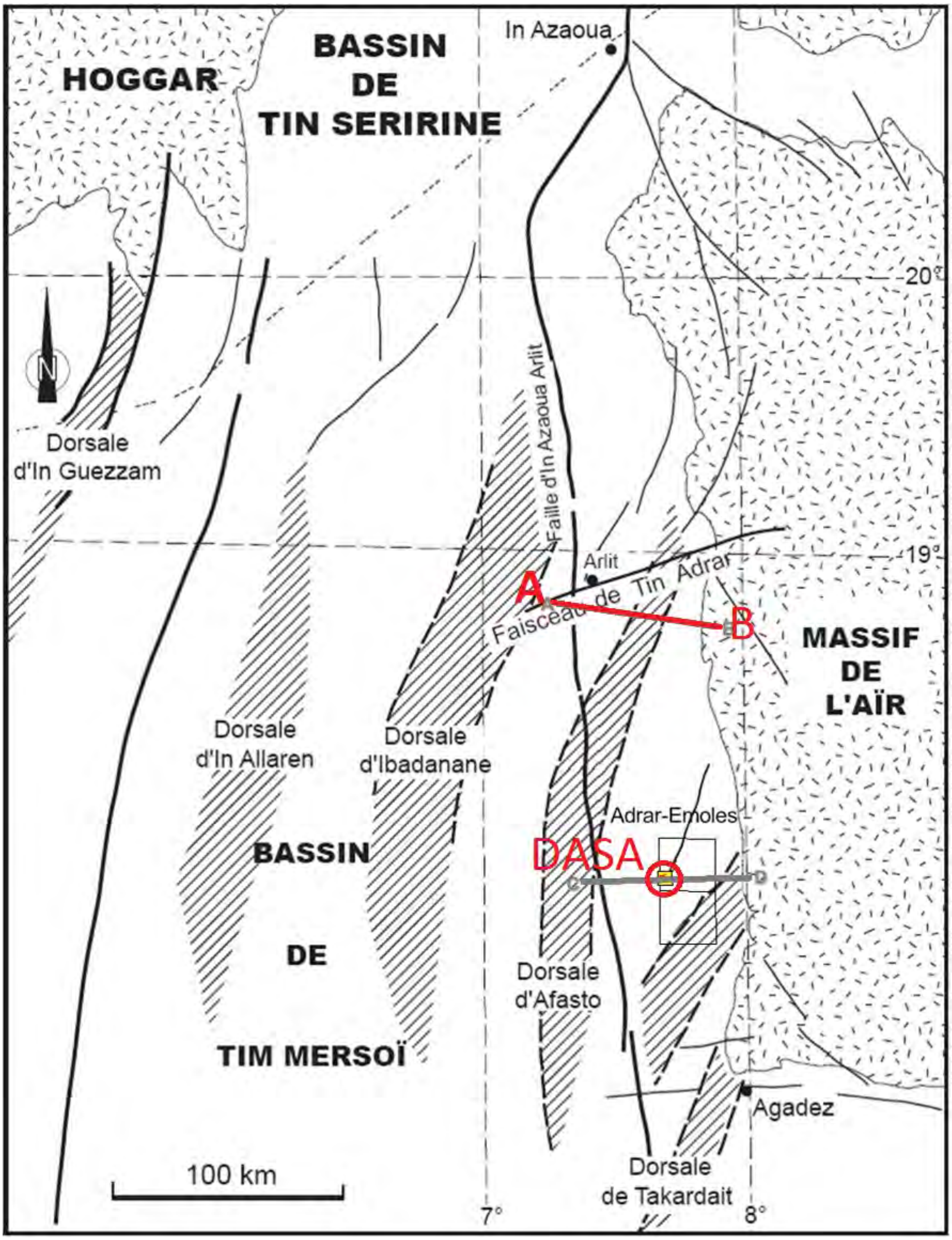

Figure 7-2: Major structures in the Tim Mersoï Basin, Shaping it in a Succession of Ridges and Basins.

Source: modified from Konaté, M. et. al. (2007).

To the east, the basin rests unconformably on the crystalline basement of the Aïr Massif, a Precambrian metamorphic terrain intruded by post Mesozoic felsic and mafic intrusive and in the north and northwest on the basement rocks of the Hoggar in Algeria. The Aïr Massif extends north into Algeria where it becomes the Ouzzalian Craton also of Precambrian age.

The Aïr Massif represents the source for all the clastic sediments that over time have filled the Tim Mersoï Basin and is probably also the source of at least some of the uranium found in the basin's clastic sediments.

The sediments of the basin range in age from Paleozoic to Cenozoic (Figure 7-3) and up to 1,500 m in total thickness deposited on a relatively stable platform.

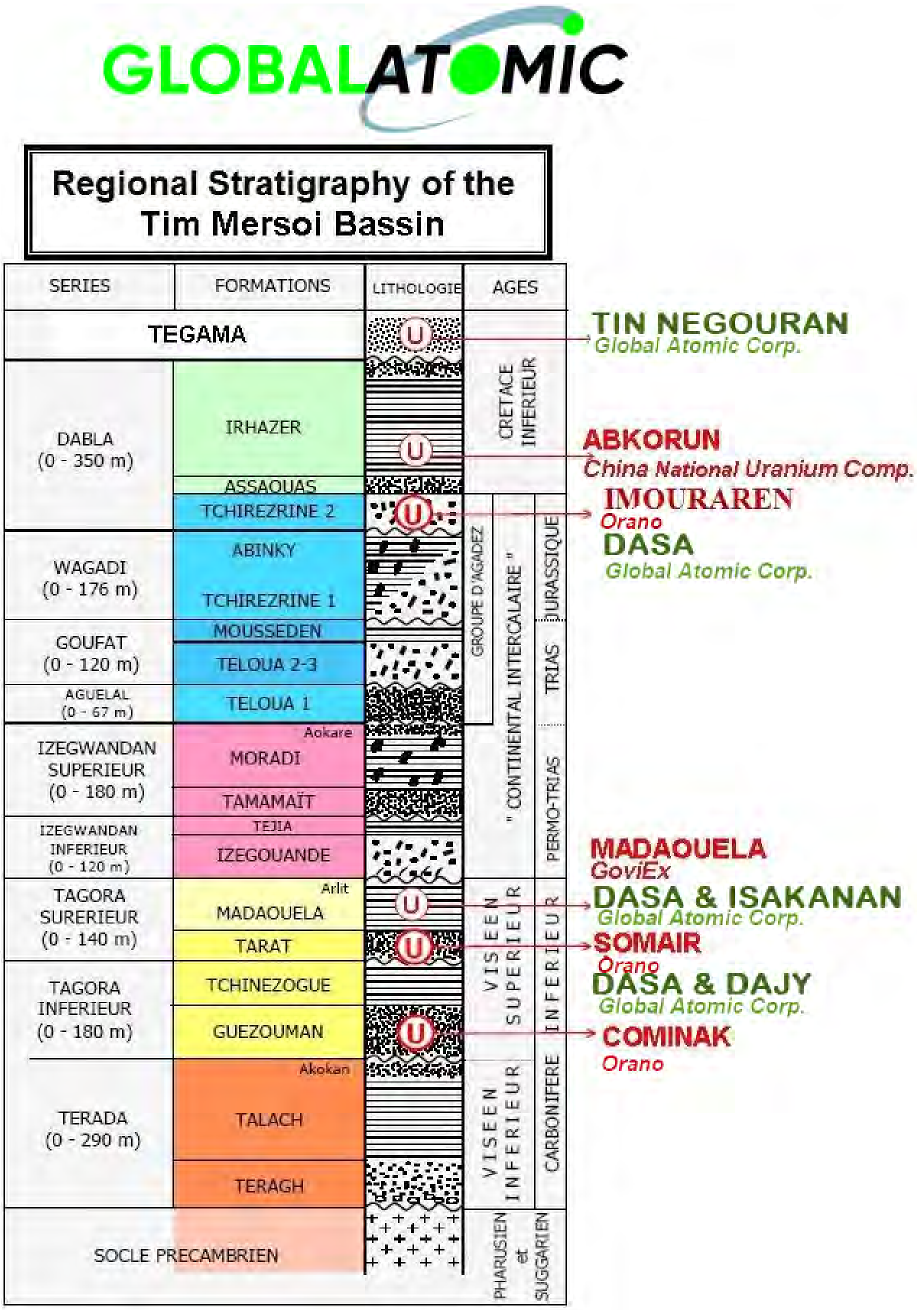

Figure 7-3: Stratigraphic Column of the Series and Formations in the Southern Part of the Tim Mersoï Basin and including the Dasa Project and Property.

Source: GAC.

There are several upward fining sedimentary cycles that have been identified, starting with coarse to conglomeratic sandstone at the base with minor intercalations of siltstone and clay fining upwards into fine-grained sandstone or argillite and clay before the next cycle starts. Each cycle is unique and reflects changes in climate, topography, tectonic events as well as changes in the source areas for the sediments.

The strata of the Tim Mersoi Basin have a shallow dip to the west caused by the uplift of the Aïr Massif (Figure 7-4). The basin deepens gradually to the west and north and shallows over the In Guezzam ridge in Mali. Since the lower Devonian, sedimentation is predominantly continental and marginal littoral comprising conglomerate, sandstone, siltstone, and shale, deposited by large meandering rivers in fluvial and deltaic settings into a slowly subsiding foreland. Further to the west, a more marine environment existed (Joulia et al., 1959).

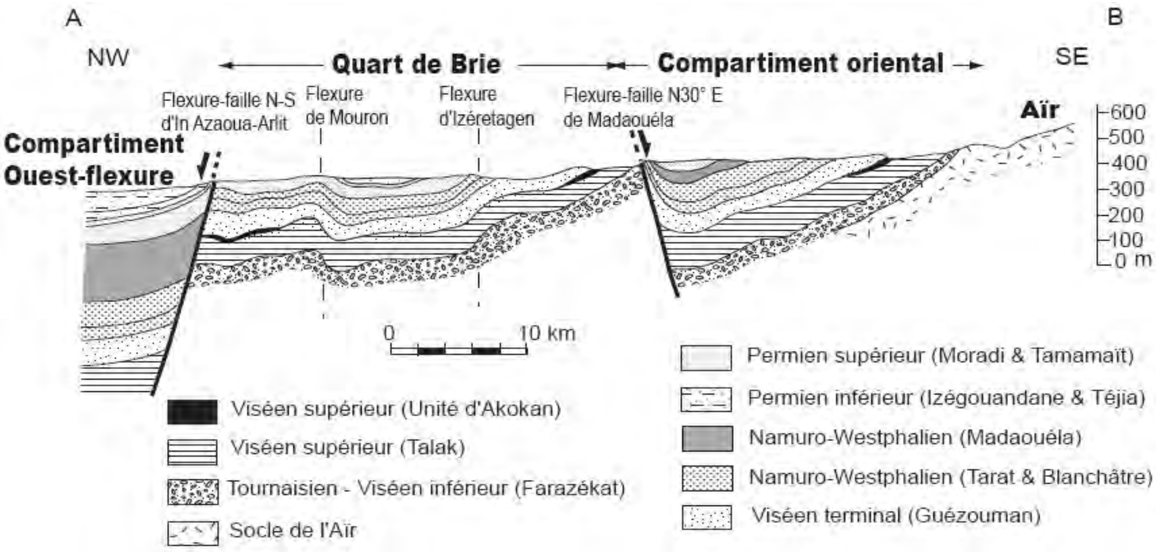

Figure 7-4: Northwest-Southeast Profile in the Arlit Area (see Section A-B in Figure 7-2).

Source: Gerbaud, 2006 (modified from Cominak document, 1989).

The general direction of transport is assumed to have been from the east to the west, and within the Project area a more northeast to southwest direction of transport would have prevailed.

In general, it can be said that the sedimentary strata become younger from north to south, possibly a combination of uplift of the Aïr Massif and erosion and transport directions.

Obelliane et al. (1971) have identified three distinct sedimentary areas within the Tim Mersoï Basin with the main depositional areas moving slowly north to south over time:

- A Lower Carboniferous basin (the Tin Seririne synclinorium) of fluvial-deltaic marine and sediments. These strata are rich in organic matter and silicified trees are common in certain areas of the basin.
- A smaller Permo-Triassic basin with intercalations of volcano sedimentary and fluvial sediments.
- A lower Cretaceous basin with lacustrine deposits overlain by fluvial-deltaic sediments.

7.3. Structural Setting

The Tim Mersoï Basin occurs as a regional scale syncline with a fold axis trending north-south, affected by a combination of brittle faults, mixed flexure-faults, or low amplitude folds or flexures.

The Tin Seririne synclonorium was formed during the Pan African Orogenic event from 550 Ma onwards and forms the northern part of the Tim Mersoï Basin with sedimentation that began during the Cambrian (Joulia et al., 1959).

The structural development of the Tim Mersoï Basin commences at the end of the Pan African Orogen event (1000 Ma). The basin develops by north-south and east-west compression with northwest to west-northwest sinistral shears caused by anti-clockwise rotation in the northeast of the basin. With the widening and deepening of the basin, its centre and the north-eastern edges see the development of sinistral shear zones and conjugate structures trending northwest-southeast and northeast-southwest. The intersections between these structures contain rotational deformation causing dome and basin structures.

Major movements are related to north-south zones which strike parallel to the eastern and the western edges of the Aïr Massif. The compressional sinistral strike slip movements have caused three main structural directions which are north-south; 40–80º; and 90–135º. Where these structures intersect, ideal pathways for circulating uranium-bearing fluids to form deposits are created – S fault system and N30°E associated structures.

The fold-fault of In Azaoua-Arlit comprises a major regional-wide north-south fault system. This family of structures is related to ancient late pan-African transform events. Its frequent re-activation, depending on the epochs, translates into faults, flexures, and flexure-faults in the sedimentary cover.

The N30° family of structures are the most evident on surface in the Tim Mersoï Basin. They appear in the Aïr basement in the east and stop at the In Azaoua-Arlit lineament in the west, where they are truncated. They are linked to the In Azaoua-Arlit history (Sempéré, 1981).

In the sedimentary cover, the deformation is characterized by flexures (Gauthier, 1972; Hirlemann et Robert, 1977, 1980), creating in some instances a substantial vertical displacement in the order of 100–200 m. According to Hirlemann and Robert (1977, 1980), these flexures are linked with sinistral reverse-strike-slip faults activity of the basement structures in a compressive regime.

According to Guiraud et al. (1981), the compressive phase associated with the formation of the N30° flexures is of Upper Cretaceous age, with a shortening direction of N140°.

N130–N140°E and N70–N80°E Conjugate Fault System

A second grouping of faults occurs with orientations of N130–N140°E and N70–N80°. These brittle structures are the most important family in the Aïr Massif. They are of late-Pan African origin according to Greigert and Pougnet (1967).

The N70–N80°E faults are conjugate to the N130–N140°E directions. They are mainly present in the southern half of the Tim Mersoï Basin. During the Carboniferous, these structures controlled the sedimentation in the basin (Wright et al., 1993). These faults played a major structural role in the regional context of the basin, by localizing large scale dextrous strike-slip faults (Gauthier, 1972; Hirlemann and Robert, 1980).

Fold-Like Structures

Fold-like structures are common within the sediments. According to geological drilling data, the thickness and dip variations in some strata from west to east are linked with synsedimentary tectonic activity (Gerbaud, 2006).

Two families of fold-like structures are distinguished:

- Anticlines/Synclines, with fold axes roughly parallel to the N30°E structures.
- Closed structures (domes), which generally appear at the intersection of the N30°E structures and N70-N80°E.

In the south, near the AE3 and AE4 permits, the north-south, east-west, and sinistral shears combine to develop folding, the most obvious being a syncline, in which the Azouza structure is an integral part. The stratigraphy is also folded on approximately layer parallel axis which gives wider exposures and repetition of units. The layers are thickened by layer parallel shortening and on echelon structures develop (Wright, 2010, 2012).

7.4. Property Geology

The rocks present within the Dasa Project area range in age from Precambrian to lower Cretaceous age (Figure 7-5). The schematic geological map is shown in Figure 7-1 and on the schematic cross-section in Figure 7-7.

The rocks are mostly clastic sediments with minor carbonates. They originated from the Aïr Massif which has been continuously eroded since at least the Mesozoic. The sediments were laid down in a continental setting and are generally comprised of fluvial and deltaic settings. In this environment, large shallow rivers meander across flat topography and create complex flow patterns where the coarse-grained sands and gravel are concentrated in the actual channels with the highest flow energies, while low energy flow regimes on the floodplains and tidal areas create silt and mudstone-type sediments. Images of several outcrops are presented in Figure 7-8 to Figure 7-16.

GLOBALATOMIC

STRATIGRAPHIC COLUMN DASA AREA
(Tim Mersoi Basin, Rep. of Niger)

Age (Ma)	Formation	Uranium mineralization Project / Company	Lithology	Depositional Environment		Color code DHLogger
Lower Cretaceous (145 - 100 Ma) IRHAZER GROUPE	Tégama	Tin Negouran / Global	Coarse sandstones, ocre to reddish colour with fine grained lenses Conglomerats with white quartz pebbels	Fluvial	Red Series	
	Irhazer	Abkorun / China National Uranium Co	Alternating carbonatic argillites, marls and clayish carbonates incl dolomitic, greyish layers of silt, mainly reddish colours Reddish argillites with intercallations of silt and sandstones	Lacustrine		
	Assaouas		Silt and argillitic silt greyish-greenish colours			
Triassic -Jurassic (250 - 145Ma) AGADEZ GROUP	Tchirezrine 2	Imouraren / Areva Dasa / Global	Alternating of medium - coarse grained arkosic sandstones with analcimolite, greenish to brownish colours , cross bedding, silicified wood	Fluvial/ lacustrine		
	Abinky		Analcimolite, very hard, red brown; massive banks; Argillites and analcimolitic sandstones partly arkosic, iron cement; med grained sandstones with microcline feldspar; blobs of altered analcimolite; frequent silicified wood in sandy layers	Lacustrine / acide fissure volcanism		
	Tchirezrine 1	Dasa / Global	Medium-coarse grained feldspatic sandstones with abundant analcimolitic cement; also silicieous cement	Fluvial and exhaustive volcanism		
	Mousseden		Reddish argillites with analcimolit; anacimolitic sandstones			
	Téloua 2-3	Dasa / Global	Equigranular sandstones with argillitic-silicieous cement, carbonatic Levels with reddish argillites and silts Carbonnatic cemented arkoses	Fluvial / lacustrine		
	Téloua 1		Fine grained equigranular sandstones sometimes without cement; qtz grains dull sheen and rounded			
Permian (298 - 250Ma) IZEGEGOUANDE SERIES	Aokare Moradi		Equigranular sandstones medium to conglomeratic with iron stains reddish argillites and very fine grained sand lenses carbonatic cemented Arkosic channels strongly oxidized Conglomerates with quart zand clay pebbles	Lacustrine with fluvial intercalations		
	Tamamaït		Medium - fine grained sandstones; carbonate cement; siltes and very fine grained sandstones clayish matrix			
	Téjia		Reddish argillites and very fine grained sand lenses	Fluvial		
	Izegouande		Arkoses and feldsphatic sandstones with carbonate cement; reddish argillite lenses; conglomeratic intercalations with pebbles of quartz, rhylolite, silicified wood, cross bedding			
Carboniferous (Namurien 326 - 313 Ma) TAGORA SERIES	Madaouéla	Madaouela / Goviex Isakanan & Dasa / Global	Silts and very fine grained carbonatic sandstones ; reduced fazies	Estuary / wetlands/eolian	Grey - green Series	
	Tarat	Somair / Areva Dasa / Global	Alternating argillites and very fine grained sandstones rich in organic matter; medium to coarse grained sandstones with organic matter and pyrite	Fluvial-Deltaic		
	Tchinozogue		Alternating black argillites and sandstones, generally abundant organic matter, silt layers	Marine-epicontinental lacustrine		
	Guezoumane	Cominak / Areva Dasa / Global	Alternating very fine grained kaolinitic sandstones and medium-coarse grained sandstones rich in organic matter and pyrite	Fluvial-Estuary		
Lower Carboniferous Visee (358 - 326 Ma) TERADA SERIES	Talak		Argillites dark brown to blueish green; cone in cone structures	Continental marin platform		
	Farazekat (Gabo)		Coarse grained sandstones with argillitic intercalations; well rounded distinct white quartz pebbels (pigeon eggs) at the base	Fluvial to fluvial-glacial		
Devonian ? (419 - 358 Ma) TERADA SERIES	Tindirenen		Alternating medium and fine grained sandstones with blackish/greyish argillite lenses; microconglomerate at the base with silica cement; horizontal strata	Fluviatil / lacustrine		
	Teragh		Sandstone conglomeratic and feldspathic, kaolinitic	Fluvial		
	Basement Precambrian		Granitoids / Pink granite with biotite; some basic dykes			

Figure 7-5: Stratigraphic Column of the Dasa Project Area.

Source: GAC.

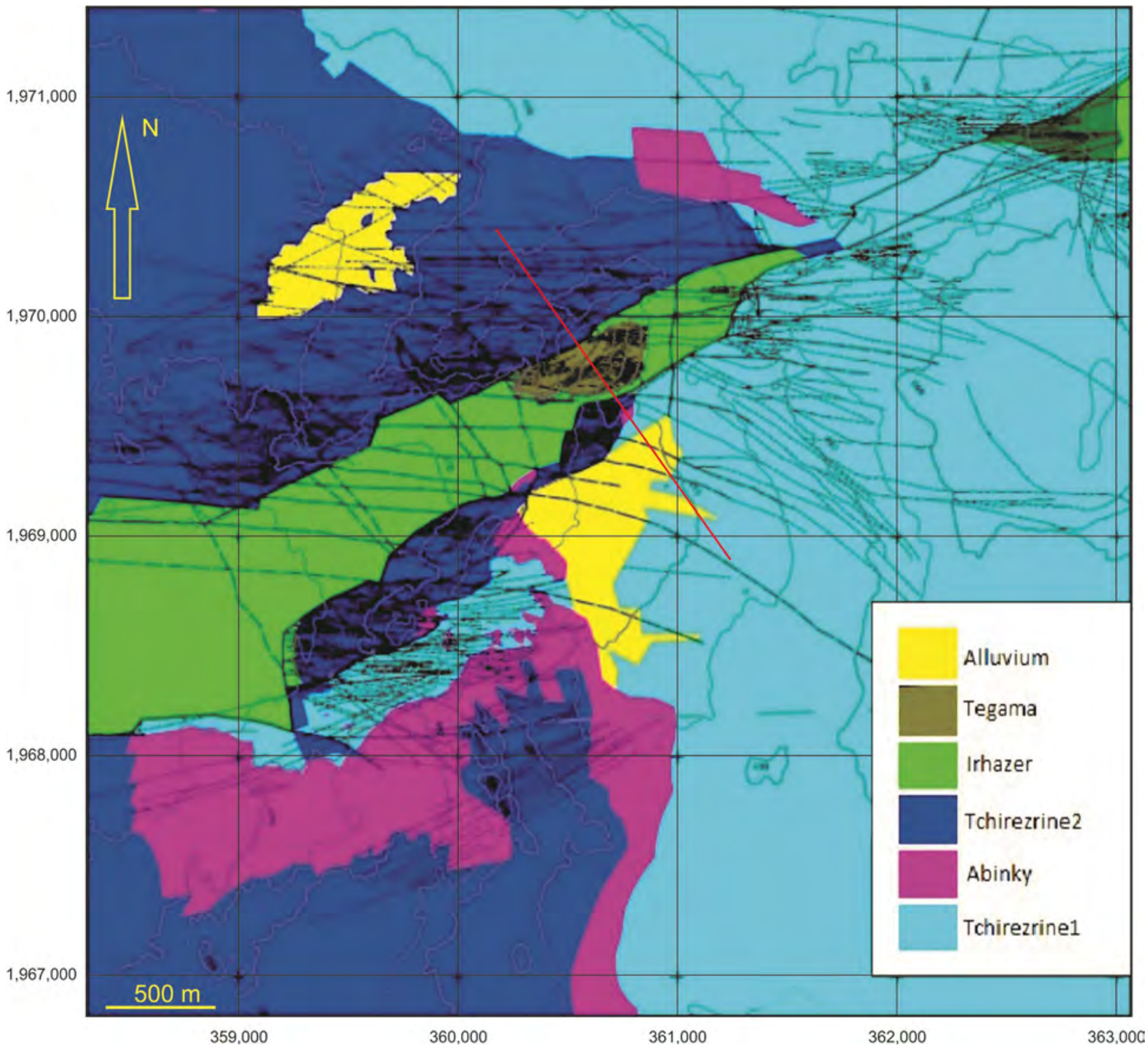

Figure 7-6: Dasa Structural Map (Figure 7 7 Section Line in Red).

Source: GAC.

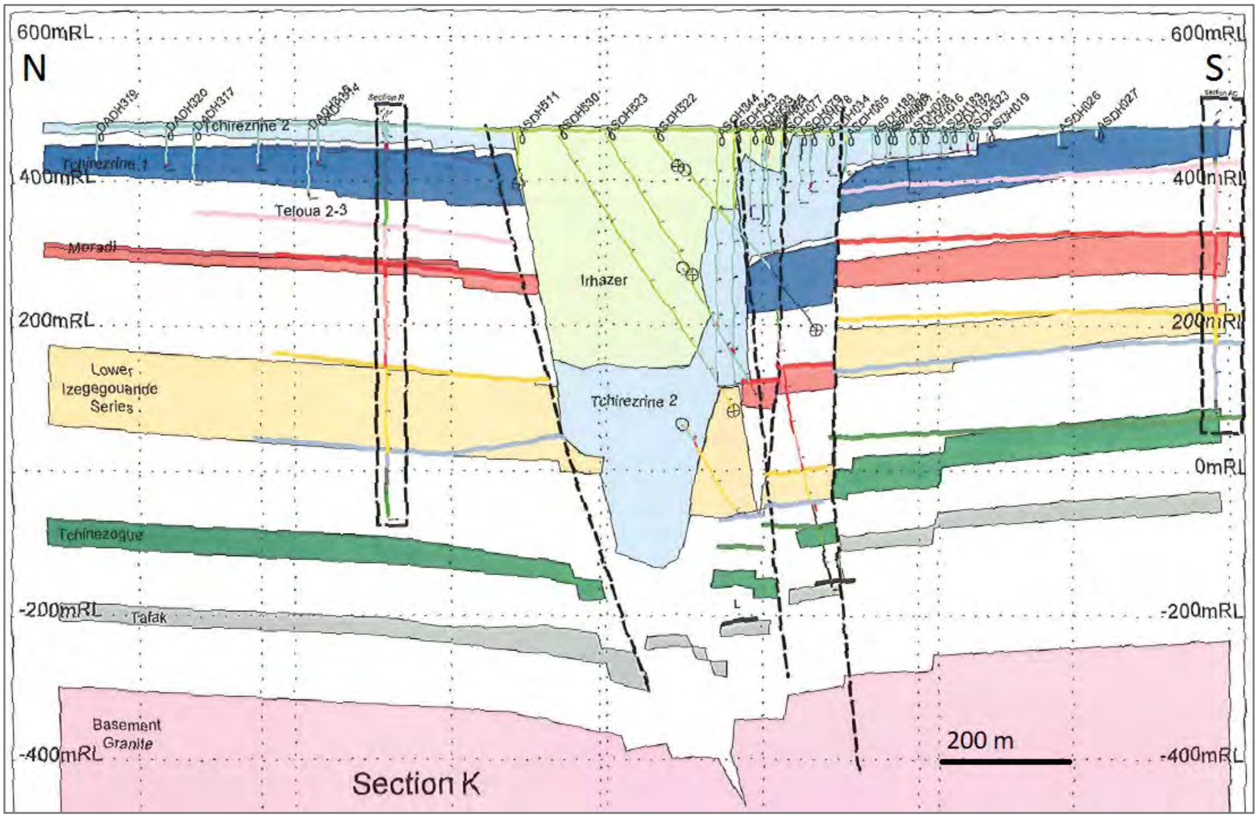

Figure 7-7: Dasa Schematic Geological Section (See Red Section Line on Figure 7 6).

Source: GAC.

The subsections below provide a summary of the lithologies of the Project recorded in drilling, surface mapping and geophysical surveys of the Project areas.

Precambrian

Metamorphic Basement is exposed in the Aïr Mountains to the east. Some of GAC's drillholes inside the Dasa graben terminated in altered and in fresh granite. Its position within the basement suites is unknown currently.

Cambro-Ordovician Undifferentiated

Cambrian to Devonian rocks exist in this part of the Tim Mersoï Basin; however, they have not yet been positively identified by GAC's work. A major discordance near the Aïr Massifs western boundary separates the basement from conglomerates and tillites of the Timesgueur Formation followed by In Azawa sandstone followed by another major discordance.

Upper Ordovician

The upper Ordovician consists of fine-grained sandstones with quartz pebbles and calcite.

Silurian

The Silurian consists of graptolite schists.

Devonian

The Lower Devonian starts with an unconformity followed by conglomerate with pebbles of schists and basalt, Idekel sandstone with silicified wood and is overlain by Middle Devonian Touaret sandstone, fossiliferous beds and the Akara schist. The Devonian is completed by Upper Devonian Amesgueuer sandstone.

Carboniferous

Carboniferous formations are major host rocks for uranium mineralization particularly in the northern part of the basin. The Carboniferous-Lower Visean begins with fossiliferous marine argillites which are overlain by the clastic terrestrial Terada Series which may reach thicknesses of up to 290 m. The Terada itself is made up of the Teragh Formation consisting of coarse-grained sandstones, which can contain coal beds, and is overlain by the siltstones and sandstones of the Aoulingen Formation. This passes laterally to the north into the marine Talach argillites.

The Carboniferous-Upper Visean continues with the important fluvio-deltaic Tagora Series which hosts uranium in the wider Arlit region of the basin. The Tagora is made up of two cycles:

- The first cycle is the lower Tagora up to 180 m thick – starting with the conglomerates of the Teleflak and continuing into the sandstones making up the Guezouman Formation. This is a major host for uranium mineralization in the Akouta area (Cominak underground mine – Orano) and which is overlain by the siltstones of the Lower Tchinezogue Formation which is a mega-sequence comprised of the whitish sandstones of the Middle and Upper Tchinezogue.
- The second cycle of the Tagora (0–140 m thickness) is often marked with a thin layer of conglomerate overlain by the sandstone of the Tarat Formation with intercalations of siltstone and argillite in an upward fining sequence. The uranium at Arlit (Somair open pit mines – Orano) is hosted in this second cycle. The top of the Carboniferous is completed by sandstones and siltstones of the Madaouela Formation (GOVIEX Madaouela project).

The Carboniferous in the whole basin is characterized by reducing conditions displayed in predominantly greyish colours, pyrite and organic matter providing ideal conditions for the precipitation of uranium.

Permian

During the Permian, a major change in climatic conditions occurred and this is reflected in the rocks of that period. The Permian sediments are generally characterized by an abundance of arkosic sandstones containing significant volcanic debris. Reddish colours and abundant calcite are dominant for the Permian strata indicating oxidizing conditions. The sedimentation occurred mostly in interwoven channels with frequent and abrupt facies changes. Within the Project area, the thickness of the Permian strata can vary considerably and reach a thickness of some 300 m.

The Lower Izegouandane Series begins with coarse grained sandstones containing pebbles of rhyolite and quartzite. It is overlain by 5–10 m of a red claystone (equivalent to the Teja Formation) and followed by the sandstones of the Tamamaït Formation. Further up in the sequence, the red siltstone of the Moradi Formation is common. The latter two Formations belong to the Upper Izegouande Series.

Triassic

Initially, the Triassic shows a continuation of the Permian conditions beginning with the conglomerates of Anou Mellé that contain many pebbles shaped by aeolian actions (windkanter). These are covered by fluvio-deltaic sandstones belonging to the Teloua 1 Formation. This package may reach 60 m in thickness and belongs to the Aguelal Series. In some areas, the Teloua 1 displays as reworked sediment with well sorted and rounded quartz pebbles reflecting the local paleo topography.

The following sediments of the Goufat Series contain masses of volcanic debris (origin volcanic tuffs) and are called the Teloua 2 (some 70 m thick). The Teloua 2 appears as distinct poorly sorted sand lenses of the original sedimentation cycle. Analcimolite begins to appear as well. It is followed by the Teloua 3 Formation generally less than 80 m thick consisting of coarse grained to conglomeratic sandstone with frequent rhyolite pebbles. This can be intercalated with analcimolite beds and lenses. These sediments were deposited by very high energy torrential floods. Massive analcimolite intercalated with sandstone layers follows on top as the Mousseden Formation, reflecting a very active eruptive volcanic phase. This formation is generally around 80 m thick but may reach up to 150 m.

Jurassic

The Jurassic consists of the Wagadi Series with a thickness of 80–110 m. It commences with the Tchirezrine 1 Formation (Figure 7-8) representing the channel sedimentation of a large river flowing from north to south. Coarse-grained sandstones are intercalated with finer-grained portions or with siltstones containing much analcimolite. Graben synsedimentary tectonics have caused the variations in thickness as known from the drilling. In general, the Tchirezrine 1 is quite similar to the higher following Tchirezrine 2, except that it does not contain uranium mineralization.

Figure 7-8: Cross-Beds in Coarse Grained to Micro-Conglomeratic Sandstone, Tchirezrine 1 Formation.

Source: GAC.

The top of Tchirezrine 1 is marked by the Abinky Formation (Figure 7-9) comprising argillites and analcimolitic sandstones overlain by a massive sequence of partly silicified or sandy analcimolite. It is testimony to a period of active volcanism. The formation can be strongly altered and mineralized with copper.

Figure 7-9: Massive Analcimolite, Abinky Formation.

Source: GAC.

The Dabla Series, up to 350 m thick, begins with the Tchirezrine 2 Formation which can reach thicknesses of 40–200 m in some parts. It lies unconformably on the Abinky Formation (Figure 7-10), showing local scouring. It was laid down in a fluvial-deltaic and lacustrine environment. The sediments are mostly coarse-grained sandstones and micro conglomerates with cross bedding at the base and with angular detritus pointing to a short and high energy transport path. This is also documented in local erosion of older sediments. The formation contains Orano’s Imouraren uranium deposit approximately 40 km northwest of the AE3 concession and much of the uranium discovered on the GAC property. It is the most important target for uranium exploration in this part of Niger.

Figure 7-10: Tchirezrine 1 Sandstone Covered in the Foreground by Analcimolite of The Abinky Formation

Source: GAC

In general, the Tchirezrine 2 is comprised of a several upward fining sandstone sequences with massive, poorly sorted sandstone beds at the base of each cycle formation with poor sorting, laid down in a high energy flow regime. Each cycle fines upward into fine grained well sorted sandstone with analcimolite on the top and in lenses within the sandstone (Figure 7-11 and Figure 7-12). This sequence is repeated several times. The analcimolites are considered to represent a similar environment and occupy a similar position to the shale in the lower Carboniferous strata. The sandstone generally consists of over 80% quartz, 4–5% feldspar and rock fragments of the Abinky or reworked sandstone.

Figure 7-11: Cross-Bed Features in the Tchirezrine 2-unit, Northern Outcrops at Dasa.

Source: GAC.

Figure 7-12: North-South Structures in Sandstone, Tchirezrine 2-unit, Eastern Outcrops Dasa.

Source: GAC.

The sandstones are generally poorly cemented. The analcimolite appears in two forms: blue, grey, or green within a chloritic matrix or massive brownish in a hematite matrix. The formation was affected by syn-sedimentary tectonics and later shearing which has contributed to the several hundred metre thickness reported in the drilling. The sediments are rich in organic matter which may include coal beds, providing a favourable environment for uranium precipitation.

Cretaceous

The Cretaceous starts with the Assaouas Formation (Figure 7-13), a transition facies to the more argillitic rocks stratigraphically above. The Assaouas reaches a thickness of up to 30 m and consists of reworked older quartz-rich sediments and is overlain by fine-grained sandstones and argillites.

Figure 7-13: Siltstone Outcrop, Assaouas Formation, Southern Outcrops.

Source: GAC.

Overlying the Assaouas Formation is the Irhazer Formation (Figure 7-14), which covers much of the basin and is a testament to a period of little tectonic activity and low erosional regime. It is confined to the Azouza Graben. It represents a lacustrine transgression probably originating in the south or southeast and covers a vast plain affected by subsidence of fine-grained sediments.

Uranium is present within the Irhazer Formation and is being mined at the Abkorun property by China National Uranium Corporation just to the west of the GAC property.

Figure 7-14: Irhazer Formation, Limestone Strata within Argilite, Crosscut by East-West Transform Faults, North-Eastern Outcrops.

Source: GAC.

The stratigraphic column of the Project area culminates in the sandstones of the Tegama Series which lie with a marked unconformity on the Irhazer sediments. Tegama rocks are present in two large hills inside the Azouza Graben and comprise cross-bedded and coarse to micro conglomeratic sandstones. The formation displays heavy quartz veining related to the faults and fractures bisecting it (Figure 7-15 and Figure 7-16).

Figure 7-15: Heavily Quartz Veined Tegama Sandstone; Mount Inside the Assouza Graben.

Source: GAC.

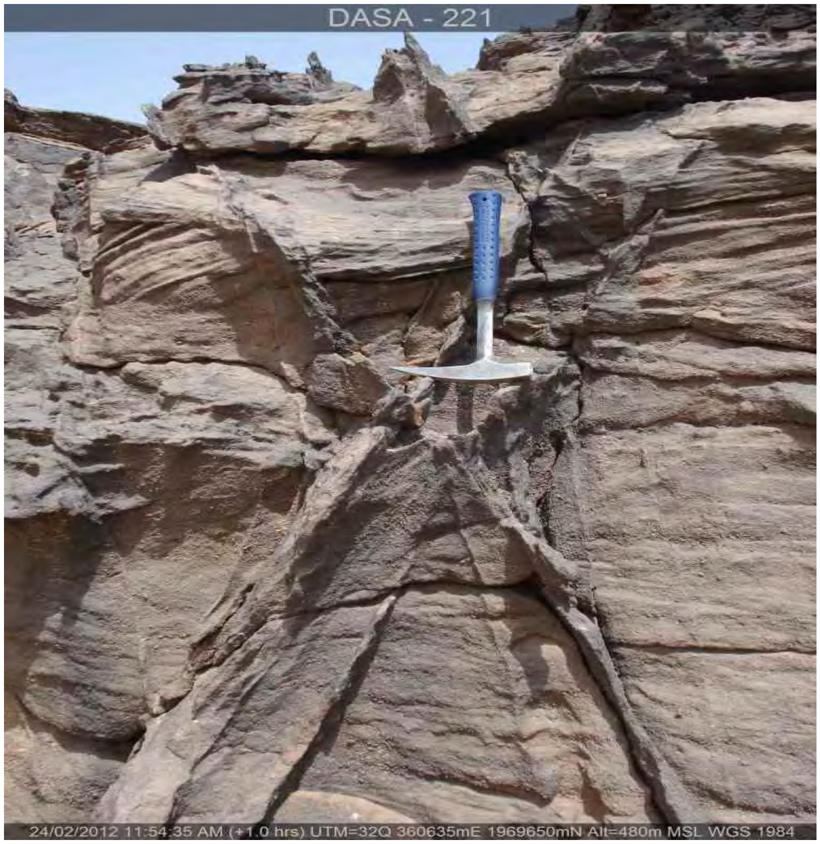

Figure 7-16: Conjugate Fracture Veined in Quartz in Coarse Cross-Bedded Tegama Sandstone.

Source: GAC.

7.5. Structural Geology of the Property

Structural control is important in the formation of most uranium deposits and the Dasa deposit is no exception. The arid climate has preserved structural features, many of which can be observed at surface.

The Dasa deposit site corresponds to a major structural intersection of the Adrar-Emoles flexure and the Azouza fault which resulted in the doming and creation of the Azouza Graben (Siebenthal, 2013). These are features that characterize other major uranium deposits in the Tim Mersoï Basin.

Adrar Emoles Flexure

The Adrar Emoles flexure-fault, one of the major northeast-southwest structures, intersects the Azouza fault at Dasa. This intersection formed a dome, the opening of which created the Azouza Graben (Figure 7-17) dropping the Cretaceous formations to the same topographic elevation as the surrounding Jurassic sandstones.

Figure 7-17: : Looking Southwest, Azouza Graben.

Cretaceous Tegama sandstones in the foreground, resting on several hundred metres of Cretaceous Irhazer Formation to the left with Jurassic Tchirezrine 2 sandstone in the background. Displacement is in the order of several hundred metres.

Source: GAC.

Azouza Fault

Major northeast-southwest vertical faults are associated with the Azouza Graben, characterized by significant vertical displacement of several hundred metres.

The creation of the graben preserved the Tegama and Irhazer formations at depth, elsewhere found much farther to the west in the deeper areas of the Tim Mersoï Basin. It also preserved the rocks of the Tchirezrine 2 Formation which are extensively eroded on the sides of the graben.

This vertical displacement has had a major impact in the continuation of potential host rock geometry and has also provided feeder faults and mineralization traps for mineralizing fluids, as evidenced by veining within the sandstones.

North-Northwest to South-Southeast Faults and Folds

Of key interest are the north-northwest to south-southeast faults observed northwest of the graben. They cut the sandstone formations of the Tchirezrine 2 unit, inducing vertical displacement, with evidence of fluid circulation, enacting localized a lteration and copper mineralization in the analcimolite formation of the Tchirezrine 2 unit.

Shearing Fractures and Veins

Shearing fractures and veins appear in the limestone, particularly of Jurassic age, near the major faults that have a strike-slip component similar to the Azouza and its branches, and the east-west strike-slip faults.

East-West Strike-Slip Faults

Within the upper, northern termination of the Azouza Graben and elevated from the surrounding plain, a limestone outcrop of the Irhazer Formation displays evidence of strike-slip faults. A close examination of satellite imagery reveals a set of roughly east-west oriented structures on both sides of the graben. These are most likely conjugate to the Azouza fault.

7.6. Uranium Mineralization

Regional

Uranium mineralization in Niger is located in sediments of the Tim Mersoï Basin and occurs in most of the thicker sandstone units described earlier; however, not always in economic concentrations and tonnage. Uranium is known in the Carboniferous Terada series, in the Carboniferous Tarat and Guezouman formations (Arlit mines), in the Permian Izegouande, the Jurassic Tchirezrine 2 Formation (Imouraren, Dasa, Azelik deposits) and the Cretaceous Dabla Series as well as in the Tegama Series.

There are three areas in eastern Niger where uranium is presently being mined or could be mined in the near future:

- Arlit-Akokan (Akouta) hosting the Somair open pit and the Cominak (presently closed) underground mines (both mainly owned by Orano) which have produced over 110,000 tonnes of uranium since the early 1980's with considerable reserves remaining.
- Azelik (Teguida open pit/underground mine) operated by CNNC, 160 km southwest of Arlit (presently not producing)
- Orano's large Imouraren deposit some 80 km south of Arlit and 40 km to the northwest of AE3, where an open pit mine is planned to be developed. Orano (2019a) reports that the deposit area holds total (100% share) Reserves of 306,048,000 tonnes Probable ore grading 0.07% U (700 ppm U) for a total of 174,196 tonnes of uranium after 82% recovery on 31 December 2018.

The Qualified Person has been unable to verify the information in the bullet points above and this information is not necessarily indicative of the mineralization on the Property that is the subject of the Technical Report.

The uranium in many of the deposits of the Tim Mersoï Basin is oxidized. Among the primary tetravalent minerals, coffinite is dominant and accompanied by pitchblende and silico titanates of uranium. Uranium hexavalent minerals such as uranophane and meta-tyuyamunite are present in the Imouraren and TGT-Geleli deposits.

The gangue is composed of quartz, feldspar, analcime and often illite, kaolinite and chlorite; with accessories such as some zircon, ilmenite, magnetite, tourmaline, garnet, anatase and leucoxene.

The uranium minerals are frequently associated with copper minerals (native copper, chalcocite, chalcopyrite, malachite, chrysocolla) and also with iron minerals such as pyrite, hematite and goethite. Organic plant materials are generally plentiful in un-oxidized facies of greyish-greenish colour.

Dasa Project

The geometry and the distribution of the uranium mineralization as seen in the Dasa drill core is to a large extent comparable with what has been reported from the uranium mines in the Arlit and Imouraren areas outside the Project:

- There is a strong control by stratigraphy and lithology – with mineralization mainly hosted within the Tchirezrine 2 sandstones, particularly in the coarser-grained micro-conglomeratic facies of greyish-greenish colour containing frequent sulphides and organic matter such as plant remains.
- The mineralized lenses are contained within northeast-southwest trending channels. The thickness of the mineralization may vary considerably between drillholes most likely an indication that channel stacking of favourable lithologies has increased the normal thickness of the sediment pile.
- There are strong indications that the mineralization is influenced by a tectonic control along late northeast and southwest faults where some leaching has been observed.
- Uranium mineralization is controlled by zones of oxidation – from surface (ground oxidation) and local/regional horizons at depth (Figure 7-18).

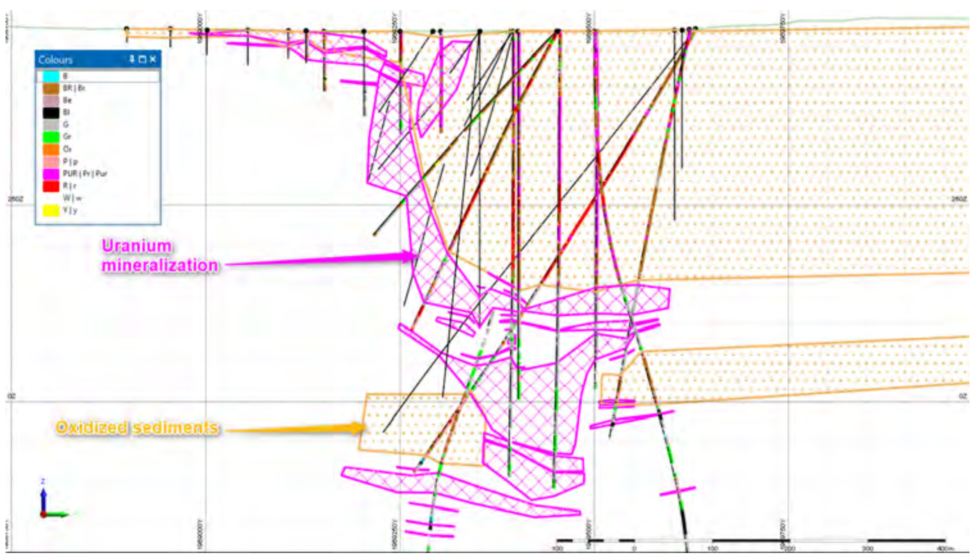

Figure 7-18: Dasa Project Uranium Mineralization Controlled by Zones of Formation of Oxidation (Section 360000me, Looking West).

Source: Pertel (2019).

Groundwater circulation has created overtime discontinuities in the mineralisation as a result of tectonic movements.

Thin section works and petrographic studies by Activation Lab (2007) on Dasa samples has revealed that the uranium host rocks are sandstone and wacke which are variably oxidized. The main component is angular quartz, some plagioclase and lesser orthoclase. They are cemented by goethite, amorphous iron-hydroxides, and various secondary uranium-rich minerals.

The original cement between the grains of quartz and feldspar consisted of sericite and carbonate which were replaced during later stages by goethite and the amorphous iron-hydroxides. The quartz and the feldspar contain micro fractures partly filled with uranium-rich oxide. The latter also rim some of the silicates. Uranophane in form of radiating aggregates forms cement between the silicates and partly replaces them.

GAC initiated a mineralogical study of the uranium mineralization on its property (Molebale, 2012) with five drill samples and five residue samples submitted for analysis. The samples were from drillholes ASDH 351, 353, 354(1), 354(2) and one DADH sample. The samples were split into representative portions and polished sections were prepared. Subsamples were pulverized for x-ray diffraction (XRD).

Five uranium-bearing minerals have been identified in Dasa samples (Molebale, 2012):

- Carnotite $K_2 (UO_2)2(VO_4)_2 \times 3H_2O$.
- Uranophane $Ca(UO_2)2SiO_3(OH)_2 \times 5H_2O$.
- U-rich titanite $(U,Ca,Ce)(Ti,Fe)_2O_6$.
- Coffinite $U(SiO_4)1\text{-}x(OH)4x$.
- Torbernite $Cu(UO_2)2(PO4)2 \times 11H_2O$.
- Autunite $Ca(UO_2)2(PO_4)2 \times 12H_2O$.

Majority of the mineralization is comprised of carnotite, uranophane and uranium-rich titanite and contribute to most of the uranium in the ASDH samples in terms of mass %, while torbernite is dominant in the DADH sample. The average grain size for the observed uranium-bearing minerals is -38 µm.

The source of the uranium is very likely leaching of the frequent volcanic tuff and ash blankets and intercalations now altered to analcimolite within the Wagadi and Dabla sediment packages. This has occurred over time in the geological history of the area and probably began as pre-uranium concentrations during the early sedimentation in favourable reducing environments such as organic matter-rich lower flow regimes and in favourable lithologies. The first stratiform mineralized bodies would have been formed during the early digenesis. Later, structural deformation and ground water movement within coarser grained organic-rich sediments aided by fluid movements and influenced by faults and tectonic activity, initiated roll front like redistribution of the uranium thus giving the mineralized bodies their present shape.

7.7. References

This section has been prepared and is based on the following reports:

Activation Lab (2007).
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8. DEPOSIT TYPES

All known uranium occurrences and deposits in Niger are located in sandstones and conglomerates within the Tim Mersoï Basin. They are all classified to belong to the sedimentary tabular, paleo channel and roll-front or sandstone types.

Sandstone-hosted uranium deposits are marked by epigenetic concentrations of uranium in fluvial/lacustrine or deltaic sandstones deposited in fluviatile continental environments frequently in the transition areas of higher to lower flow regimes such as along paleo ridges or domes. Roll-front type deposits contain impermeable shale or mudstones often capping or underlying or separating the mineralized sandstones and ensure that fluids move along and within the sandstone bodies, thus imitating roll-front systems in Wyoming and Colorado in the western USA.

In the sandstone-type deposits, uranium was typically precipitated from oxidizing fluids by reducing agents such as plant matter, amorphous humate, sulphides, iron minerals and hydrocarbons. The oxidation and reducing facies display typical colours and can assist in exploration target selection. The fluid migrations and deposition of uranium leaves behind a distinct colour change from red hematitic (oxidized) to grey green (reduced). The primary uranium minerals in most sandstone-type deposits are uraninite, pitchblende, coffinite and some secondaries.

Uranium deposits hosted in sandstone make up some 30% of the world's known uranium resources and contain up to 500,000 tonnes of uranium with average grades between 0.1% U and 0.5% U.

In general, it can be noted that in eastern Niger, from north to south the uranium mineralization seems to occur in younger and younger strata. This is most likely a combination of a change in source areas and delivery of uranium over time as well as the fact that to the south the younger strata are exposed on surface necessitating deeper and deeper drilling in southern areas to explore (e.g., for the Carboniferous-aged targets).

In the Dasa deposit, characteristics more consistent with the paleo channel tabular type seem to prevail.

The best uranium grade and tonnage on GAC's property found to date is hosted in sandstones of the Tchirezrine 2 Formation, the same formation that also contains Orano's large Imouraren deposit, located just 40 km to the northwest of AE3 (300,000 tonnes of uranium; Cazoula, 1985). GAC's exploration work demonstrates that many of the characteristics of the Imouraren deposit may exist within GAC's tenure. These include:

Stratigraphy and sedimentology:

Uranium is primarily found in the Tchirezrine 2 Formation, especially in heterogranular sandstones with analcimolite pebbles.

Palaeography:

Mineralization is found in the vicinity of the main channel, the formation of which was partly controlled by post- and syn-sedimentary tectonics while the Tchirezrine 2 was laid down.

Tectonics:

Some remobilization of uranium along faults is known along east-northeast directions, which are post Tchirezrine 2 faults.

Paleohydrology:

Groundwater circulation has affected an earlier concentration stage and has dissolved uranium in some parts of the deposit and re-concentrated it in other parts.

Uranium mineralogy:

Contrary to the Carboniferous mineralization in the Arlit area, the uranium in the Tchirezrine 2 appears mainly as uranium hexavalent minerals in an oxidized environment. Uranophane is the most abundant mineral. It may form small aggregates or appear as continuous coating parallel to the stratification.

Uranophane is commonly associated with chrysocolla and in small quantity also associated with boltwoodite. Metatyuyamunite has also been found. Some coffinite exists in residual reduced zones along with chalcocite and native copper. Pitchblende was noted in small amounts.

The uranium mineralization occurs in two main types: Interstitial within the sandstones, and massive mineralization associated with sulphides in micro fissures with galena and blende.

The above is described graphically in Figure 8-1 below.

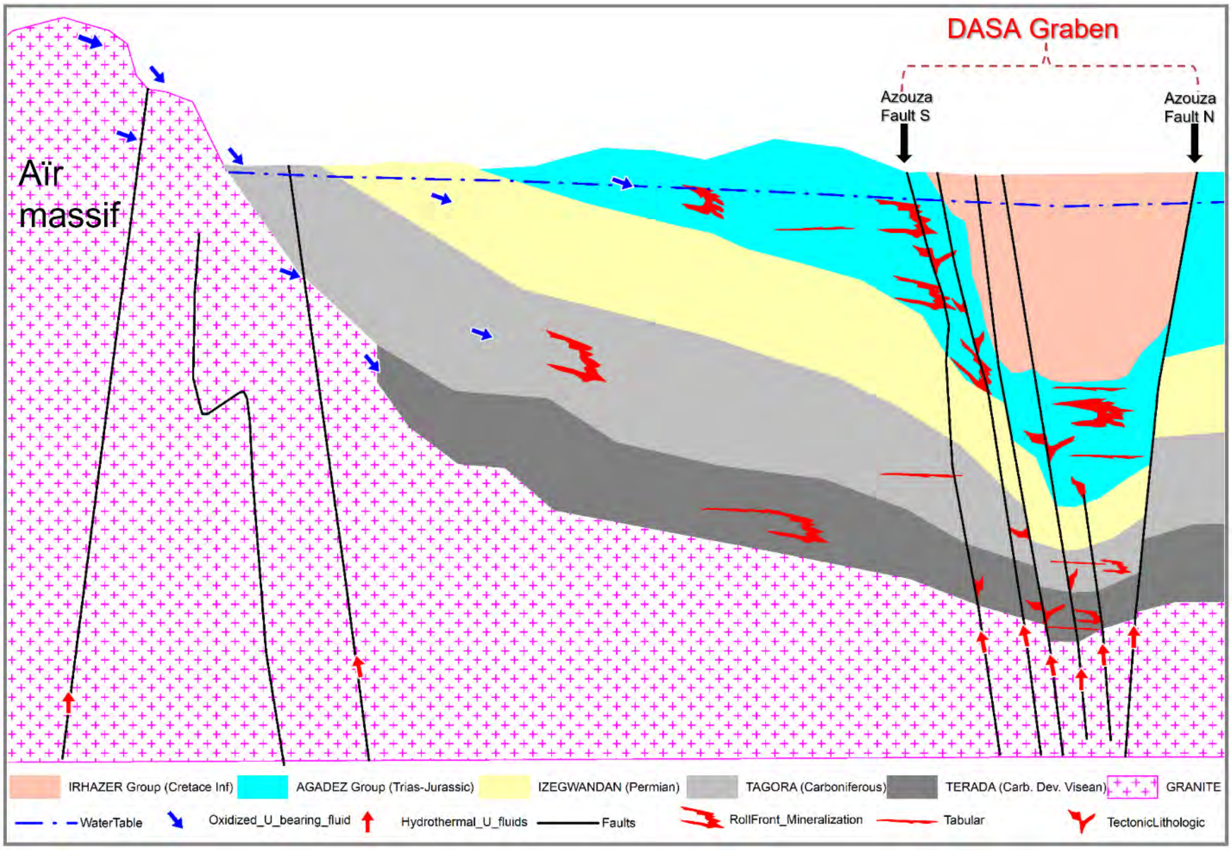


Figure 8-1: Global Atomic Corp DASA Uranium Genetic Model, November 2021 (graphic, not to scale).

Uranium leaching from eastern and mainly northern large basement (Granite/gneisses) and acid intrusive; in fluids into reducing Carboniferous and Triassic-Jurassic sediments in channel settings.

Uranium leaching from volcanic eruptive produced during Jurassic events; intercalated within sandstones or as massive layers of analcimolite.

Faults and structures play important roles in fluids movements and locals for precipitation.

9. EXPLORATION

GAC, through its 100% owned subsidiary GAFC, entered into the Tin Negoran 1, 2, 3 and 4 Mining Conventions in January 2007. Exploration work was initiated by resampling material residual from historical PNC exploration activities. This resampling confirmed high uranium values in the material.

In September 2007, the government of the Republic of Niger granted GAFC the AE3 and AE4 Mining Conventions. Ongoing exploration work and metallurgical studies have confirmed that significant uranium mineralization is located around the Dasa area within the AE3 Exploration Permit. Other uranium occurrences exist within the AE3 and AE4 permits.

GAC has undertaken exploration and evaluation activities on the Dasa Project since 2010. The Dasa Project area covers an area measuring approximately 10 km along the strike of the Azouza Graben by about 2 km. However, drilling has only focused on a small portion of this area.

In 2012, drilling efforts were realigned to achieve two goals: expand the Mineral Resource, particularly the deeper higher-grade uranium mineralization, and to understand the geological controls on the distribution of the uranium mineralization.

In 2017–2018, additional drilling was completed mostly in the central part of the deposit. Infill drilling in 2017–2018 targeted the southern Flank Zone of the graben to improve confidence in the geological model in this area. Additional drilling allowed more confident interpretation of that section of the deposit and an upgrading of its classification.

In 2021-2022, further drilling was completed, mostly in the areas adjacent to the Flank Zone to enable an upgrade from the inferred resource category to the indicated resource category. This program was very successful and substantially increased the quantities and grade of indicated resources applicable to an underground mining scenario.

9.1. Data Compilation and Old Drillhole Locations

In 2008, GAC started data compilation to physically locate historical drillholes, mainly from the previous operations of the Japanese company, PNC. This work was successful at locating many holes at the Azouza Northeast prospect (holes G030, G094, G097, G130) and the Dajy prospect (G120 to G136) located south of the Dasa deposit. Only peak radiometric value records were available (Table 9-1).

Table 9-1: PNC Significant Drillholes.

Hole ID	Location X (UTM WGS84/32N)	Location Y (UTM WGS84/32N)	Peak Radiometric value (c/s)	Depth (m)	Prospect	Location
G030	366591	1973277	6,600	174	Azouza Northeast	Northeast of actual Dasa deposit
G034	361739	1968205	2,150	438	Dajy	South of actual Dasa deposit
G067	361746	1970731	2,000	581		Northeast of actual Dasa deposit
G094	364216	1971980	5,899	528	Azouza Northeast	Northeast of actual Dasa deposit
G096	361165	1969340	4,467	412	Dajy	South of actual Dasa deposit
G097	362183	1971953	2,811	475	Azouza Northeast	Northeast of actual Dasa deposit
G120	361256	1969305	5,417	428	Dajy	South of actual Dasa deposit
G129	362697	1970250	2,360	421	Azouza Northeast	Northeast of actual Dasa deposit
G130	365843	1972250	2,327	276	Azouza Northeast	Northeast of actual Dasa deposit
G132	361735	1969110	1,547	408	Dajy	South of actual Dasa deposit
G133	361436	1969235	3,542	428	Dajy	South of actual Dasa deposit
G134	361720	1970070	4,461	398	Dajy	South of actual Dasa deposit
G135	360889	1969449	5,727	428	Dajy	South of actual Dasa deposit
G136	360825	1968195	1,000	453	Dajy	South of actual Dasa deposit

GAC's first exploration activities were then concentrated on the above areas, and included:

- Radiometric ground survey.
- Geology and structural studies.
- Topographic 3-D survey.
- Drilling.

9.2. Radiometric Ground Survey and Geo-Structural Mapping

GAC conducted a ground scintillometer survey on Dasa, the area (Dasa 1, Dasa 2 and Dasa 3 prospects) covering about 4 km^2 using a SAIC Exploranium GR-135 Plus radioisotope identification device. Natural gamma peak value was recorded for each sampling station.

The Dasa 1 prospect was covered at a sampling grid of 100 m ×100 m; 100 m × 50 m; to 25 m × 25 m locally for a total area of 1.5 km^2 covered, and 105 points surveyed. The objective was to delineate the surface anomaly of this area's Tchirezrine 2 sandstone.

The Dasa 2 prospect was covered at a sampling grid of 100 m × 100 m; 50 m × 50 m; to 25 m × 25 m locally for a total area of 1.39 km^2 covered, and 124 points surveyed.

The Dasa 3 prospect was covered at a regular sampling grid of 100 m × 100 m over a total area of 2.4 km^2; 13 points were surveyed.

A total of 15 rock samples were collected on the highest radiometric count survey point for assays (Table 9-2).

Table 9-2: First Rock Samples of Dasa Area.

Rock sample	Location X (UTM WGS84/32 N)	Location Y (UTM WGS84/32N)	Peak Radiometric value (c/s)	Prospect	Assay sample no.	% U_3O_8	U_3O_8 ppm	U_3O_8 lb/t
Dasa-1-001	360978	1970418	4,218	Dasa 1	D1 – 1	0.447	4,470	9.85
Dasa-1-002	361078	1970393	4,800	Dasa 1	D1 – 2	0.554	5,540	12.21
Dasa-1-003	361178	1970368	4,700	Dasa 1	D1 – 3	0.025	250	0.55
Dasa-1-004	361178	1970343	3,850	Dasa 1	D1 – 4	1.920	19,200	42.32
Dasa-1-005	361203	1970368	65,535	Dasa 1	D1 – 5	24.300	243,000	535.57
Dasa-2-001	360440	1970280	57,200	Dasa 2	D2 – 1	1.430	14,300	31.52
Dasa-2-002	360415	1970280	3,617	Dasa 2	D2 – 2	0.042	420	0.93
Dasa-2-003	360465	1970280	21,542	Dasa 2	D2 – 3	0.056	560	1.23
Dasa-2-004	360490	1970280	3,434	Dasa 2	D2 – 4	0.010	100	0.22
Dasa-2-005	360515	1970255	3,870	Dasa 2	D2 – 5	0.013	130	0.28
Dasa-3-001	360360	1969241	1,500	Dasa 3	D3 – 1	0.028	280	0.62
Dasa-3-002	360160	1969110	1,800	Dasa 3	D3 – 2	0.008	80	0.18
Dasa-3-003	360060	1969080	1,800	Dasa 3	D3 – 3	0.012	120	0.26
Dasa-3-004	359964	1969031	33,000	Dasa 3	D3 – 4	0.836	8,360	18.43
Dasa-3-005	359848	1968998	1,720	Dasa 3	D3 – 5	0.003	30	0.07

The highest radiometric peak survey points were designated to be the first drill sites in the year 2010 (Figure 9-1).

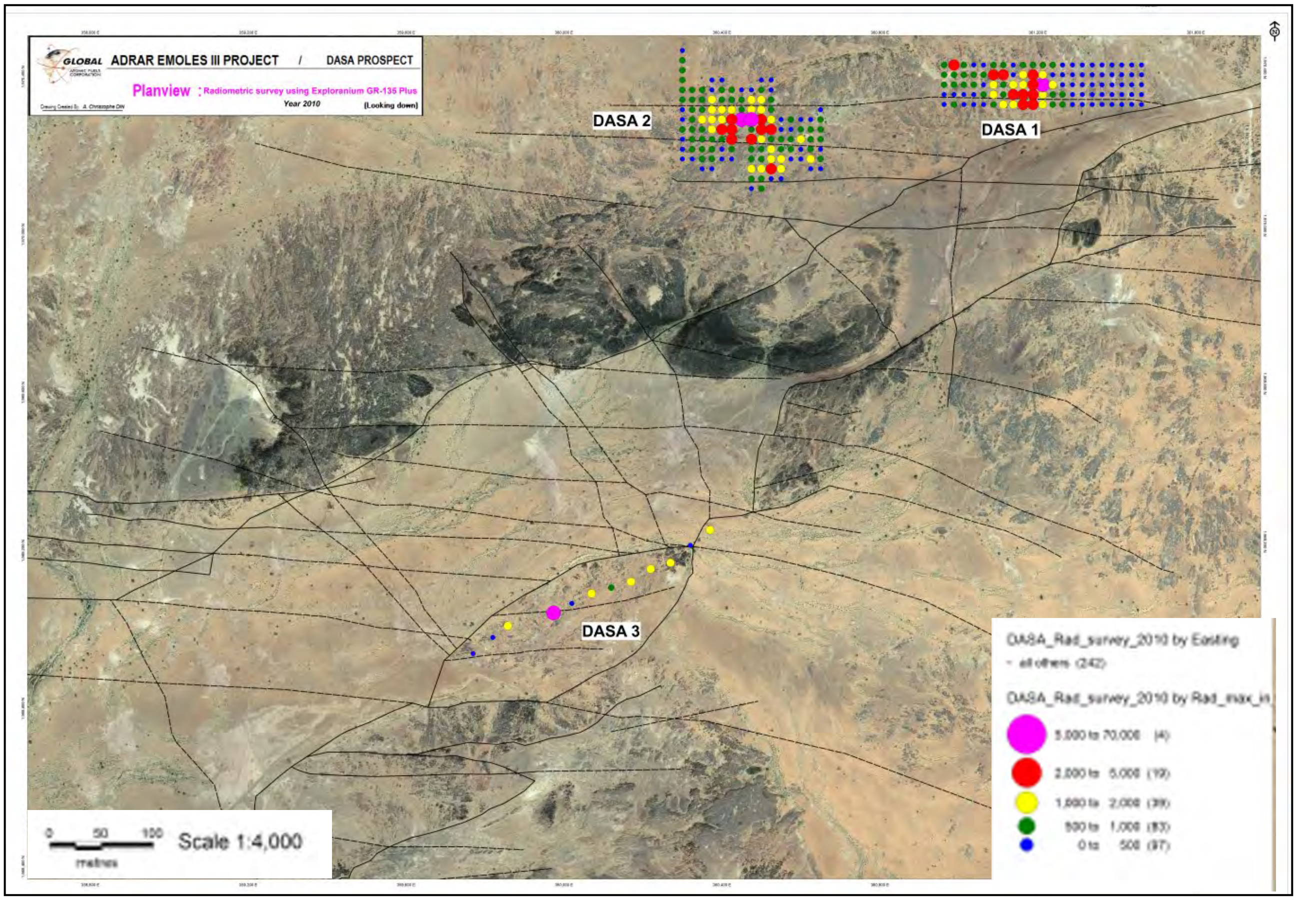


Figure 9-1: Radiometric Sampling Points at the Dasa Project, AE3 Concession.

Source: GAC Internal Report.

Following this survey, Dr Leslie Wright from NewMines Management Services Ltd was hired to complete a study of the mineral potential of the concession during May 2010. Dr Wright conducted an interpretation of the tectonic structures, their age and influence in the control of the uranium mineralization using the initial radiometric survey results and the earlier drilling results as mineralization evidence.

The study concluded that the Dasa area was affected by a main N010 fault system crosscut by the N075 (Azouza fault). The intersection of the first N010 and N075 with the N090-110 structures appear to be key to creating higher grades which are strongly focused at, the location of the prospect but are concentrated also at two areas to the south in this area and pretty much along the line of the main north-south UTM grid coordinate. Dasa 1 and Dasa 2 prospects are affected by a rotated continuation of the 120° trending faults axial planar to the dome structure which hosts the mineralization. Dasa 3 shows a slightly different picture in terms of the definition of targets, with the fold/fault repetition of the mineralized layer appearing likely with the structure being faulted by a 160° trending fault set.

The 010° fault in the east of the Dasa 3 area is only marginally deformed but the rotational interaction between the N045 (Adrar Emoles regional fault) and N010 in the middle of the prospect area creates a compressional environment which may focus mineral deposition.

9.3. Topographic Survey

In order to better define the topographic level of the Dasa area, GAC hired Terrascan airborne for the LiDAR survey and aerial photography totalling approximately 120 km^2. The detailed aerial survey was conducted in December 2013 by CK Aerial Surveys, appointed as a sub-consultant on behalf of Terrascan airborne. The survey was conducted from a fixed-wing platform and consisted of three-dimensional (3-D) laser scanning (LiDAR) and high-resolution aerial photography.

Ground Control

Ground control points were surveyed throughout the site using survey grade global positioning system (GPS) receivers. The surveying was done by means of baseline post-processing. All surveyed baselines had resolved integer ambiguities and therefore none of the surveyed baselines were rejected.

Aerial Survey

Following is a summary of the aerial data capture dates and equipment:

- The survey was done on 31 December 2013 using Diamond DA42 MPP Aircraft equipped with a Leica ALS50-II Laser scanner and a 39-megapixel Leica RCD105, 60 mm lens Camera.

During the execution of the aerial survey, a GPS base station was operated to enable accurate differential processing of the aircraft trajectories. In addition to the position of the aircraft being determined along the flight trajectory, its orientation angles were determined at every point along the trajectory using a state-of-the-art Inertial Measurement Unit (IMU). Using the orientations and GPS-based positions of the aircraft, an accurate point cloud was generated from the continuous laser scanning and aerial photographs were also captured throughout the flight. The laser scanning data was fitted onto the ground control survey. Thereafter, the points were thinned to only include ground points to generate a digital terrain model (DTM).

The pixels from each individual photograph were projected onto the DTM to create rectified photos. Corresponding pixels on overlapping photographs were identified as so-called tie-points. The ground control points were also added as tie-points on the photos and the image orientations were adjusted by means of a statistical least-squares adjustment in order to fit onto ground control and each other. Finally, the individual photos were adjusted to match seamlessly onto each other to form an orthophoto mosaic.

The final DTM is used as the topographic surface on which all the drillhole collars are now pressed to get the homogenized elevation (Z).

10. DRILLING

10.1. Geological Exploratory Drilling

GAC started drilling on the AE3 property in 2010. To date, 1,061 holes (Figure 10-1) including 876 rotary holes and 185 diamond drillholes have been drilled for a total of about 146,100 m on the Project area delineating the Dasa deposit. Drilling of these holes was executed by local drilling companies including TIDIT, ENYSA, ESAFOR, LEGENI (owned and managed by Nigerians), ULC a small French geo-consulting company, and the West African branch of the French drilling company, FORACO. The detailed drilling statistics is summarized in Table 10-1.

The drilling undertaken in 2010–2011 was concentrated on the Dasa surface anomalies with drill depths less than 300 m and mostly drilled by rotary techniques (Table 10-1). These led to the discovery of the surface mineralization of Dasa 1, Dasa 2 and Dasa 3 hosted in Tchirezrine 2 sandstone.

Table 10-1: GAC Dasa Project Drilling Statistics.

Year	Rotary Drillholes		Diamond Drillholes		Total	
	Holes	m	Holes	m	Holes	m
2010	46	1,142	3	437	49	1,195
2011	607	38,420	18	986	625	39,367
2012	195	37,508	41	7,421	236	44.929
2013	17	10,701	29	17,341	46	28,042
2014	0	0	13	8,564	13	8,564
2015	0	0	1	500	1	500
2018	11	4,508	52	21.978	63	26,486
2019	0	0	0	0	0	0
2020	0	0	0	0	0	0
2021	0	0	9	5,006	9	5,006
2022	0	0	19	11,362	19	11,362
Total	**876**	**90,393**	**185**	**73,194**	**1,061**	**163,587**

In 2012, a deeper drilling campaign (up to 754 m depth) below 350 m of Irhazer mudstone targeted the Triassic-Jurassic sandstones (Tchirezrine 2 which hosts Orano's Imouraren deposit, and the Teloua formations) and even deeper, the Carboniferous formations hosting the Orano Cominak and Somair deposits at Arlit. During this program, the main Graben deposit was discovered at Dasa. The drill hole locations are depicted in Figure 10-1.

Drilling during 2013 to 2015 focused on exploration of the central part of the deposit where the Graben Zone was discovered with high uranium grades.

In 2017–2018, GAC drilled 63 holes in a combination of diamond (DD), rotary destructive (RD) and rotary destructive with diamond tails (RD + DD). The drilling targeted mineralization in the Flank Zone on the southern side of the graben structure at depths of less than 350 m and also extensions of mineralization to the northeast, southwest, and at greater depths, for a total of 26,486 m. The average drilling depth was 420 m.

IN 2021-2022, GAC drilled a further 28 diamond drill holes for a total of 16,368 m, targeting areas of inferred resources, so that they could be upgraded to the indicated category.

The 2010 to 2022 drill programs at the Dasa deposit have enabled GAC's geologic interpretation of the deposit and permitted the estimation of Mineral Resources within five major zones (Figure 10-2 and Figure 10-3):

- Dasa 1, Dasa 2 and Dasa 3 zones – hosted in Tchirezrine 2 Formation rocks.
- Graben Zone – hosted in Tchirezrine 2 Formation/Carboniferous rocks.
- Flank Zone – (south side of graben structure) – hosted in Tchirezrine 2 Formation rocks.

The 2017–2018 drill program also identified five distinct areas of new mineralization which were incorporated into the 2019 MRE (Section 14 of this Report) and will continue to be followed up by future drill programs:

- Northeast Extension Zone – (extension of the Graben Zone) – hosted in Tchirezrine 2 Formation rocks.
- Southwest Extension Zone 1 – hosted in Teloua Formation rocks.
- Southwest Extension Zone 2 – hosted in Tchirezrine 2 Formation rocks.
- Tegama Hill Main Zone – hosted in Carboniferous rocks.
- Tegama Hill South Zone – hosted in Carboniferous rocks.

Based on Dasa's previous MRE, that was effective as of June 1, 2019, the Company identified specific areas of Indicated Resources and significant areas of Inferred Resources, particularly between Zones 2 and 3. This information guided the location of the 16,000-metres infill drilling program in 2021-2022. The new MRE was calculated by AMC Consultants, ("AMC"), of Perth, Western Australia, incorporating drill, probe and chemical assay data compiled from the 16,000-meter drill program initiated in September 2021 and finished in September 2022.

Selected mineralized drill intervals from these new zones are presented in Table 10-2 below.

Table 10-2: Select Mineralized Drill Intervals from New Dasa Deposit Zones.

Zone	Hole	From (m) – To (m)	Length (m)	Grade (ppm/% eU_3O_8)
Southwest Extension Zones 1 and 2	ASDH 556G	704.4 – 752.2	47.8	1,806
	including	704.8 – 706.2	1.4	13,511 (1.4%)
	ASDH 558	404.6 – 414.1	9.4	19,933 (2.0%
	including	406.1 – 409.4	3.3	54,101 (5.4%)
	ASDH 574E	490.1 – 576.0	85.9	1,737
	including	497.7 – 576.0	3.6	5,597
	including	511.5 – 517.2	5.7	4,484
	ASDH 578F	717.0 – 768.7	51.7	2,425
	including	718.8 – 721.5	2.7	7,756
	including	731.7 – 734.5	2.8	5,536
	including	765.2 – 768.4	3.2	11,867 (1.4%)
	including	765.7 – 768.3	2.6	15,308 (1.5%)
	ASDH 578G	786.7 – 799.4	11.8	1,478
Tegama Hill South Zone	ASDH 559B	451.1 – 459.3	8.2	4,820
	including	453.2 – 455.1	1.9	20,538 (2.1%)
	ASDH 559I	661.0 – 677.2	16.2	2,098
	including	664.9 – 667.0	2.1	13,286 (1.3%)
Tegama Hill Main Zone	ASDH 577E	521.2 – 591.0	69.8	3,353
	including	552.9 – 557.5	4.6	38,653 (3.8%)
	DADH 388C	494.2 – 561.0	66.8	1,228
	including	511.7 – 513.4	1.6	8,380
	DADH 389D	492.1 – 589.1	97.0	2,348
	including	498.7 – 500.3	1.6	12,752 (1.3)
	including	543.2 – 544.3	1.1	21,285 (2.1%)
	including	546.3 – 550.0	3.7	13,875 (1.4%)
	including	585.2 – 587.6	2.4	8,465

Zone	Hole	From (m) – To (m)	Length (m)	Grade (ppm/% eU_3O_8)
	DADH 390A	420.6 – 437.5	16.9	1,175
	including	431.1 – 433.9	2.8	5,348
	DADH 390B	505.6 – 525.3	19.7	1,017
	including	513.6 – 516.2	2.6	2,576
	DADH 390C	529.6 – 547.7	18.1	848
	including	539.4 – 541.4	2.0	2,393
	including	543.3 – 544.8	1.5	2,018
Northeast Extension Zone	DADH 379C	430.5 – 435.5	5.0	1,454
	DADH 381 C	359.0 – 378.0	19.0	1,010
	including	360.5 – 363.5	3.0	1,863
	including	368.0 – 369.0	1.0	2,455

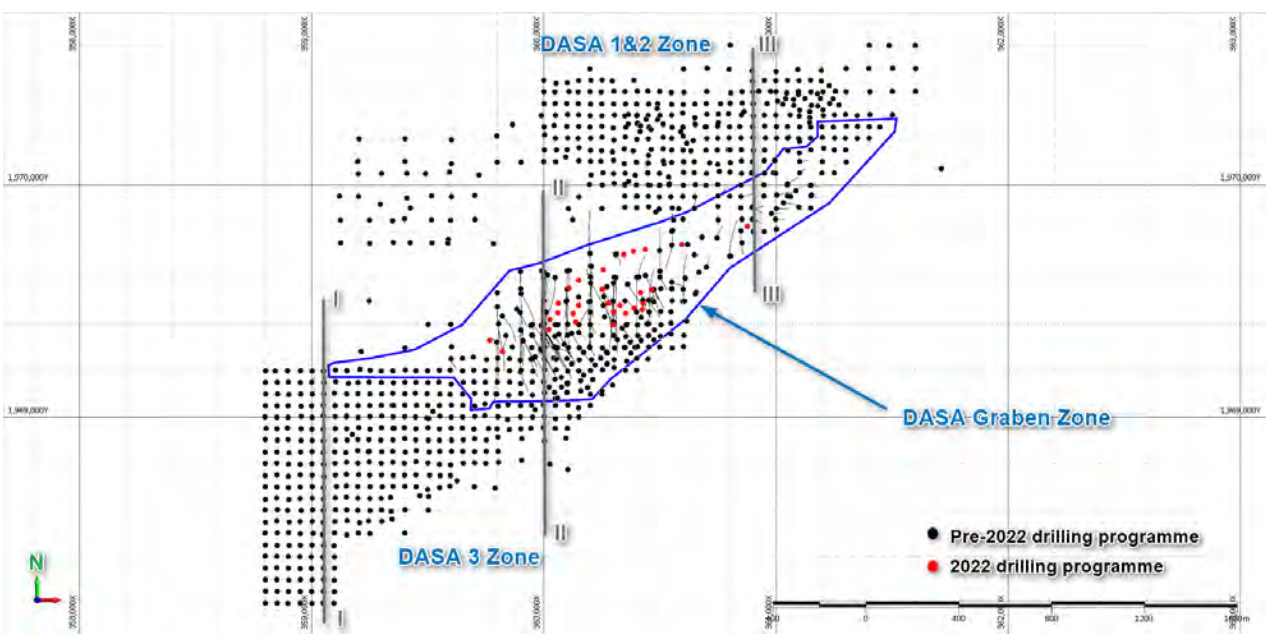


Figure 10-1: Dasa Drill Holes Location Map.

Source: Pertel (2023).

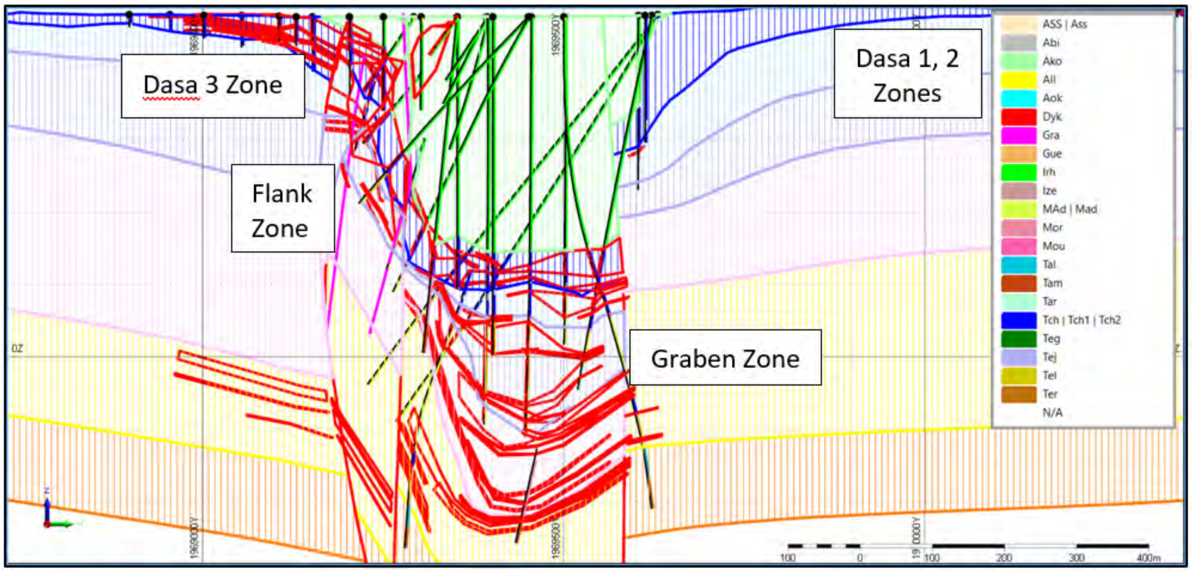


Figure 10-2: Dasa Project Schematic Drill Section – Geology (Section 360000mE, Looking West – see Figure 10 1).

Source: Pertel (2023).

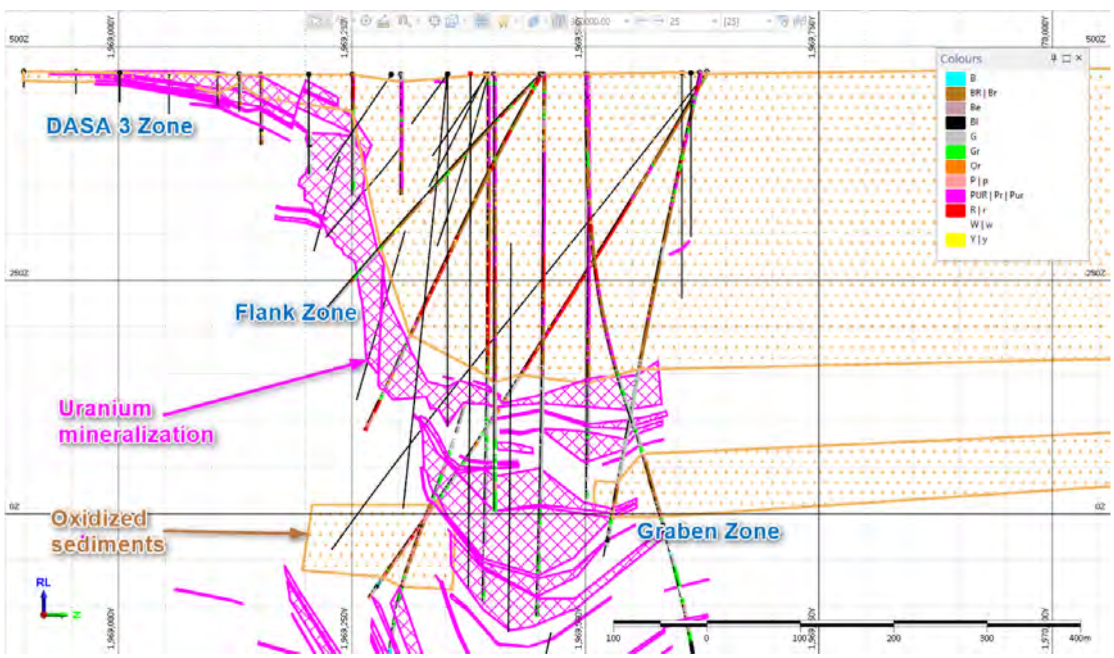


Figure 10-3: Dasa Project Schematic Drill Section – Uranium Mineralization Controlled by Zones of Formation of Oxidation (Section 360000me, Looking West – see Figure 10 1).

Source: Pertel (2023).

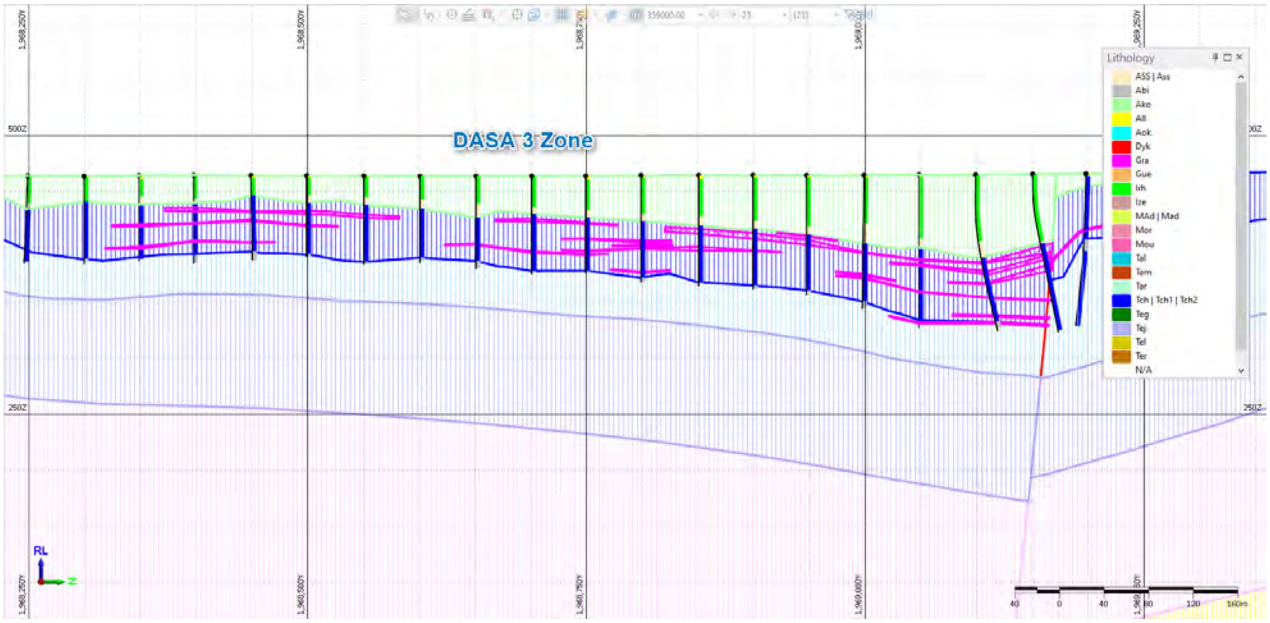


Figure 10-4: Dasa Project Schematic Drill Section – Geology (Section 359000mE, Looking West – see Figure 10 1).

Source: Pertel (2023).

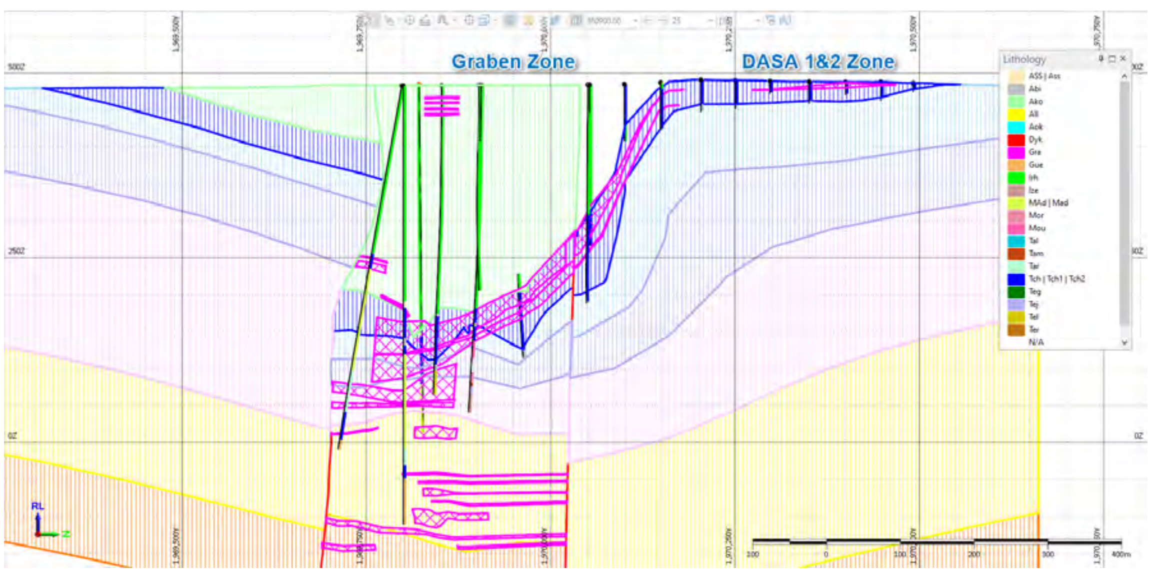


Figure 10-5: Dasa Project Schematic Drill Section – geology (Section 360900mE, Looking West – See Figure 10 1)

Source: Pertel (2019).

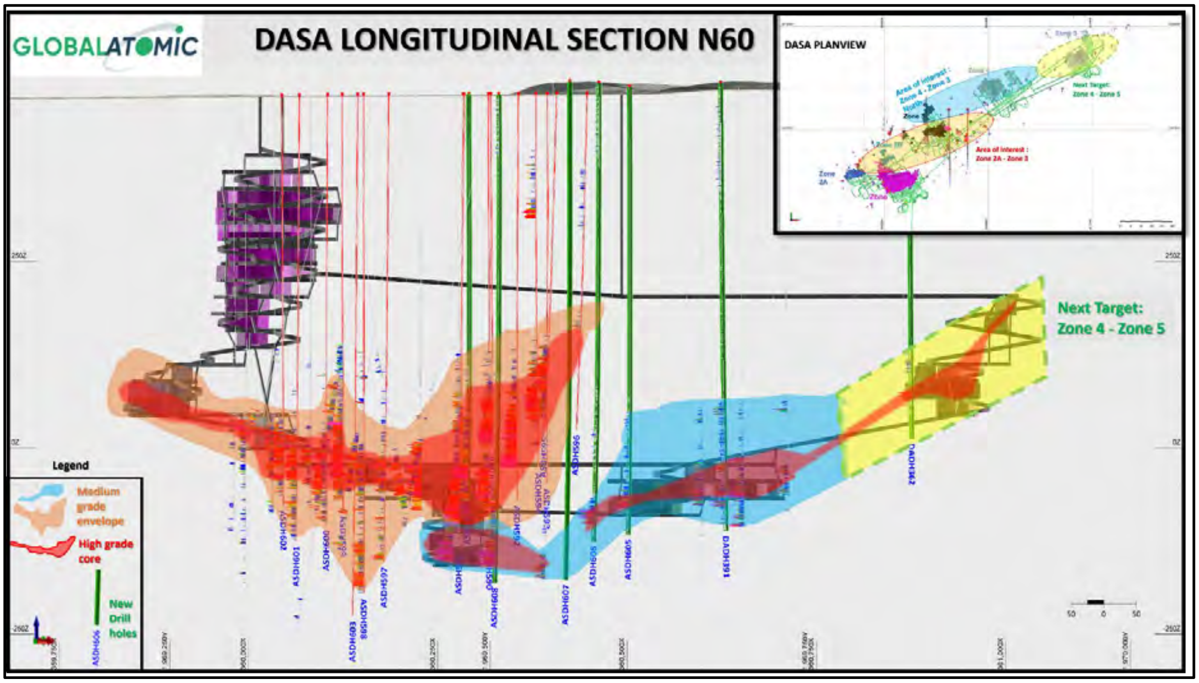


Figure 10-6: 2021 – 2022 Exploration Results: Significant Increase of Resources Expected.

Source: GAC 2022.

Drilling - Procedures

The drilling process through to the sampling is guided by the GAC procedures validated by the Qualified Person. The drill programs were designed by GAC staff in Toronto and implemented by the GAC Exploration Manager based in Niger with the contribution of the Niger exploration team.

The planned holes' locations were pegged by a surveying crew using appropriate surveying tools (Leica DGPS when available or simple GPS). The geologist in charge of drilling checked the hole location before the drilling commenced. A subset of the drillhole collars was verified by the Qualified Person during the site visit and were found in the appropriate locations.

Each new drill setup on a hole was undertaken with the geologist present. The geologist checked the rig settings: azimuth and dip of the mast, before leaving the drill monitoring technician to follow up on the drilling.

Drilling - Monitoring

All drilling was monitored by a GAC technician or geologist. Records were kept of the time of each drill rod and of any technical issues that occurred during the drilling.

Rotary Drilling

On a rotary drill rig, the rock chips come out with the mud. The drilling company workers collected the drill chips from the drill pipe at the hole collar every metre and arranged them in individual piles for the lithological logging. From 2014, a selection of the chips of each 1 m run were washed and put in a chip tray for further description and archived in the core shed at the GAC base camp.

Each 1 m run was tested with a handheld radiometric scanner by GAC workers. The depth and the radiometric counts were recorded. For holes drilled before 2014, these records were not always kept for further depth corrections (lithology versus gamma probe depth). Observations of recovery and suspected contamination were recorded.

Diamond Drilling

For diamond drilling, it is GAC procedure to have a geologist physically present for the drill supervision with at least one technician. For each run, the GAC technician collected the core from the drillers. The core was cleaned and laid down in the core box. In advance of the core being placed in the core box, the box was labelled with the hole ID and the box number by the technician. Cores were arranged as they would be in situ. A wooden or plastic block was placed at the end of each run, recording the depth. The recovered core was measured to determine the recovery percentage. Any detected core loss was recorded and marked with a tag indicating the length of core loss. The core depth was then marked on the core at 1 m intervals.

When an orientation survey was done, the core was marked by the geologist using a solid line with arrows pointing downhole as orientation survey marks. When the core orientation was not reliable, the core was marked using a broken line with arrows pointing downhole. All diamond drillholes from 2012 onward were oriented using an ACT II Reflex tool when ground conditions allowed it.

Each core run was scanned using a Thermo Scientific RADEye PRD-ER to record the radiometric response in counts per second (c/s). Measurements were taken at 10 cm intervals for 5–10 seconds duration. The exposure time can vary up to 10 seconds when the count rate was over 200 c/s.

The core was collected daily and transported to the core storage facility for detailed geological logging. The core was photographed (Figure 10-7) at the dedicated core photography facility.

Figure 10-7: Diamond Core Photograph Example.

Source: GAC.

10.2. Downhole Survey

Common practice during uranium deposit investigations, is the use of the following list of downhole geophysical surveys to help define the geological deposit:

- Gamma-ray logging (GR).
- Electrical methods (resistivity logging (RL) and spontaneous polarization (SP) logging).
- Directional survey (DS).
- Calliper logging (CL).
- Prompt fission neutron (PFN) logging.

During exploration and evaluation, GAC used some of these methods; the results and methods are discussed in the following sections.

Gamma-Ray Logging

GR was done routinely in the open hole conditions. In most holes (rotary or diamond core), the holes were filled with water or mud. In areas of problematic ground conditions, the logging was done inside the drill string or casing. It is very important that this method is done routinely and with precision according to the QA/QC procedures, as it is used to derive the equivalent uranium oxide (eU_3O_8) values used in Mineral Resource estimation.

Several probes were used on the Project for the gamma logging. The parameters of each probe used before 2017 are summarized in Table 10-3: Gamma-ray Probes Parameters.. Five probes were used in the 2017–

2018 program: UEP1805, DGGG1307, DGGG1734, DIL801 and DIL1125. A number of probes were used in the 2021-2022 program with the UEP1805 and UEP2226 being the ones used for gamma logging.

Holes DADH-081 and DADH-011 were used as calibration holes. Each hole was logged once a week to calibrate the gamma tool. In 2018, two deeper and high-grade holes (ASDH264 and ASDH126B) were tested and validated with Orano master probes thus providing additional standard holes to verify the high grades being intersected.

97% of downhole logs were interpreted in Germany by Terratec Geophysics Services; the remaining 3% of holes were interpreted by Semm Logging in France. The logging companies were based at the GAC base camp, and all logging was started within 30–60 minutes of completion of the drillhole. Terratec Geophysics, Semm Logging and their employees are independent from GAC.

Table 10-3: Gamma-ray Probes Parameters.

Probe ID	Probe K factor (U)	Probe diameter (mm)	Mud shielding factor. (mm 1)	Probe dead-time (s)	Casing shielding factor (mm 1)	Probe length (mm)	CRISTAL reference
DIL38 #1125	0.1305	38	0.0047	0.000004	0.043	2,120	1” × 2” Nal crystal
DIL38 #1126	0.1305	38	0.0047	0.000004	0.043	2,120	1” × 2” Nal crystal
DIL38 #1250	0.1362	38	0.0047	0.000004	0.043	2,120	1” × 2” Nal crystal
DIL38 #80	0.126	39	0.0047	0.000004	0.043	2,120	1” × 2” Nal crystal
BDVG #735	0.1119	42	0.0047	0.000004	0.043	140	1” × 2” Nal crystal
DGGG1307, PM	0.8089	42	0.0047	0.000004	0.043	150	2 cm × 5 cm Nal
DGGG1304, PM	0.8089	42	0.0047	0.000004	0.043	150	2 cm × 5 cm Nal
DGGG9354, PM	0.8089	42	0.0047	0.000004	0.043	150	2 cm × 5 cm Nal

Prior to 2014, a logging protocol was not clearly defined. Based on investigation by CSA Global, most work comprised dual induction and gamma log measurements (DIL). The logging speed had been estimated at 3 m to 4 m per minute, which was deduced from the time spent on hole logging. Sampling intervals varied from 0.01 m to 0.05 m or 0.1 m.

Starting in 2014, Terratec geophysical services used the following logging methods:

- Dual induction and gamma log measurements of the rock conductivity; total count gamma was used for the determination of the equivalent radiometric grades of eU_3O_8.
- Combination tool including verticality/focused electric resistivity/Natural Gamma (DGGG).
- The first measurement run was performed inside the fully cased borehole or drill string with the DGGG or DIL probe with an approximate logging speed of 4–6 m per minute and a sampling rate of 0.1 m.
- After the rods were removed, the drillhole was filled with water and relogged using the Combined Verticality/Focused Electric Resistivity/Gamma Probe (as long as the drillhole was still open). The measurement speed of approximately 5 m per minute was used in unmineralized intervals at a sampling rate of 0.1 m. Within the mineralized zones, the logging speed was decreased to approximately 1.5 m per minute. One metre beneath the mineralized zones the logging speed was increased again to 5 m per minute.

Calculated eU_3O_8 was determined by GAC consultants taking into account a steel correction factor when the logging was completed inside the casing or drill rods. The initial probe equivalent U3O8 (eU3O8) is derived using a formula of this type:

$$\mathbf{eU_3O^8\ ppm = 1.1792 * Kf * n / (1 - n * t) * (1 + d * C1 * (Øt - Øs)/2) * (1 + et * C2)}$$

where:

- Kf: probe K factor (ppm U/cps).
- n: probe count (cps).
- Øs: probe diameter (mm).
- et: cumulative casing and rod thickness (mm).
- Øt: drill hole diameter (mm).
- C1: mud shielding factor (mm-1).
- t: probe dead-time (s).
- C2: casing shielding factor (mm-1).
- d: in hole mud density.

A report in *.LAS format was sent to GAC including the radiometric survey and the calculated eU_3O_8.

For quality control and calibration control, the calibration holes were tested at least twice a month and always just before probing a new drillhole. Records were kept by the contractor and delivered to GAC.

Terratec has indicated that all probes used on the Project were correctly calibrated to a defined U-Standard. One calibration U-Standard used, was located in Saskatoon-Saskatchewan/Canada and the second one in Straz Pod Ralskem/Czech Republic. The September 2013 calibration report from Terratec returned good results and the calibration was performed at the Saskatchewan Research Council Uranium Test Pits in Canada.

AMC Consultants also received calibration certificates for the logging tools used for work completed in 2017–2018. These were tested at Orano's calibration facilities at Bessines, France and also at Arlit, Niger. The calibrations for the instruments used were performing within specifications prior to commencing the work on site. For the 2021-2022 program, robe certificates were issued by Orano's Bessine site in France.

The Qualified Person reviewed the calibration results and was satisfied.

AMC Consultants believes that the applied GR procedures were correct and to industry standards.

Radiometric Determination

The basic analysis that supports the uranium grade reported in the Dasa database of uranium grades and thickness of drill intercepts is the downhole gamma log created by the downhole radiometric probe. This data was gathered as digital data and composited to 10 cm data as the radiometric probe was extracted from a drillhole.

The downhole radiometric probe measures total gamma radiation from all-natural sources, including potassium (K) and thorium (Th) in addition to uranium-bearing minerals. In most uranium deposits, K and Th provide a minimal component to the total radioactivity, measured by the instrument as counts per second (c/s). At the Dasa Project, the uranium content is high enough that the component of natural radiation that is contributed by K from feldspars in sandstone and minor Th minerals is expected to be negligible. The conversion of counts per second to equivalent uranium concentrations is therefore considered a reasonable representation of the in-situ uranium grade. Thus, determined equivalent uranium analyses are typically expressed as parts per million (ppm) eU_3O_8 (“e” for equivalent) and should not be confused with U determination by standard X-ray Fluorescence (XRF) or Inductively Coupled Plasma (ICP) analytical procedures. The conversion process can involve one or more data corrections; therefore, the process used for Dasa is described here.

The gamma probes are either 42 mm (Geiger Muller [GM]) or 38 mm (DIL) in diameter and both about 1.5 m in length. The GM probe has a standard sodium iodide (NaI) crystal that is common to both handheld and downhole gamma scintillation counters. GAC constructed GM probes include the scintillation counter and the GM, both of which function similarly to count natural radiometric emanation from uranium and its daughter products (the uranium decay series). GAC initially used only PM DIL probe readings for uranium grade determinations. However, due to the high uranium grades encountered in this program, GAC also used a GM probe which is considered more reliable at higher grades.

The logging system consists of the winch mechanism (which controls the movement of the probe in and out of the hole) and the digital data collection device (which interfaces with a portable computer and collects the radiometric data as CPS at defined intervals in the hole). Radiometric readings are collected digitally into WellCad software for correlation with geology and resistivity. Subsequently, the data is transferred to Utimine software for conversion to eU grade data (G), along with thickness (T), and accumulation (GT; Grade-thickness product).

Raw data can be viewed and plotted graphically from WellCad software, to provide a graphic downhole plot of CPS. The CPS radiometric data may need corrections prior to conversion to eU or eU_3O_8 data. Those corrections include:

- Accounting for water in the hole (water factor) which depresses the gamma response.
- Hole diameter variations.
- The instrumentation lag time in counting (dead-time factor), and
- Corrections for reduced signatures when the readings are taken inside casing (steel-casing factor).

The water factor and casing factor account for the reduction in CPS that the probe reads while in water or inside casing, as the probes are typically calibrated for use in air-filled drillholes without casing. Water factor, and dead-time factor corrections were made to the data at Dasa; all instances of radiometric determination of eU_3O_8 mineralization in core holes were from inside casing at Dasa.

Conversion of CPS to eU or eU_3O_8 was done by calibration of the probe against a source of known uranium (and thorium) concentration. Conversion was also done by determining the relationship of core to radiometric data for a set of core-hole sample intercepts and developing a correlation curve.

The procedure used by GAC at Dasa is to convert CPS per anomalous interval using a correlation curve developed by comparing core intervals with gamma-log intervals for the core hole. The process involved repositioning the core pieces for the whole-core interval of mineralization and determining the contacts and peak radiometric reading with a handheld scintillometer on the core. This is then matched with the radiometric curve developed from a downhole plot of CPS. The core was cut and analysed for uranium content for the same interval as the radiometric interval. A best-fit line defines the relationship of GT as follows:

GTcore = Ucore x Tcore = Factor (CPS x Tprobe) = GTprobe.

The same can be done on composited grade (U%) vs (CPS) at a given composite interval for each; the relationships have been found to be like that for GT. The factor is then used to convert CPS to eU grade as parts per million stated as either ppm U or kg/t (%) U. Database conversions are to U rather than U_3O_8; however, resource tabulations are converted to U_3O_8 as the international standard for which uranium is reported and sold.

Melabar Geoconsulting recalculated the correlation line by inclusion of the core holes from the 2021-2022 drilling campaign with the previous drilling campaigns. Assays were carried out by ALS Vancouver (Canada) and for very high-grade samples (>120,000 ppm) they were re-assayed by SGS Lakefield in Canada. These assay grades (U_3O_8) were compared against the radiometric grades (RA on the plot, eU_3O_8) on a logarithmic cross plot to determine the correlation coefficient (Figure 10-7).

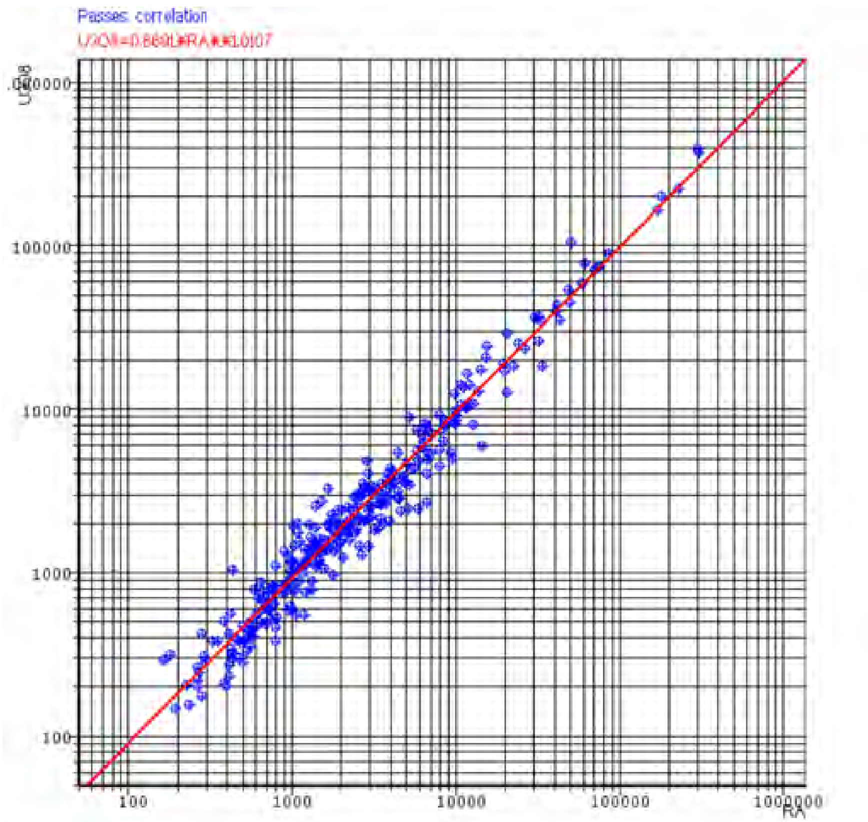


Figure 10-8: Assay vs. Probe eU3O8 Correlation Cross Plot.

Source: Melabar Geoconsulting.

Melabar Geoconsulting has found that the coefficient of correlation between the chemical grades and 10 cm calculated grade composites is 0.7492 with a 2-sigma precision on the mean of 5.8%: a relatively close clustering of data along a linear relationship.

Melabar Geoconsulting provided AMC Consultants with a detailed report describing the uranium equivalent grade calculation based on GR results. The report was reviewed by AMC Consultants, and it was concluded that the applied methodology of eU_3O_8 calculation is acceptable for Mineral Resource estimation purposes.

Downhole Directional Survey

Prior to 2012, GAC drilled shallow vertical holes, and no deviation surveys were completed. From 2012, all the holes drilled, especially in the graben area, were systematically measured for deviation (if the hole remained open).

Both Terratec and Semm Logging recorded the azimuth and the dip of the drillhole at the same time as gamma logging using a combination tool.

GAC also owns a Ranger Explorer Mark II wireless magnetic multi-functional survey system that was used to measure azimuth and inclination for drillholes not surveyed during downhole logging.

GAC also rented a Reflex tool EZTRAC (same system as the Ranger Explorer), operated by its rig monitoring technicians. Some holes were surveyed using this tool.

Each completed drillhole was marked on surface using a heavy cement concrete slab containing:

- The project company name: GAFC.
- The hole name/number.
- The hole type (DD, RD).
- Total length (core length or the reconciled depth after comparing probe and handheld radiometric scanner depth when rotary drilling).
- The azimuth and dip.
- The drill date (year).

The hole collar was then surveyed using the Leica differential GPS or Total Station by the surveying crew or appointed technician/geologist.

10.3. Rotary Chips and Core Logging

GAC uses Constellation Software Inc's commercial data management software called Fusion. GAC uses four main modules of Fusion for data capture and storage:

- FUSION ADMINISTRATOR: To manage user rights and data transfer instructions.
- DHLOGGER: For logging the geology, structure, and geotechnical aspects; for both core and chip logging. It is also used to merge downhole logging and assay import and depth correction.
- FUSION CLIENT: To facilitate data transfer from the field to the office server (intermediate based in Niger and called Fusion Remote, and Central based in Toronto).
- QUERY BUILDER: To export stored data for external use.

The workflow for this system is summarized in Figure 10-9 below.

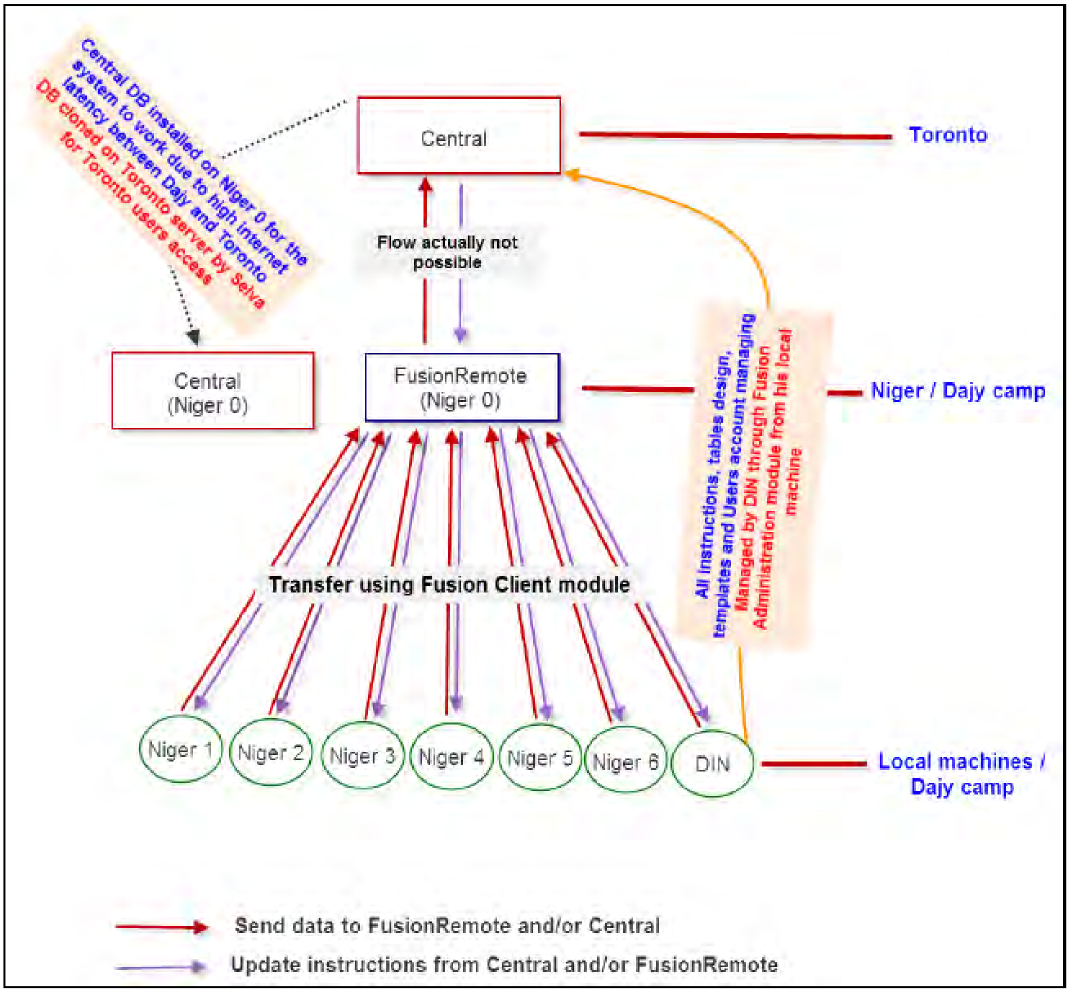


Figure 10-9: GAC Data Collection and Handling System/ Constellation Software FUSION.

Source: GAC.

Rotary Chip Logging

All rotary drillholes have been geologically logged based on 1 m subsamples. Initially, these were based on piles of chips presented by the drillers or the GAC technicians at the logging facility. However, more recently GAC has collected washed reference samples into chip trays for logging and future reference. Initially, logging was completed on paper logs, but since the implementation of Fusion, all data capture has been done digitally.

Core Logging

A more detailed logging procedure was implemented by GAC for core logging to ensure more detailed data was captured. The procedures used are outlined below:

- A geologist remained at the rig at all times during coring.
- All core was processed at site including depth measurement, recovery, and core cleaning.
- Core was then transported to the logging facility daily.
- A core library was established at the base camp to aid in the identification of lithology and rock type to ensure consistent descriptions by the logging crews.
- Special procedures were in place for the handling of radioactive core for logging and sampling. The procedures were made available in hard copy at the logging facility.
- Radioactive core was hand scanned with a personal radiation detector to allow comparison with the downhole probing. The radiometric core was taken from the box and hand scanned every 10 cm on a table inside the core shed. Measurements were recorded in a Microsoft Excel spreadsheet.
- The core boxes were laid down on the logging table at the core shed and geologically logged using DHLOGGER. When geological logging was complete, the core was marked for geotechnical logs when the core was oriented.
- Each hole was then marked up for sampling by the geologist.
- All core was photographed wet and dry in a dedicated facility and transferred to the commercial TEC-CORIM software, allowing image manipulation including transfer to the Fusion database.

Geological logging was completed for the following attributes:

- Geological formations (Table 10-4).
- Colour (Table 10-5), which is important for definition of initial reduced sediments and epigenetic oxidized rocks.
- Sediments/rocks(Table 10-6).
- Alteration and mineralization (Table 10-7).

Table 10-4: Codes of Geological Formations.

Formation	Code	Formation	Code	Formation	Code
Abinky	Abi	Izeguandane	Ize	Tchirezrine 1	Tch1
Akokan	Ako	Madaouela	Mad	Tchirezrine 2	Tch2
Aokaré	Aok	Moradi	Mor	Tegama	Teg
Arlit	Arl	Mousseden	Mou	Tejia	Tej
Assaouas	Ass	Talach	Tal	Teloua 1	Tel1
Farazekat	Far	Tamamaït	Tam	Teloua 2-3	Tel2-3
Guezouman	Gue	Tarat	Tar	Teragh	Ter
Irhazer	Irh	Tchinezogue	Tch	Tindirenen	Tin

Table 10-5: Codes of Colour.

Colour	Code	Colour	Code	Colour	Code
Beige	Be	Gray	G	Purple	Pur
Black	Bl	Green	Gr	Red	R
Blue	B	Orange	Or	White	W
Brown	Br	Pink	P	Yellow	Y

Table 10-6: Codes of Sediments/Rocks.

Code	Lithology	Code	Lithology	Code	Lithology
1	Sand	12	Limestone	22	Granite
2	Alluvium	13	Marl	23	Diorite
3	Clay	14	Muddy sandstone	24	Amphibolite
4	Mudstone	15	Sandy mudstone	25	Gneiss
5	Siltstone	16	Calcareous sandstone	26	Schist
6	Fine sandstone	17	Carbonate mudstone	27	Organic matter sandstone
7	Medium-grain sandstone	18	Arkosic sandstone	28	Pyritic sandstone
8	Coarse-grain sandstone	19	Analcimolite	29	Coal
9	Very coarse-grain sandstone	20	Dolomite	30	Graywacke
10	Micro conglomerate	21	Muddy siltstone	31	Analcimolitic sandstone
11	Conglomerate				

Table 10-7: Codes of Alteration and Mineralization.

Alteration	Mineralization
Carbonate: Ca	Uranium
Iron: Fe	Pitchblende: Pe
Chlorite: Cl	Uraninite: Ur
Sulphides: Su	Coffinite: Co
Manganese: Mn	Carnotite: Ct
Clay: Cy	Yellow products: Pj
	Others
	Pyrite: Py
	Organic material: Om

10.4. Sampling

No rotary chips were sampled for assaying.

For core sampling, a mineralized interval was established from the downhole logging. Prior to 2014 drilling, the eU_3O_8 results were composited at 100 ppm cut-off (allowing 3 m internal dilution of grade lower than 100 ppm). The mineralized interval was sampled from 1 m above and below the interval. Starting in 2014, the cut-off grade was changed to 300-ppm from the downhole gamma logging; the sampling regime followed the system implemented for the 100-ppm cut-off.

After geological and geotechnical logging of the core, the designated mineralized interval was marked for sampling. Sampling was done to reflect the lithological contacts and then routinely at 1 m intervals. For the most recent drilling, the intervals were reduced to 0.5 m intervals.

Sampling was lithological facies related: samples were taken in the same lithological unit (each texture of sandstone should be considered as separate lithological unit, mudstone etc.).

The sample number was written on each core sample using a red marker pen. The marked cores were sent to the splitting facility in the base camp where half core was sampled, bagged, and sealed for mechanical preparation at the ISO 17025 certified Sahel Lab facility in Niamey. The remaining half core prior to 2018 is stored in the core boxes at the base camp. IN 2018 and thereafter, full core was shipped for processing at Sahel Lab, with the residual pulps returned to site. Pulp was shipped from Sahel Lab to an assay facility in Canada.

According to Niger mining legislation, half of any core collected on mining/exploration projects is dedicated to the Ministry of Mines, unless the company has a special authorization to use the entire core. GAC has obtained such authorization for some of their sampling. Subject to the Ministry of Mines authorization, the full core of each marked length was broken and sampled.

A 5–10 cm piece of sample was taken for a specific gravity test prior to bagging and sealing of the to-be assayed sample.

Each sample was packed in a dedicated plastic bag on which the sample number was marked on both sides. A GAC-designed sample tag with the sample number printed on it was also inserted into the bag before the bag was sealed.

The sample numbering was designed to include 10% quality control material:

- Certified reference materials (CRMs) (from ORE Research & Exploration Pty Ltd, Australia) were inserted in the sample bag at a rate of 5:100 samples.
- Certified blank material (from ORE Research & Exploration Pty Ltd, Australia) was inserted at a rate of 2%.
- Blank material sourced from rocks near Niamey was inserted at a rate of 1:100 samples.
- Pulp duplicate samples (taken from the same half core sample) were made for two out of every 100 samples and submitted for analysis.

11. SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1. Sample Preparation and Analyses

Core sampling was undertaken by GAC staff. Samples were collected from quarter, half or full core and appropriately bagged and labelled (before 2013 only half core was retained and hence only quarter core was bagged and labelled; in the 2018 and 2021-2022 campaigns, full core was bagged and labelled). Samples were sent by truck to the Sahel Laboratory in Niamey for preparation. Until April 2013, pulps prepared by the Sahel Laboratory were sent to ALS Geochemistry in Johannesburg, South Africa for analyses. From April 2013 onwards, pulps have been sent to ALS Geochemistry in North Vancouver, Canada for analyses.

The Sahel Laboratory in Niamey is accredited ISO 17025:2005 by Universal Registrars, Bangalore, India for sample preparation. Both ALS Minerals laboratories in Johannesburg and in North Vancouver are also accredited ISO-9001:2000 by QMI Management Systems and to ISO/IEC Guideline 17025:2005 by the Standards Council of Canada for conducting certain testing procedures. The scope of accreditation includes the procedures used for assaying of the samples submitted by GAC. ALS laboratories also participates in several international proficiency tests, such as those managed by CANMET and Geostats. Sahel Laboratory, ALS Minerals and their employees are independent from GAC.

At Sahel Laboratory samples were prepared using a standard rock preparation procedure. Quarter, half or full core was crushed using a jaw crusher until 95% of the material passed a 2 mm mesh. One-eighth of this was taken and pulverized until 90% of the material passed through a 75-micron mesh. One-hundred grams of the resulting pulp was sent to the ALS laboratory for assay. The remaining rejects were returned to GAC and transported back to the field camp for storage.

From the start of drilling until April 2013, prepared pulp samples were sent to ALS Geochemistry in Johannesburg and were assayed for a suite of elements (including uranium) using inductively coupled plasma-atomic emission spectroscopy (ICP-AES) (ME-ICP61) and XRF spectroscopy (ME-XRF05).

From April 2013, prepared pulp samples were sent to ALS Geochemistry in North Vancouver, where samples were assayed for uranium using XRF spectroscopy (ME-XRF05; ME-XRF10).

The switch between ALS laboratories was made primarily to gain access to the XRF10 method of assaying, which can measure more accurately the concentration of uranium exceeding 10,000 ppm. The XRF05 method used in South Africa is accurate to concentrations of uranium up to 10,000 ppm.

The SGS Lakefield laboratory in Lakefield, Canada was used as an umpire laboratory. The SGS laboratory in Lakefield, and Mintek laboratories in Randburg, South Africa were also used to conduct metallurgical testing on surface and core samples representative of the uranium mineralization found on the Dasa Project. The SGS Lakefield and Mintek laboratories are accredited ISO-9001 and to ISO Guideline 17025 for the testing procedures undertaken on material from the Dasa Project. SGS Lakefield, Mintek and their employees are independent from GAC.

11.2. 2018 Sampling Program

During the 2018 drilling program, the full core was sampled over every half metre. A total of 4,983 samples including 10% of control material (6% CRM, 2% field duplicates and 2% field blanks) were collected from 38 holes. The control percentages were doubled due to the importance of the expected high grades from the program holes.

The collected samples were all prepared at Sahel Lab in Niamey. Each whole sample (except the CRM) is crushed to 90% less than 2 mm, riffle split off 1 kg, pulverized split (250 g) to 90% passing 75-microns sieve.

A pulp of 30 g to 100 g was sent to the laboratory for assay and the remaining pulp is kept for further control.

The program samples were all sent to ALS Vancouver for assaying and results were reported through 28 job certificates.

11.3. 2021 – 2022 Sampling Program

During the 2021-2022 drilling program, the full core was sampled over every half metre. A total of 5,307 samples including 10% of control material were submitted: 2% of Duplicate samples (20 samples), 2% of preparation blank (19 samples), 2% of blank (certified grade, 20 samples) and 4% of certified reference material (40 samples standard).

The collected samples were all prepared at Sahel Lab in Niamey. Each whole sample (except the CRM) is crushed to 90% less than 2 mm, riffle split off 1 kg, pulverized split (250 g) to 90% passing 75-microns sieve.

A pulp of 30 g to 100 g was sent to the laboratory for assay and the remaining pulp is kept for further control.

The program samples were all sent to ALS Vancouver for assaying. Extremely high grades (above 15% U) were sent to SGS Lakefield Canada for assaying.

11.4. Core Depth Adjustment

Once the handheld radiometric survey was achieved, the data were imported in WellCad software, and associated to those of the in-hole probe data. The significant peak values of the scintillometer and the probe data are identified and superimposed: the probe depth being the reference depths, only the scintillometer data are moved up or down to fit (Figure 11-1 and Figure 11-2) .

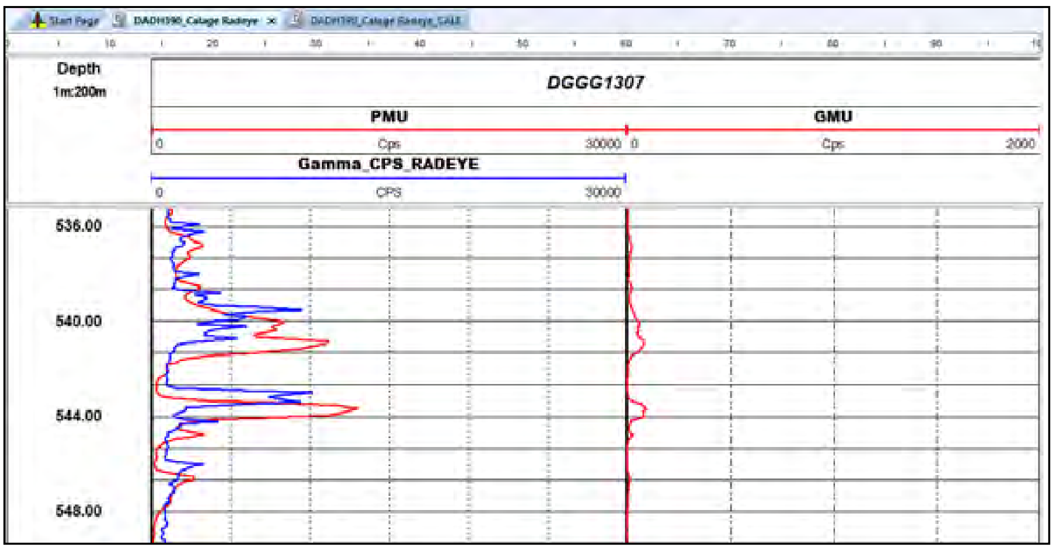


Figure 11-1: Hole DADH390, RadEye (blue) vs. Probe (red) Depth Before Rescaling.

Source: GAC.

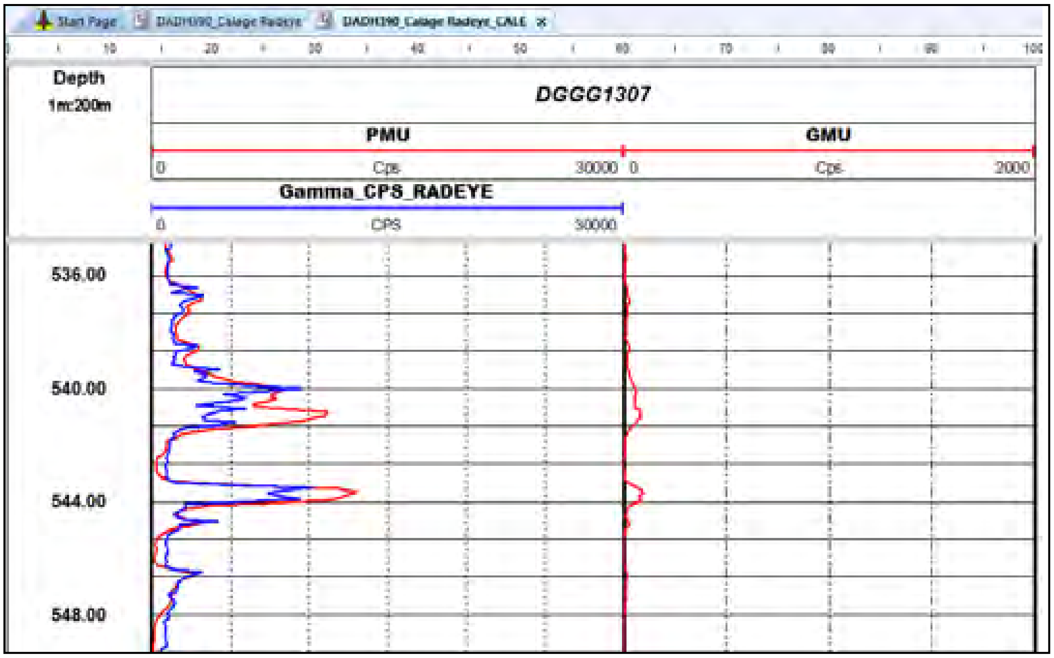


Figure 11-2: Hole DADH390, RadEye (blue) vs. Probe (red) Depth After Rescaling.

Source: GAC.

The modified scintillometer data depths were the corrected depths of the core. The new corrected depths were then exported from WellCad and used for sampling (Table 11-1).

Table 11-1: Depth Rescaling and Core New Depth (from GAC report, 2019).

DADH390 RADEYE / PROBE RESCALING					
Core initial depths		Shift	Core depths after rescaling		Gamma
From	To		From	To	
(m)	(m)	(m)	(m)	(m)	CPS
536.00	536.10	0.50	536.5	536.6	108
536.10	536.20	0.50	536.6	536.7	69
536.20	536.30	0.50	536.7	536.8	178
536.30	536.40	0.50	536.8	536.9	143
536.40	536.50	0.50	536.9	537	112
536.50	536.60	0.50	537	537.1	99
536.60	536.70	0.50	537.1	537.2	101
536.70	536.80	0.50	537.2	537.3	113
536.80	536.90	0.50	537.3	537.4	108
536.90	537.00	0.50	537.4	537.5	81
537.00	537.10	0.50	537.5	537.6	82
537.10	537.20	0.50	537.6	537.7	71
537.20	537.30	0.50	537.7	537.8	74
537.30	537.40	0.50	537.8	537.9	69
537.40	537.50	0.50	537.9	538	69
537.50	537.60	0.50	538	538.1	71
537.60	537.70	0.50	538.1	538.2	77
537.70	537.80	0.50	538.2	538.3	73
537.80	537.90	0.50	538.3	538.4	79
537.90	538.00	0.50	538.4	538.5	82
538.00	538.10	0.50	538.5	538.6	159
538.10	538.20	0.50	538.6	538.7	118
538.20	538.30	0.50	538.7	538.8	80
538.30	538.40	0.50	538.8	538.9	84
538.40	538.50	0.50	538.9	539	77
538.50	538.60	0.50	539	539.1	82
538.60	538.70	0.50	539.1	539.2	71
538.70	538.80	0.50	539.2	539.3	107
538.80	538.90	0.50	539.3	539.4	233

The Qualified Person accepts that the depth adjustment is a required procedure so that the probe data could be directly compared with the assay data, and correct depths are used for the MRE. The applied depth correction methodology appears to be reasonable to the Qualified Person.

11.5. Bulk Density Data

During the 2012–2015 drilling campaigns, GAC hired the ISO 17025 certified laboratory SAHEL Lab in Niger to perform bulk density tests on core samples. A total of 3,594 core samples sizing about 5 cm each were submitted during the 2012–2015 period and this gives an average bulk density value of 2.36 t/m^3. The bulk density of 2.36 was thus used for the current study.

The SAHEL Lab bulk density test of these samples was determined by the water displacement method. This method consists of weighing the sample in air after covering it with wax, and then measuring its apparent volume through water displacement.

The water displacement is noted, and the sample apparent volume determined (v). The bulk density is then calculated by bulk density = m/v. A relative error (E) is also calculated by Sahel Lab using this formula

$$E = |dm/m - dv/v|$$

Where:

- dm – the precision of the weighing scale used (0.001 g).
- dv – the precision of the cylinder used (1 ml).

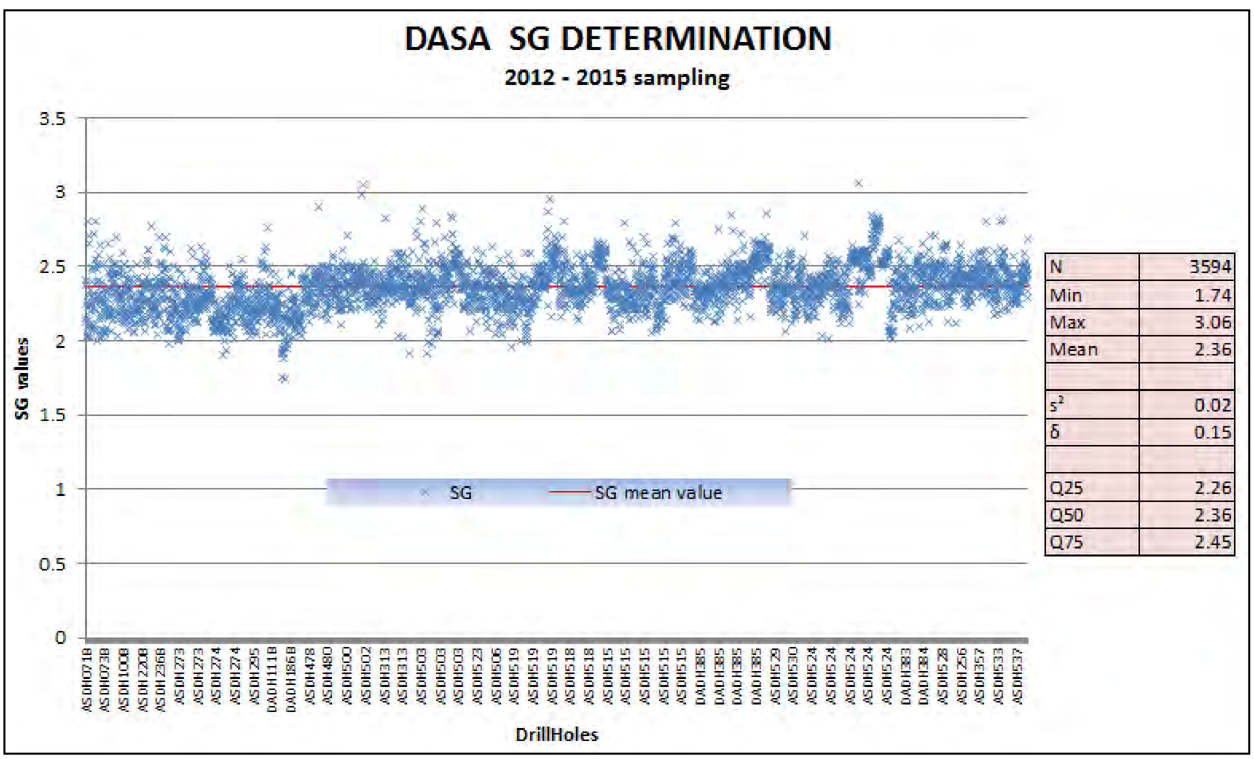


Figure 11-3: Average Bulk Density Determination from Core Samples.

Source: GAC.

11.6. Quality Assurance and Quality Control Programs

Pre-2018 Sampling Programs

Quality assurance (QA) and Quality Control (QC) measures are typically put in place to ensure the reliability and trustworthiness of exploration data. The QA measures include written field procedures to ensure reliable and systematic performance during logging, drilling, surveying, sampling and data management and database integrity. These QA procedures are just as important as the QC protocol to test the precision and accuracy of the data collected. Appropriate documentation of QA and QC measures and regular analysis of QC data are important as a safeguard to ensure the data collected during exploration is reliable and fit for purpose.

QC for analytical data typically involves internal and external laboratory control measures implemented to monitor the precision and accuracy of the sampling, preparation, and assaying. They are also important to prevent sample mix-up and to monitor the voluntary or inadvertent contamination of samples.

Five different reference materials were employed and sent to the assay laboratory for analysis – these reference samples were labelled such that it was not evident to the laboratory that there were reference samples. Field duplicate and blank samples were also inserted into the assay stream. The QC programs also included a small check assaying program at the SGS laboratory in Lakefield, Canada, which is ISO/IEC 17025 accredited. Check assaying program is not undertaken on an ongoing basis.

Comparison of ordinary assays of CRM material samples with control limit parameters is shown in Table 11-2. Results show that the quality of sampling and assaying is acceptable. Comparison of original samples and duplicates is provided in the QA/QC reports on Dasa Project (GAC 2012, 2013) (Figure 11-4).

Table 11-2: Comparison of Ordinary Assays of CRM Samples with Passport Parameters.

Number of CRM	Parameters of CRM			Ordinary Assays of CRM samples			
	LL	Nom.	UL	NN	Minimum	Average	Maximum
ALS_JOHANNESBURG							
AMIS0028	4,200	4,670	5,150	5	4,220	4,382	4,590
AMIS0054	1,320	1,470	1,620	8	1,385	1,456	1,480
AMIS0090	809	903	997	4	880	884	889
AMIS0098	774	848	922	8	850	855	865
AMIS0114	491	550	609	9	521	538	542
GBM908-5	4	5	5	14	4	15	128
GEOMS-03	3	4	4	4	3	4	5
MRGeo08	5	6	6	19	4	6	9
SARM-98	181	205	230	12	198	200	205
UTS-1	44	49	54	10	36	45	52
ALS_VANCOUVER							
BL-1	210	220	230	89	200	216	231
BL-4a	1,241	1,248	1,255	88	1,205	1,238	1,290

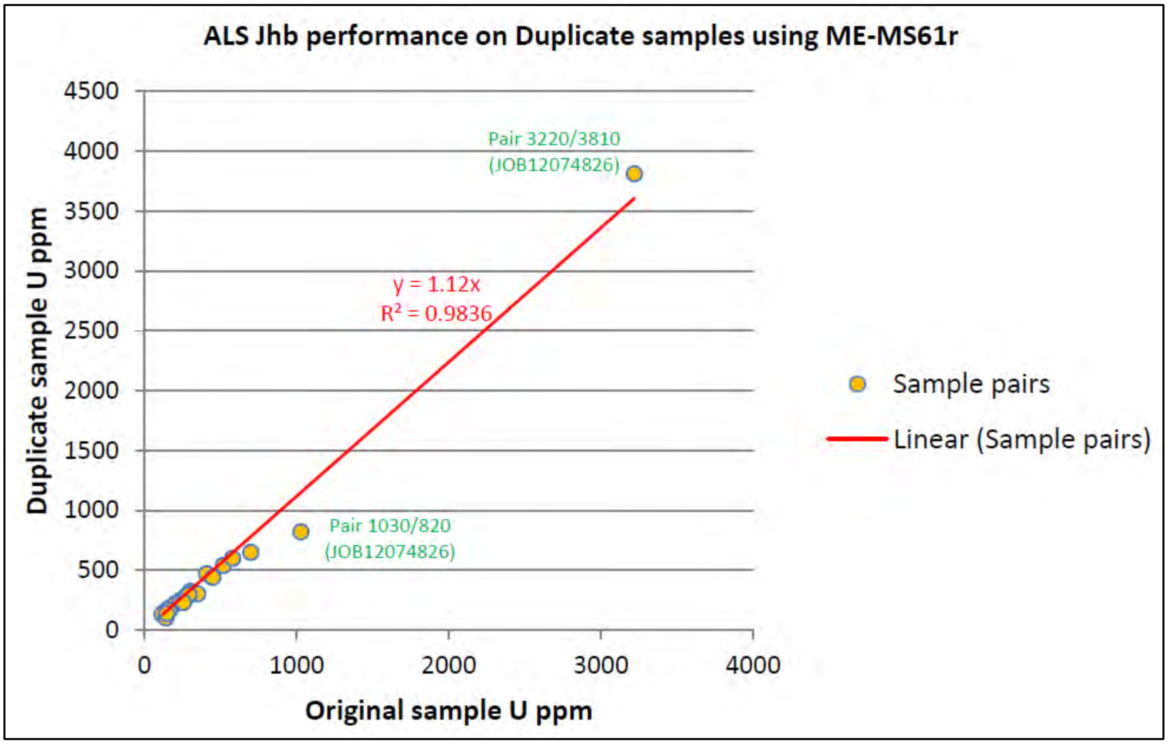


Figure 11-4: ALS JHB Performance on Duplicate Samples Using ME-MS61r.

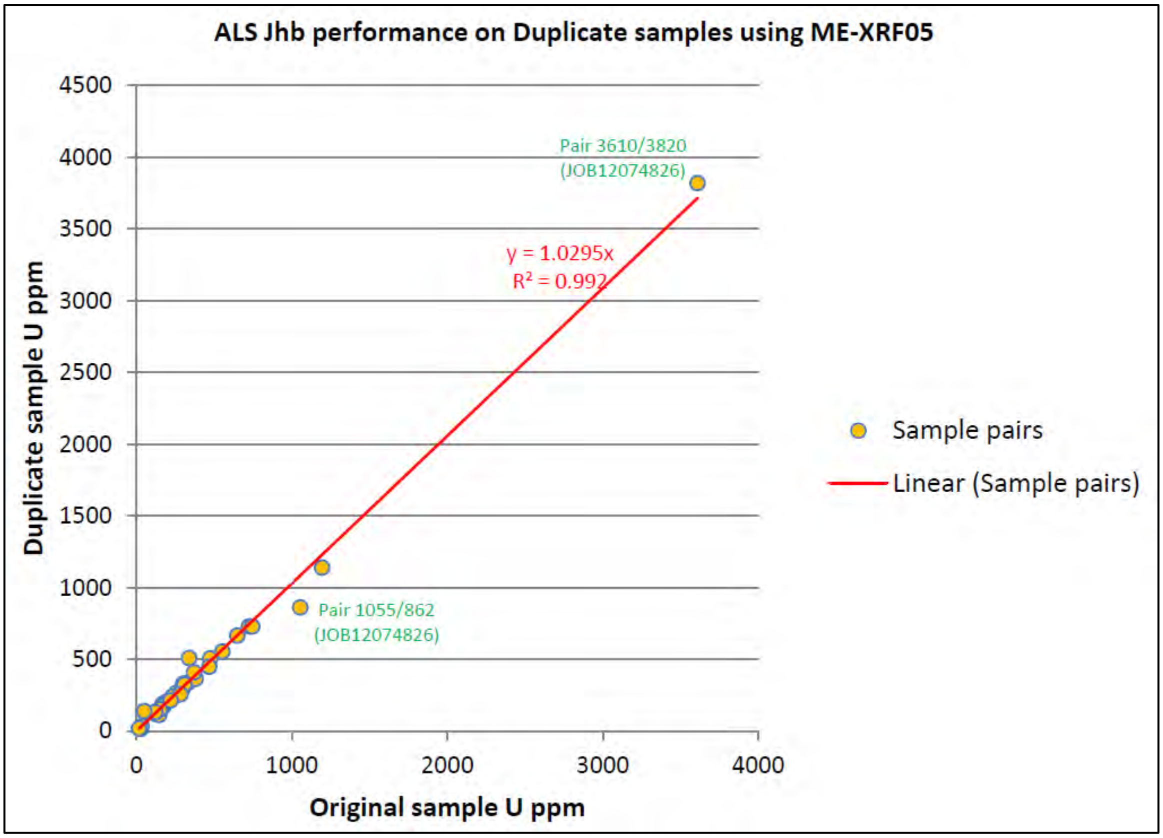


Figure 11-5: ALS JHB Performance on Duplicate Samples Using ME-XRF05.

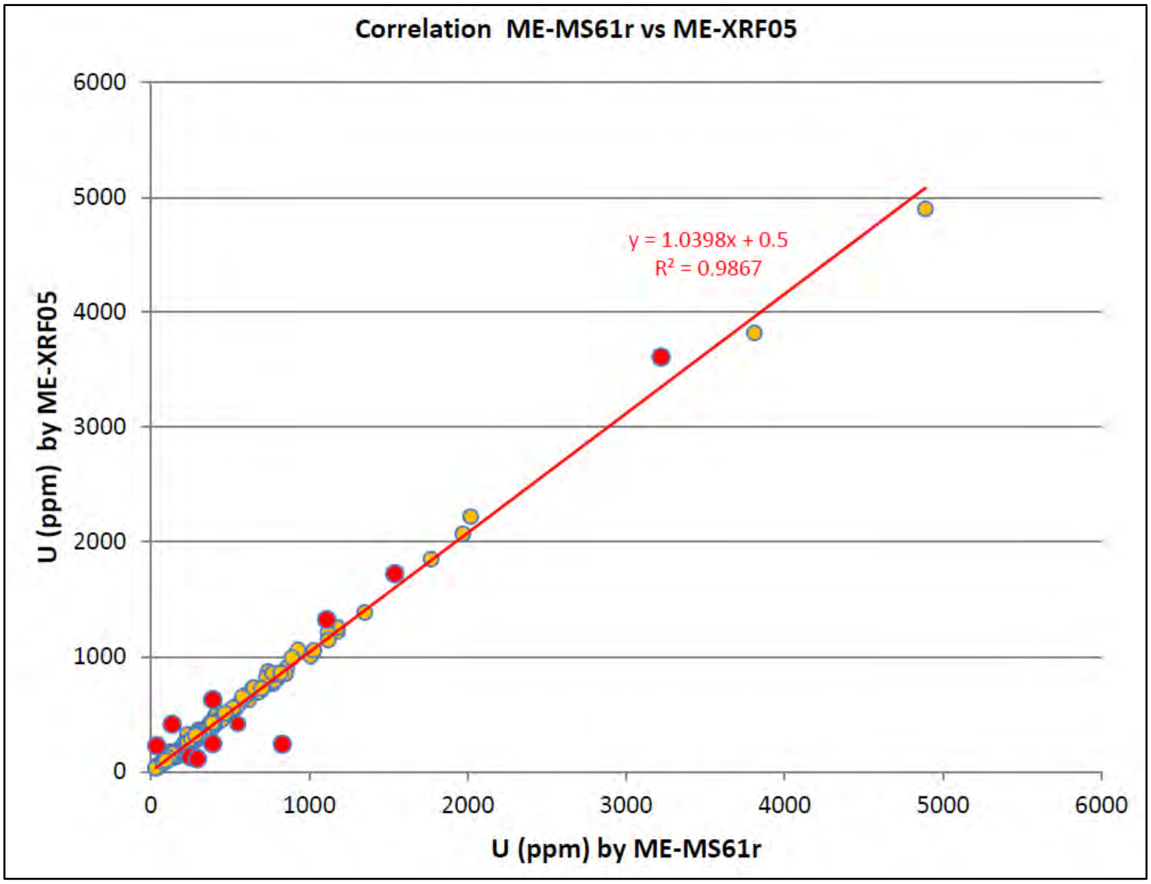


Figure 11-6: Comparison of the Original Samples and Duplicates for the Dasa Project. Source: GAC.

11.7. 2018 Sampling Program

Summary

GAC prepared and provided a full QA/QC report describing the applied methodology and all results (Christophe, 2019). The results from the report are summarized in this section of the Report.

The 4,983 samples submitted to ALS for assay by XRF (Press pellet analysed by wavelength dispersive XRF – ALS code ME-XRF05) for uranium then by fusion XRF method (coded ME-XRF10) for the samples grading above 1,500 ppm U. Selected hole samples were assayed for 48 elements plus rare earth elements (REEs) by four-acid digestions ICP (48 elements + REEs by HF-HNO3-HClO4 acid digestion, HCl leach followed by ICP-AES and ICP-MS analysis). The method is coded ME-MS61r by ALS.

As a result of the program, 4,731 samples were assayed by ME-XRF05, 900 samples by ME-XRF10 (252 directly and 648 of the 4,731 by ME-XRF05) and 1,571 by ME-MS61r.

Seven samples that were higher than the detection limit range after ME-XRF10 by ALS Vancouver were transferred by ALS to SGS Lakefield, Ontario to be assayed by a high-grade determination XRF method (coded GC_XRF76B by SGS) able to assay >15% U.

Blanks, Sample Preparation Control (Sahel Laboratory)

To control the sample preparation laboratory contamination level, a barren coarse quartz sample (collected south of Niamey in the granitic basement context) was inserted after every 40 samples of core to be crushed and pulverized. A total of 91 samples were inserted. The grade of the quartz is not certified but expected to not include significant uranium. Most of the samples returned grades below 50 ppm U, and nine sample grades were above this indicative value of 50 ppm U. This may suggest a level of contamination of these nine samples. It was noted that samples 16169, 16849, 17319 and 20850 each followed high-grade (>1.5% U) samples. This may indicate that the crushers were not cleaned well before the blanks were processed. The contamination for these four samples was less than 1%, which the QP believes is still within the acceptable limits. GAC investigated those samples and took measures to ensure that it does not happen in the future.

Pulp Duplicates at ALS

A total of 104 core samples were duplicated to check the sampling quality, the laboratory consistency, but also the repeatability of the grades. The duplicates pairs results indicate very low variation of the grades both for the ICP and XRF assay. The correlation coefficients are close to one, with all assaying methods indicating a good performance of ALS on duplicates samples (Figure 11-7 and Figure 11-8).

With the assaying methods ME-MS61r and ME-XRF05, the ratio Original/Duplicate is relatively dispersive above 5,000 ppm U, this indicates that above this limit, the methods may not be suitable to assay accurately the relatively high grades. Therefore, GAC re-assayed the grades above 1,500 ppm U using the method ME-XRF10 (Figure 11-9).

The duplicates were analysed by HARD (Half Absolute Relative Difference) plots to assess the sampling error and control the assay quality. Table 11-3: Compilation of HARD % per Assaying Method (Christophe, 2019).is a compilation of selected HARD values for the three assaying methods.

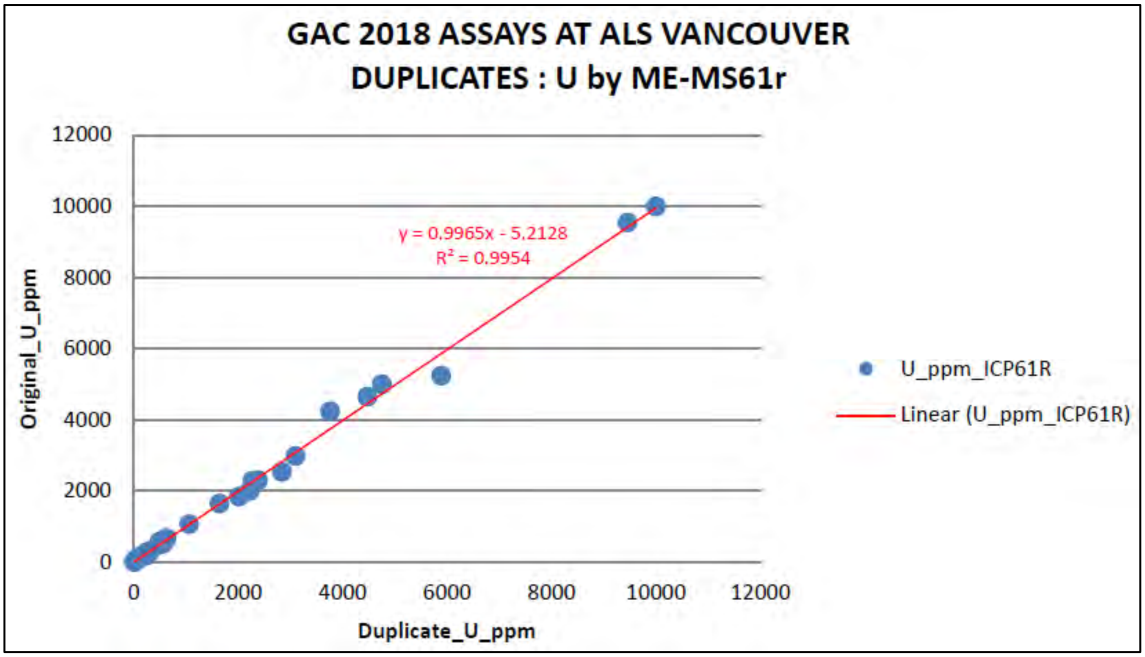


Figure 11-7: Duplicates Correlation Plot for U, ME-MS61r.
Source: Christophe (2019).

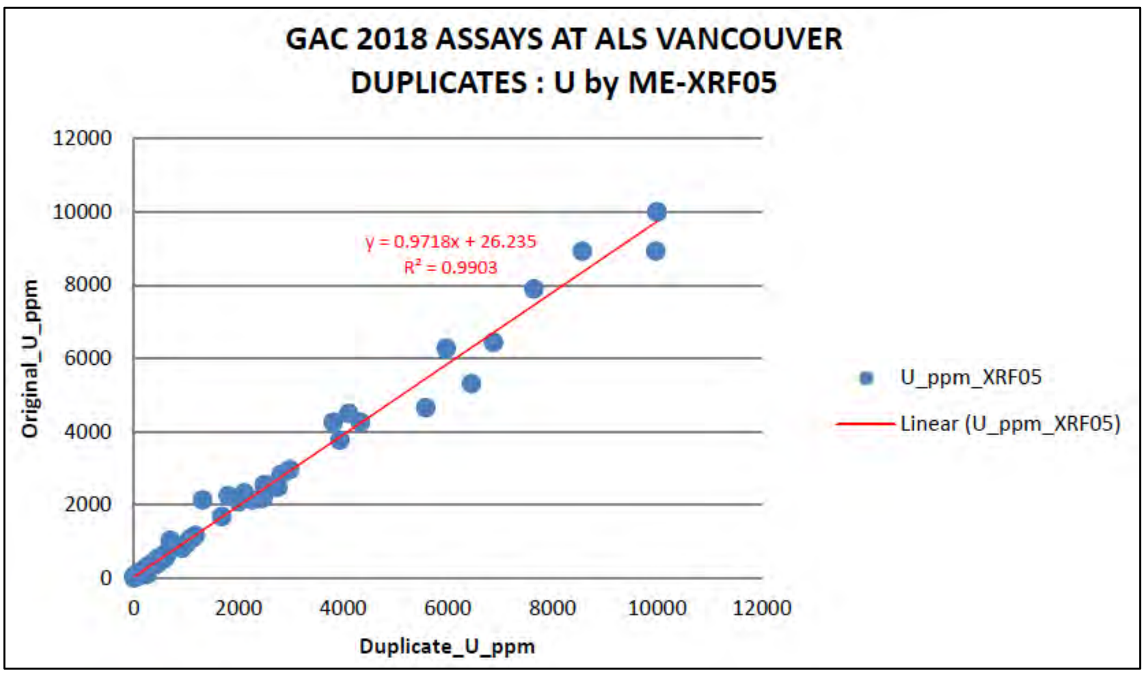


Figure 11-8: Duplicates Correlation Plots for U, ME-XRF05.

Source: Christophe (2019).

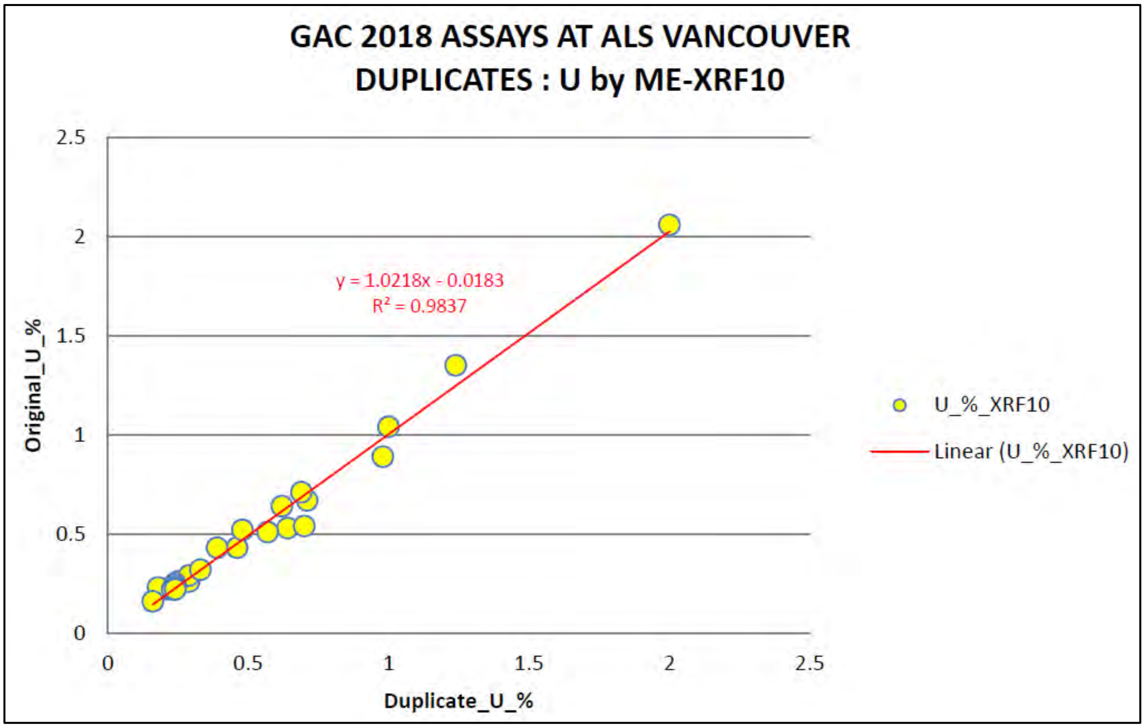


Figure 11-9: Duplicates Correlation Plots for U, ME-XRF10.

Source: Christophe (2019).

Table 11-3: Compilation of HARD % per Assaying Method (Christophe, 2019).

	HARD %	Non-filtered data	Filtered data (Duplicate value <3DL is removed)
		Population proportion (%)	Population proportion (%)
U by ME-MS61r	10	66	
	25	90	
U by ME-XRF05	10	66	68
	22	90	90
U by ME-XRF10	10	81	
	14.2	90	

Long (1998) recommends that for coarse rejects, agreement of ±20% on 90% of pairs is desirable, whilst for pulp duplicates, agreement of ±10% on 90% of the pairs is required. The ranked Absolute Relative Deviation (ARD) is used by Long (1998) to monitor its precision. Applying this, the reference precision should be 20% as recommended by Long (1998).

For the method ME-MS61r, the HARD plot shows 90% of the data are within 25% error, which is close to the acceptable 20% (Figure 11-10) .

Similarly for the correlation plots, one can observe that the errors are minimized with the assaying methods ME-XRF05 and XRF10 indicating these are increasingly better (Figure 11-11 and Figure 11-12). For the method ME-XRF05, the HARD plot shows 90% of the data are within 22% error: three Duplicate pairs inferior to the 3DL were removed from the dataset to minimize the small samples bias.

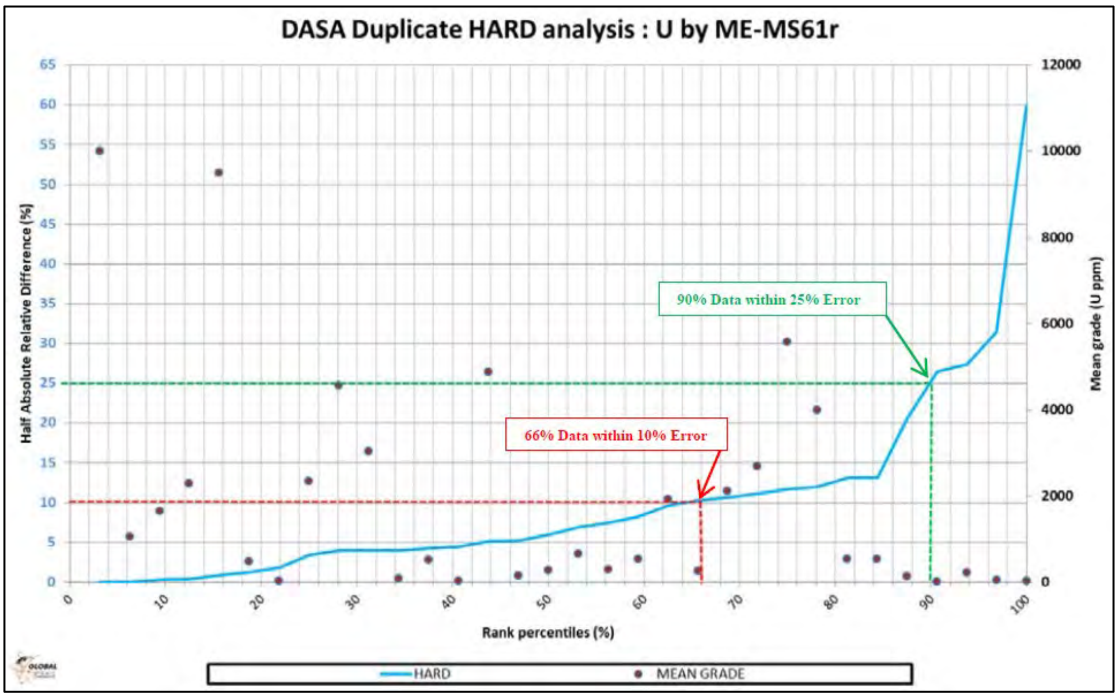


Figure 11-10: HARD Plot for ME-MS61r.

Source: Christophe (2019).

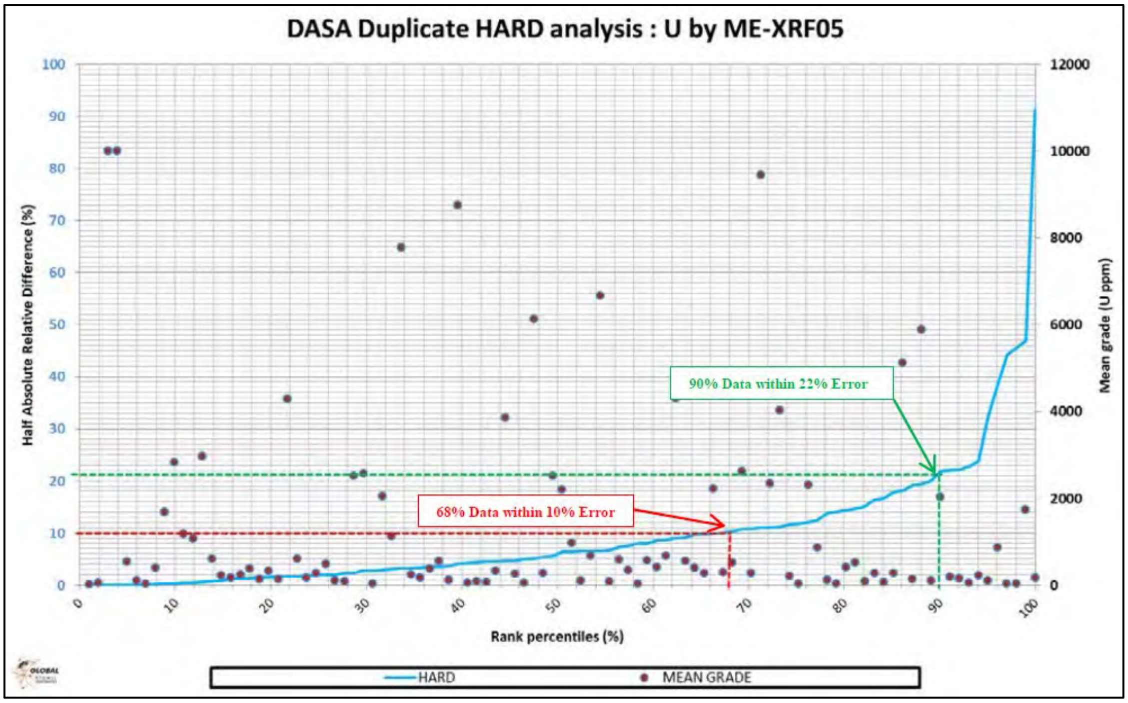


Figure 11-11: HARD Plot for ME-XRF05.

Source: Christophe (2019).

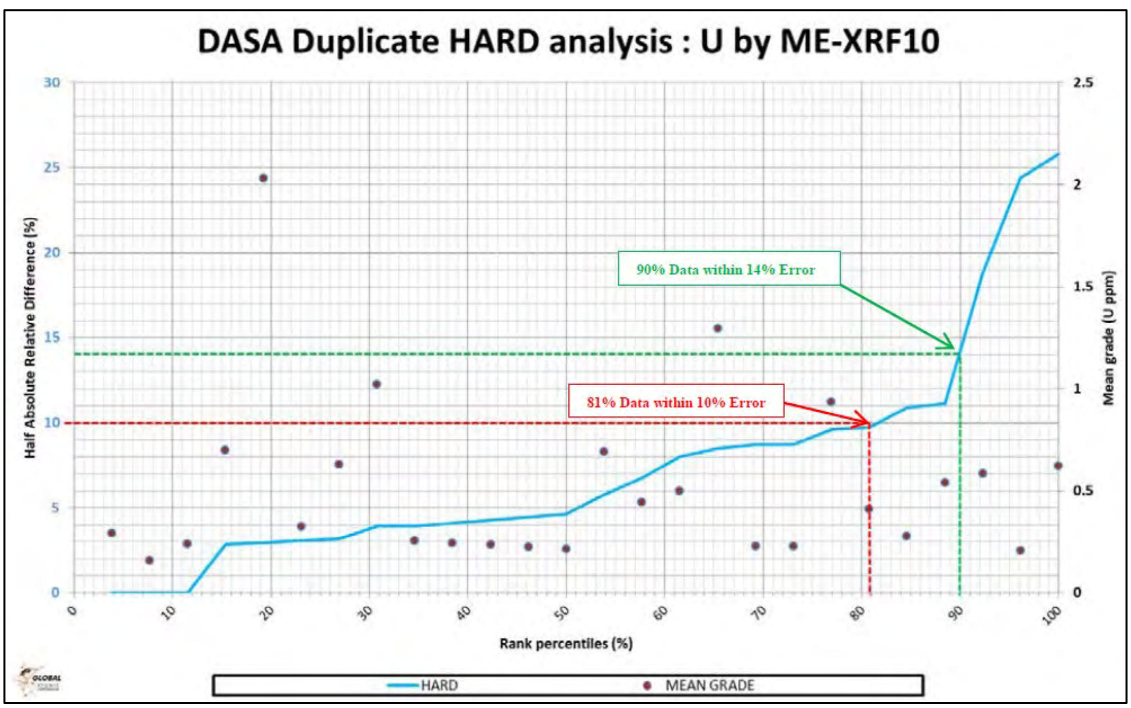


Figure 11-12: HARD Plot for ME-XRF10.

Source: Christophe (2019).

The Qualified Person concludes that the results of the duplicate analyses are acceptable – the majority of the samples will have a high likelihood to reproduce their grade with minor error in the event of re-assay, and the fusion XRF method is the best assaying method to honour the grades of the samples collected.

Certified Reference Materials and Blanks at ALS

GAC used five different types of CRMs from the Australian company, Ore Research & Exploration Pty Ltd (ORE) as control material. Detailed parameters are shown in Table 11-4 below.

Table 11-4: Parameters of the Standards and Blanks Used.

Standard type	XRF Fusion (ppm U)	XRF SD (ppm U)	ICP Four-acid (ppm U)	ICP Four-acid SD (ppm U)
Blank OREAS22e	0.13	0.04	0.13	0.04
Standard OREAS121	215	9.6	206	7.1
Standard OREAS122	423	13	407	13.4
Standard OREAS123	858	29.7	825	35
Standard OREAS124	1,845	40	1,779	89.7

Results of the blank analyses were presented for 35 batches for ME-MS61r and 36 batches for ME-XRF05 methods. The demonstrated level of contamination was acceptable. In addition, ALS internal blanks results were within the expected limits and did not indicate any instrument drift or calibration issues.

The GAC report on QA/QC analysis included all the performance charts for each standard (Christophe, 2019).

Standard OREAS121 was inserted 53 times. Values returned for this CRM using the ME-MS61r method were all within the certified mean value plus or minus one standard deviation (SD), except for one sample. The assays values for this standard using the ME-XRF05 technique were almost always in the interval mean grade plus one SD value, thus demonstrating slight over-estimation of the standard value.

Standard OREAS122 was inserted 35 times. The grades returned by ALS using the ME-MS61r technique were mostly around the mean grade plus/minus one SD value. Only one sample was slightly over-estimated, 1 ppm U above the mean grade plus two SD value. Except for one sample, all the grades returned by ALS using the ME-XRF05 technique were mostly around the mean grade plus/minus one SD value.

Standard OREAS123 was inserted 56 times. For the ME-MS61r technique, the returned grades were erratic but all around the mean grade. With the ME-XRF05 method, the grades returned are slightly trending above the certified mean grade of the standard but always below the CRM mean grade plus one SD, except one sample where the grade is slightly above the CRM mean grade plus one SD but below the accepted mean grade plus two SD value.

Standard OREAS124 was inserted 54 times. All grades returned are more regularly around the mean grade with the ME-MS61r technique. U ppm grades returned with the ME-XRF05 technique were generally slightly above the CRM mean grade, but all below the mean grade plus one SD value.

Analysis of the CRMs and blanks demonstrated that an insignificant number of samples were outside of the upper or lower warning limits; however, the analysis of the results did not reveal any significant bias that could be introduced by the main laboratory. The qualified person believes that the results of CRM and blanks analyses are within acceptable limits.

11.8. 2021-2022 Sampling Program

Summary

This is to report results from two batches of samples sent to ALS VANCOUVER in March and May 2022.

A total of 993 pulp samples (samples number 21351 to 22343) including 10% of control material were submitted: 2% of Duplicate samples (20 samples), 2% of preparation blank (19 samples), 2% of blank (certified grade, 20 samples) and 4% of certified reference material (40 samples standard).

The first batch was assay using exclusively Fusion method XRF while the second batch was first assayed by Press pellet XRF then the grades above 1500 ppm U were assayed by Fusion XRF. ALS released results into four batches, the last results were received on May 20th, 2022. Below is the summary table of the samples.

Table 11-5: 2022 Sampling Campaign Samples.

Received at Lab Date	Assay Date	Certificate Received Date	Job Number	# Samples Reported	XRF05	XRF10
8-Mar-22	7-Apr-22	7-Apr-22	VA22059577	250		250
9-Mar-22	9-Apr-22	9-Apr-22	VA22059583	254		254
2-May-22	20-May-22	20-May-22	VA22114548	250	250	94
2-May-22	20-May-22	20-May-22	VA22114553	239	239	76
			Total	**993**	**489**	**674**

The turnaround for the first batch was 30 days (job number VA22059577 and VA22059583) while the second one turnaround was 18 days (job number VA22114548 and VA22114553).

The results indicate a general grade increase of about 6% from the probe calculation.

Sample Preparation

Before pulp samples being obtained then sent to ALS for analysis, core samples were sent to Sahel Lab in Niger for mechanical preparation. Core samples were two stages crushed using jaw crusher from where a 1/8 split sample is taken to be pulverized. The remaining 7/8 coarse reject is put in plastic bag, tagged with the received sample number, and stored to the lab for now, waiting to be transferred to site at a given occasion. In the past, a selective granulometric analysis made on some samples show that more than 97 % of the 1/8 sample split passed 2 mm sieve, this is what we still consider. The 1/8 split sample is then pulverized to a nominal 75 µm using a tungsten steel bowl, the objective being 90% passing the 75 µm sieve (>85% requested by ALS VANCOUVER). 30 g pulp is sampled from the obtained powder to be sent to ALS VANCOUVER for analysis. The remaining sample is packed in plastic bag, tagged as received and stored at Sahel Lab by the time to be transferred to the site.

Analysis

ALS QA/QC on Received Pulps

The first batch pulp samples received at ALS VANCOUVER (samples number 21351 to 21854) failed to pass the sample lab preparation QA/QC process. Each sample is weighed and sieved to comply with ALS standard about pulp grain size: 85% of pulp sample should pass through the 75 µm sieve to be qualified. As consequences, all the 504 samples in this batch were entirely pulverized by ALS VANCOUVER to meet its requirement before assaying.

The situation was addressed to Sahel Lab in Niger and necessary action made to solve that. The second batch samples were thus reviewed by Sahel before being sent to ALS VANCOUVER. They all passed the sample preparation QA/QC validation by ALS VANCOUVER, no warning reported from the lab.

Analysis

Samples 21351 to 21854 were analyzed by fusion XRF for U (detection limit 0.1% U), method called ME-XRF10 by ALS.

The second batch samples were analyzed by ME-XFR05 (Pressed powder pellet analyzed by wavelength dispersive XRF) for Uranium, then the samples with grade above 1500 ppm U are re-analyzed by XRF10 method. A total of 170 samples over 489 in the batch were analyzed by XRF10.

Results / Interpretations

QA/QC on Sahel Lab Sample Preparation

The ALS VANCOUVER sieving to check the quality of Sahel Lab sample preparation is one level of control. But more importantly, we are including 2% of Preparation blank (PREP BLANK) to control the level of possible contamination of the lab. The PREP BLANK is made of quartz sample grabbed nearby Niamey and inserted in the batch of the to be prepared samples. The quartz is known to not carry Uranium, and the zone of Niamey is not a Uranium rich environment.

The results of the PREP BLANK for the first batch show a level of minor contamination.

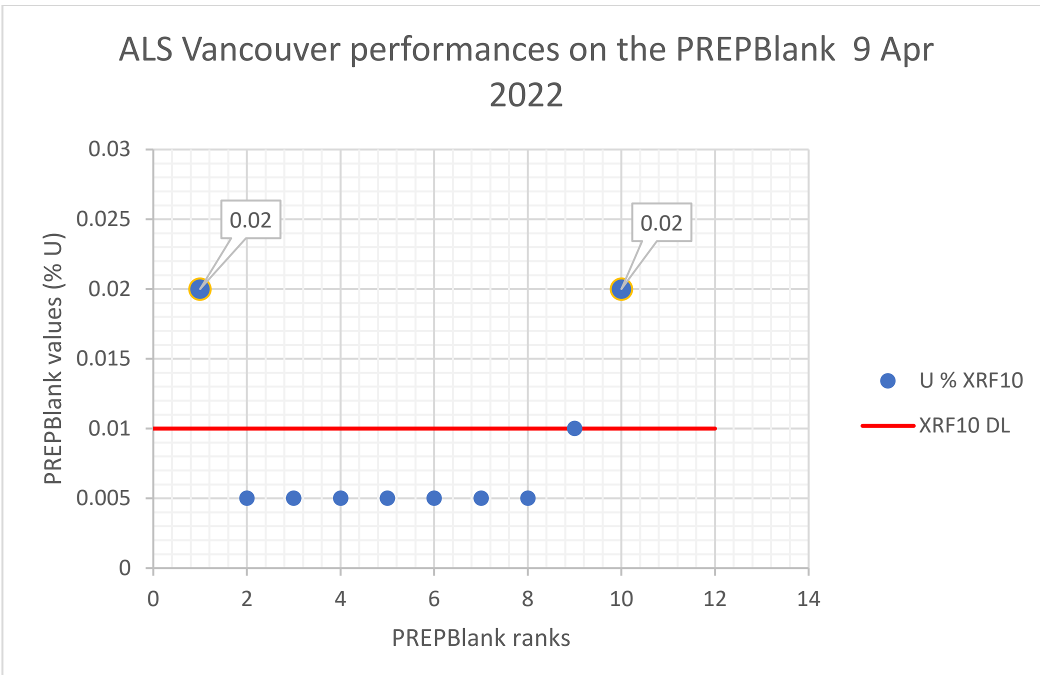


Figure 11-13: XRF10, ALS performance on PREP BLANK.

Source: Christophe (2022).

The level of contamination here is marginal but indicates approximate quality of Sahel Lab preparation: the two samples grading 0.02% U are all occurring just after or in a set of higher-grade samples preparation. A control of the lab processing show it appear more importantly during the crushing where the bowl seems to not be properly flushed before introducing the next sample.

The situation was addressed to the Lab, and they performed better on the next batch samples where there was no real contamination noticed. The bowls were properly flushed and cleaned after each sample crushed; the tungsten bowl for pulverization were cleaned every time after each sample processed.

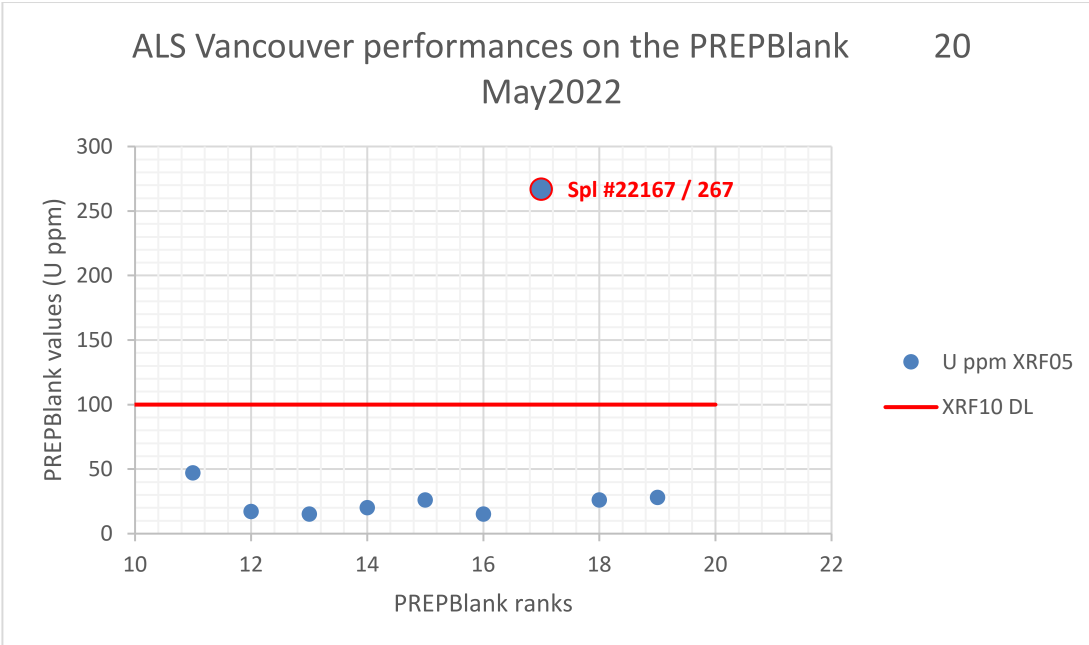


Figure 11-14: XRF05, ALS performance on PREP BLANK.

Source: Christophe (2022).

The sample 22167 seems to indicate a level of contamination. But a close control indicates a mishandling of data: in fact, the lab is using two pulverisers one using one bowl and the second one is using three bowls. The one that takes the three bowls, so having the capacity to pulverize at once three samples, is where the problem come from: a control of the pulp and the coarse reject shows that the sample 22167 was mislabelled 22170 and vice versa while being removed from the pulveriser.

A request was addressed to ALS VANCOUVER to check and confirm the results of an interval including sample 22167, but now this is solved: the sample 22167 real grade is 6 ppm U while the 22170 grade is 267 ppm U, inline with the core samples around.

This situation leads to think that this may happen elsewhere where we cannot prove. But the decision was taken to no more process three samples together in this pulveriser. It will slowdown the process, but at least, it will preserve the data integrity.

Data Handling at ALS VANCOUVER: Performances on Blank CRM

The performance of ALS Vancouver on the blank material inserted show that ALS VANCOUVER is well handling the samples and there is no contamination from their manipulations.

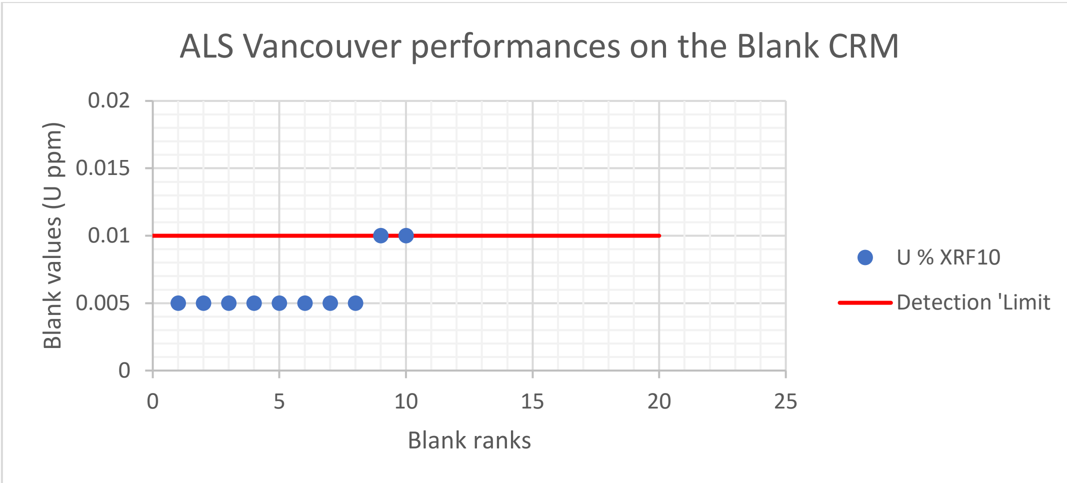


Figure 11-15: XRF10, ALS performance on BLANK OREAS22e.

Source: Christophe (2022).

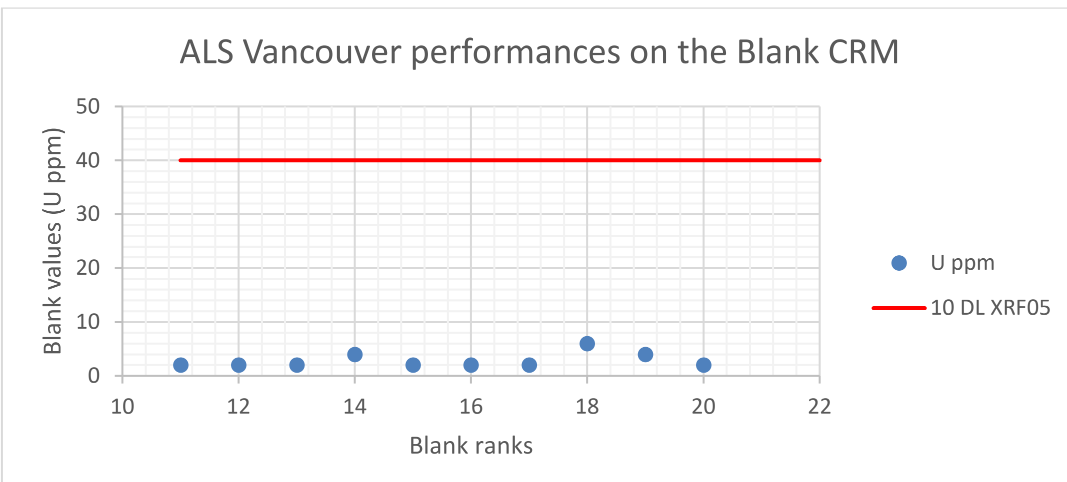


Figure 11-16: XRF05, ALS performance on BLANK OREAS22e.

Source: Christophe (2022).

Performances on Duplicates

Sample duplicates (a same pulp sample divided into two samples with one called original and the other duplicate) were introduced into the batch to be analysed for the lab consistency. The results show that ALS performs well on the duplicates analysis as there is reasonable correlation between original and duplicate samples. The correlation coefficient = 1 for both batch 1 and batch 2 results (Figure 11-13 and Figure 11-14).

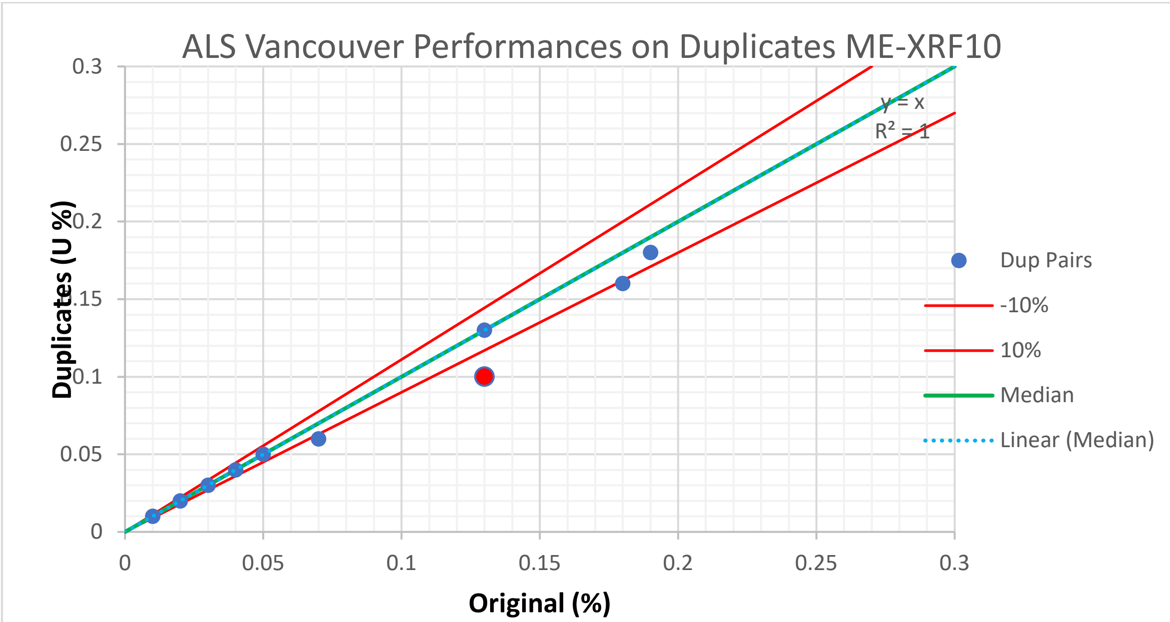


Figure 11-17: Duplicates Correlation Plot for ME-XRF10.

Source: Christophe (2022).

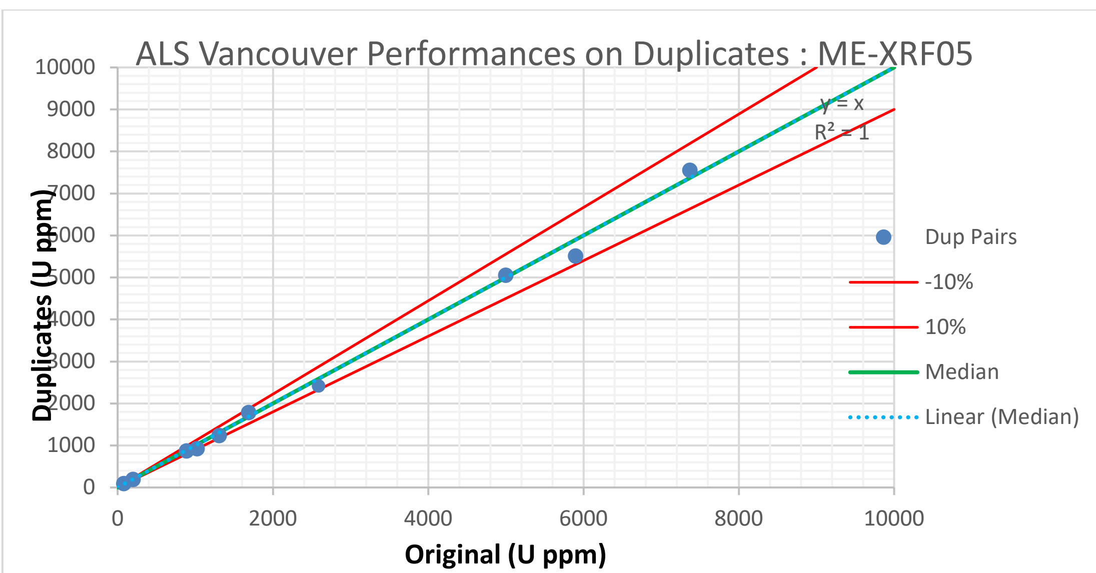


Figure 11-18: Duplicates correlation plot for ME-XRF05.

Source: Christophe (2022).

Only one pair of samples (Figure 11-11) is fully out of the 10% margin. A close check shows that the difference is 3 times the detection limit of the method, so not critical.

ALS performances on Certified reference materials

Five standards from the Australian company ORE (Ore Research & Exploration) were used to test the accuracy of the ALS lab analysis.

Table 11-6: Five Australian Standards Used to Verify ALS Laboratory Analysis Accuracy.

CRM Id	Analyse Method	Constituent	Certified Value (ppm U)
OREAS120	XRF Fusion	U ppm	41
OREAS121	XRF Fusion	U ppm	215
OREAS122	XRF Fusion	U ppm	423
OREAS123	XRF Fusion	U ppm	858
OREAS124	XRF Fusion	U ppm	1845

ALS VANCOUVER performances on these CRMs were erratic and generally underestimated for the first batch of samples (job number VA22059577 and VA22059583) using straight the XRF10 method for U analysis.

But the choice of doing first the XRF05 on all the samples, isolate the higher grades and avoid doing unnecessary the XRF10 method analysis on the lower grade’s samples. The performances on the CRMs in the second batch (job number VA22114548 and VA22114553) is better, the grades all being around the certified grades as shown below.

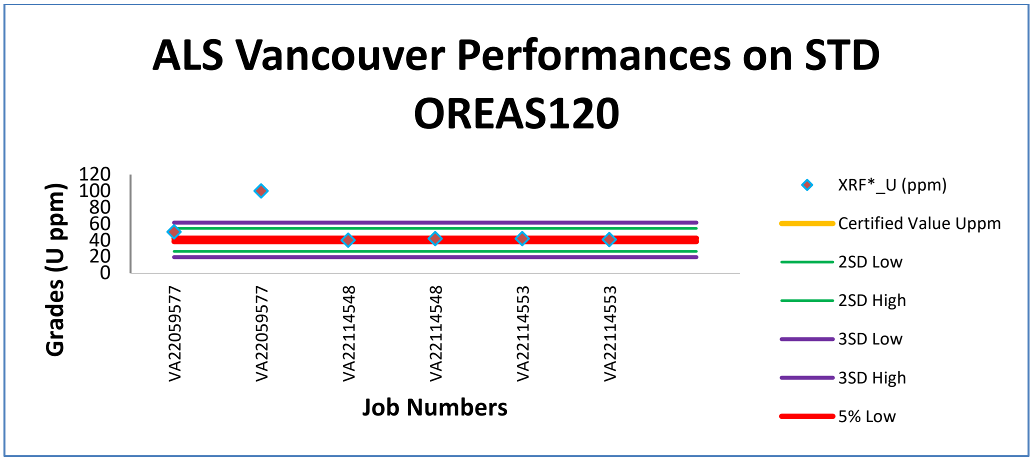


Figure 11-19: ALS VANCOUVER performance on CRM OREAS120.

Source: Christophe (2022).

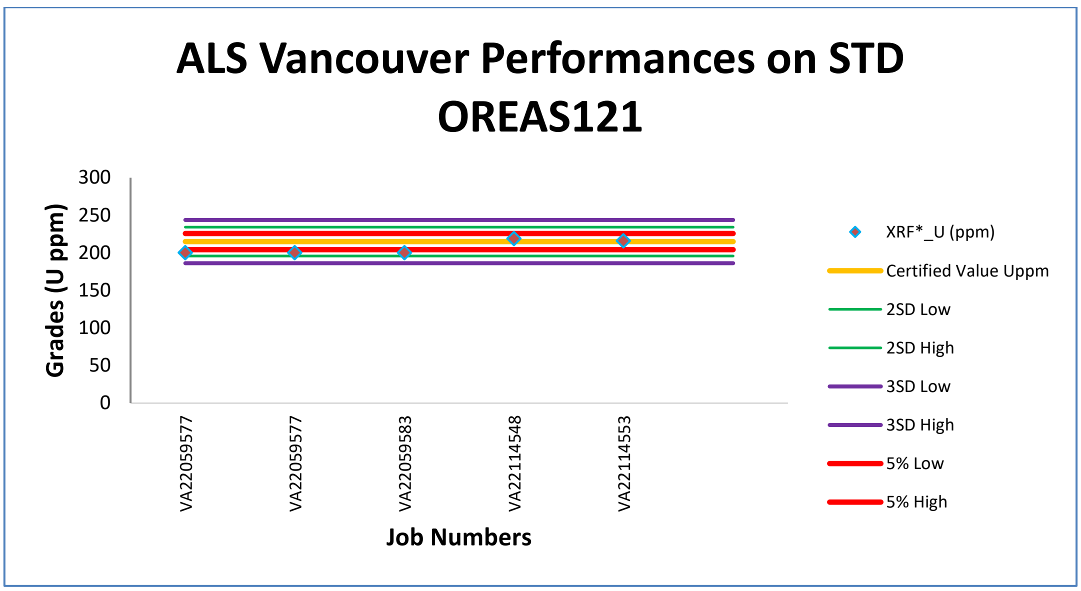


Figure 11-20: ALS VANCOUVER performance on CRM OREAS121.

Source: Christophe (2022).

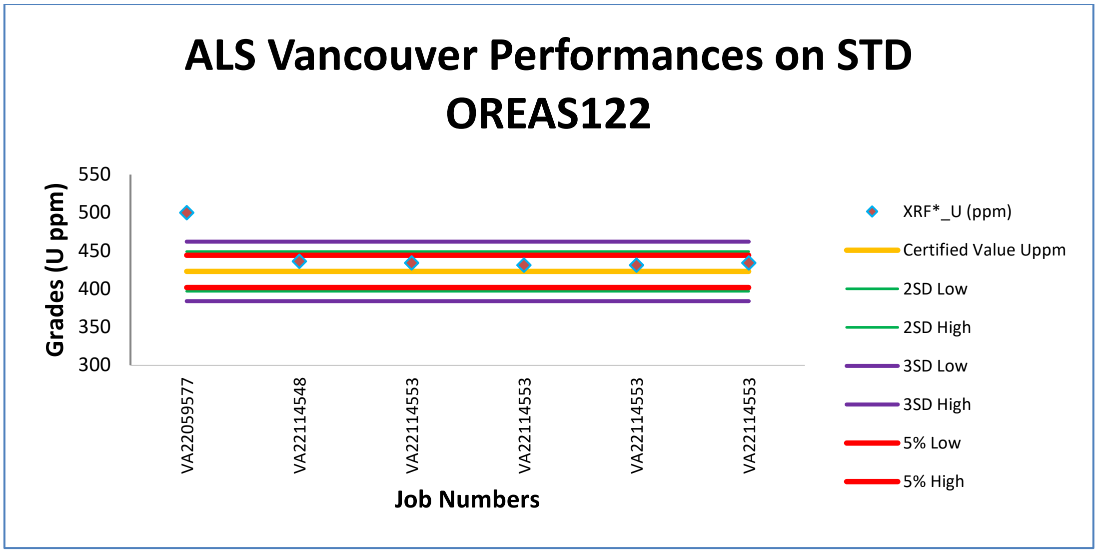


Figure 11-21: ALS VANCOUVER Performance on CRM OREAS122.

Source: Christophe (2022)

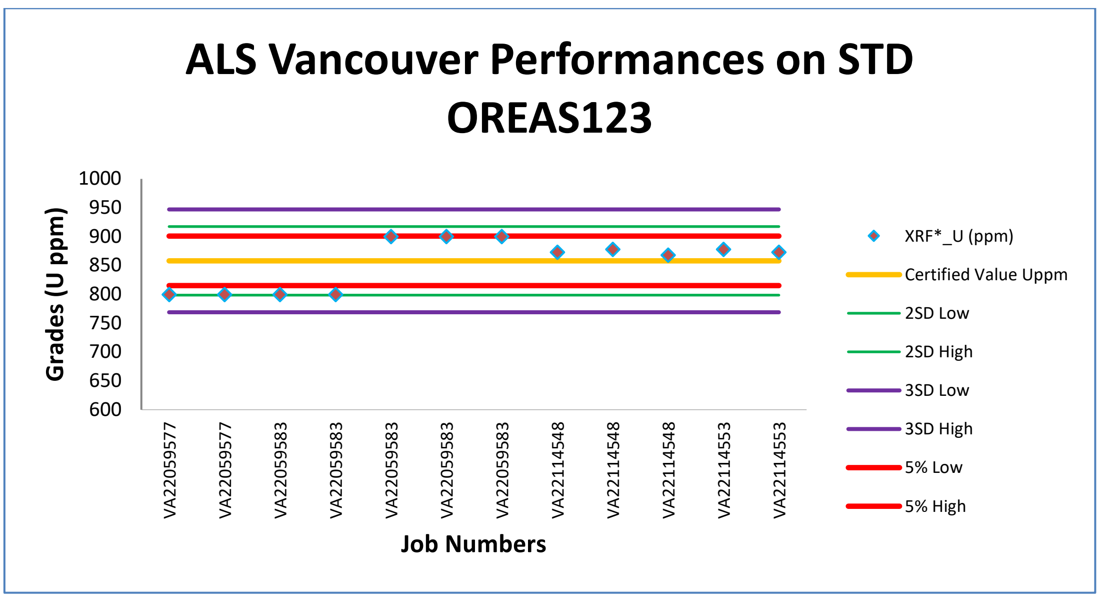


Figure 11-22: : ALS Vancouver Performance on CRM OREAS123.

Source: Christophe (2022).

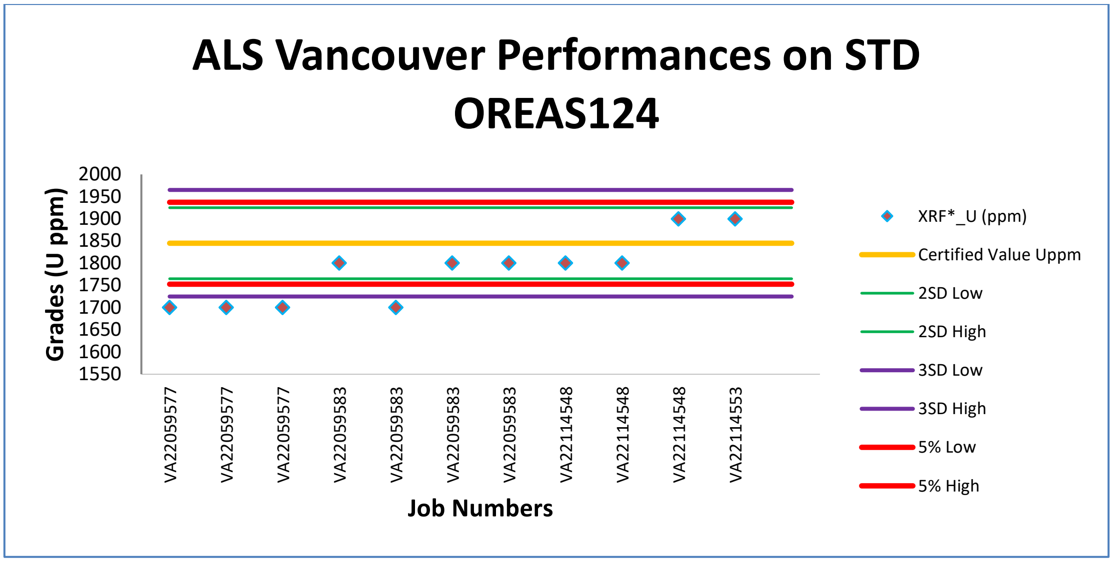


Figure 11-23: ALS VANCOUVER Performance on CRM OREAS124.

Source: Christophe (2022).

Conclusion Sampling Program 2021-2022

ALS Vancouver performances on the Blank show a good data handling at the lab. The good correlation on the Duplicates shows a consistency in the assaying and a homogeneity of the samples.

The approximate performances on the first batch samples are due to the low-grade CRMs we are using, below or very close to the XRF10 method detection limit: so, it is difficult to get clear understanding of the lab performance through our own CRM. The lab internal samples are giving good idea of the lab consistency, and we should rely on it.

The performances on the same CRM once the low grades are separates, show it was a good decision to proceed like that.

The results reported by ALS give an idea of the mineralization and comply somehow with the radiometric probe. A compilation of the grades shows a general increase of 6% from the initial probe equivalent U derived by Henri Sanguinetti.

Recommendation is made to acquire higher grade CRMs to efficiently control the labs assaying on DASA high grade materials.

Review of Downhole Probe Quality Assurance and Quality Control Programs

GAC provided detailed information on how the QA/QC protocol was followed during the 2017–2018-2022 exploration program (Christophe, 2019, 2022). The qualified person reviewed the procedures and has summarized them below:

- The probing (data collection) was done by a third party, using its own equipment. The eU3O8 was determined by another third party, different from the probing company.
- During the exploration campaign, all holes were probed using the following equipment:
- A Geiger Muller (GM) probe.
- A DIL probe.
- The Gyro probe for downhole deviation.
- The probing started immediately at the end of drilling of each hole. There was minimal time gap between the completion of drilling and commencement of probing. At the end of the last drilled metre, the drill fluid was renewed, and the hole was cleaned for about one hour before leaving the hole for the probing. The probing duration depended on the hole length: run in of DIL and GM is at 5–6 m per minute and pull out is at 3 m per minute for DIL, while the GM is 3 m per minute with speed reduced to 1 m per minute in the mineralized sections. The Gyro could go up to 20 m per minute.
- Raw data was sent after pre-processing (depth matching) to the external consultant for eU3O8 determination. The necessary steel and mud corrections were made by the consultant before returning grades based on the data provided by the probing contractor.

QA/QC logging included the following:

- At the start of the program, the probes were calibrated on a certified calibration pit.
- The probes were run on a third-party external test pit (reference pit on Orano Niger project in Arlit); and repeated on a 3 monthly basis.
- The first hole of the program and randomly selected holes were probed with all the available probes on the project for correlation between probes.
- All the Project probes were passed once a week on two standard holes. These standard holes of the Project were found to be lower grade with regards to the first hole's results, thus two high grade test holes were built for that purpose.
- If the third party required it, the test pits were revisited for verification.
- The quantity of control samples amounted to 10% of the exploration program in 2018 and is considered sufficient.

The Qualified Person is satisfied with the protocol applied.

Radioactive Equilibrium Factor

Geophysical gamma logging data is the primary information source used for uranium resources estimation. When Gamma logging is applied, it is possible to determine:

- Mineralized intervals.
- Conversion of radium grade to uranium grade based on REF and probe results.

Radioactive Equilibrium Factor (REF) = C (radium) / C (uranium) should be estimated using uranium assays and radium assays sampled into sealed cans. Until now, radium grades have not been determined and hence, comparison of the eU_3O_8 based on gamma logging and actual U_3O_8 based on assays can be used to estimate the REF. It is also possible to use the scintillometer readings made on the core to compare and to correct gamma logging data.

The database provided for chemical assays had 9,784 records in the file, of which 9,772 records had uranium or U_3O_8 grades. All U grades were converted to U_3O_8, and the file was then merged with the gamma logged intervals. It was found that 9,079 assayed intervals were also probed, and all other intervals did not have either the probe or the assays data results and, therefore, were excluded from the analysis.

All probed intervals were length composited within the limits of each sampled interval, and the resultant grades were then directly compared to estimate the presence of uranium disequilibrium factor.

The average length weighted U_3O_8 grade for all chemical assays was 1,316 ppm while the composited eU_3O_8 grades for the same intervals that were assayed had an average length weighted value of 1,231 ppm. The global difference was 6.4% and, therefore, the REF = 0.94.

Comparison of eU_3O_8 based on gamma logging and U3O8 based on assays shows acceptable correlation close to 1 (Figure 11-24), the coefficient of correlation is 0.88.

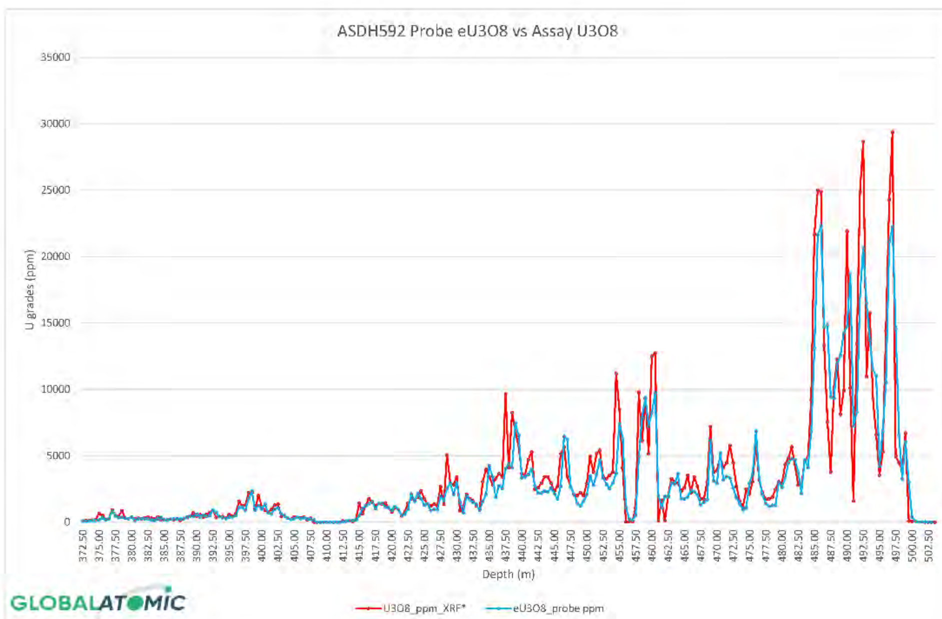


Figure 11-24: Correlation Graphs Comparing eU3O8 (Porbe) vs. U3O8 (Assay).

Source: Christophe (2022).

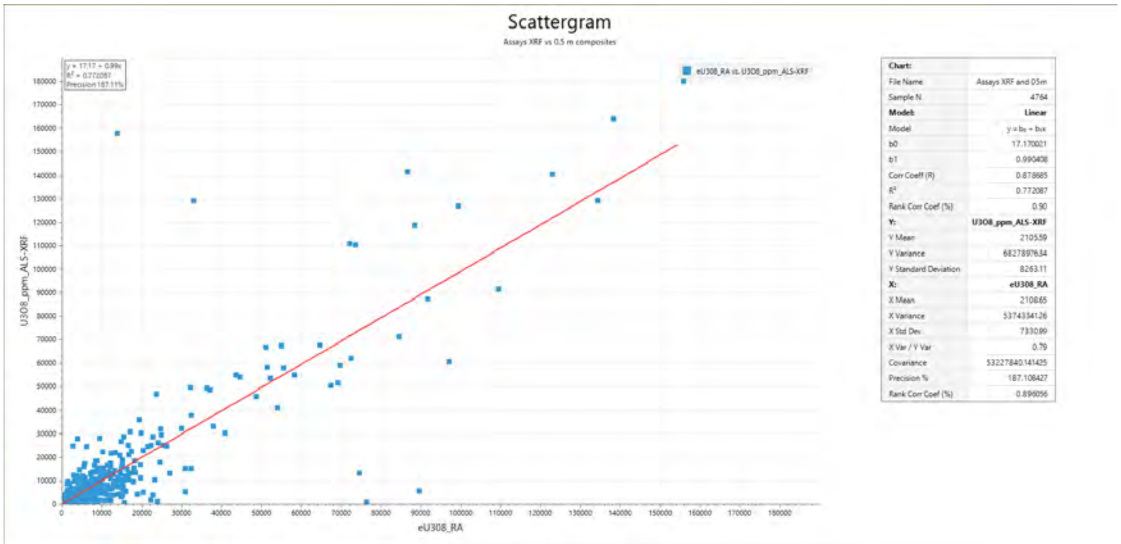


Figure 11-25: Scattergram Showing Results of XRF Assays Versus 0.5 m Composited eU3O8 Grades.

Coefficient of correlation 0.88. That demonstrates good correlation between XRF assays and 0.5 m composited deconvolved grades.

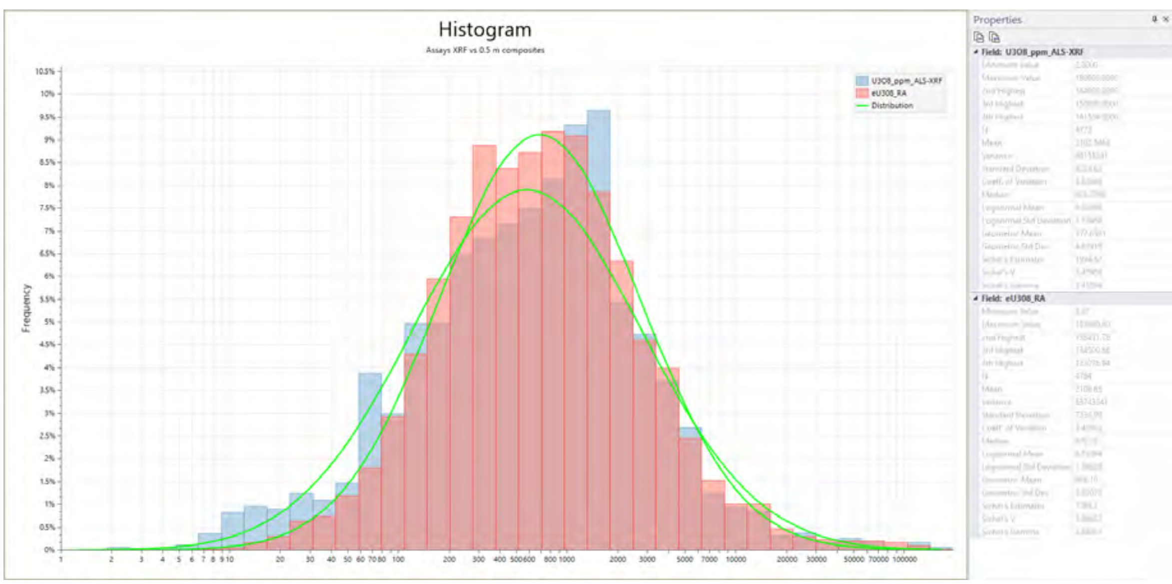


Figure 11-26: Histogram Showing Results of XRF Assays Versus 0.5 m Composited eU3O8 Grades.

Overlayed histograms of XRF results and 0.5 m composites of deconvolved grades show that both histograms are very similar. The mean grades for both data sets are almost identical (2103 vs 2109 ppm). That confirms the uranium disequilibrium factor is likely to be close to 1.0.

11.9. Comments by the Qualified Person

In the opinion of the Qualified Person, the sampling preparation, security, and analytical procedures used by GAC are consistent with generally accepted industry best practices and are therefore adequate for the purpose of Mineral Resource estimation.

Additional drilling programs completed in 2021-2022 in the most prospective and high-grade areas of the deposit, as well as an additional significant number of chemical assays analysed in 2021-2022 allowed more accurate calculation of eU_3O_8 grades, as well as a more robust estimation of uranium equilibrium factor. All of which allowed development of a more robust Mineral Resource model and upgrading of Inferred category to Indicated in some areas of the deposit, where the geological understanding and confidence in the model were supported by additional drilling.

Overall, the Qualified Person is of the opinion that GAC's QA/QC programs provide adequate confidence in GAC's collection and processing of the data.

Additional investigations are recommended to further support the estimated REF, including assaying of radium in closed cans and uranium by XRF. Comparison of radium and uranium assays allows more reliable definition of the REF and comparison of radium assays and gamma logging allows definition of a radon degassing factor. This factor may also influence the calculation of eU_3O_8 grades.

12. DATA VERIFICATION

Dmitry Pertel (author of the MRE and Qualified Person) visited the Project site from 20 March 2017 through to 6 April 2017, spending five days at the deposit site and exploration camp and several days in Niamey at GAC's office. During the visit, Dmitry Pertel reviewed geological reports, drilling procedures and surveys, logging facilities and overall deposit geology. Geological exploration drilling procedure, core recovery methods and documentation and geophysical logging have been analysed from the provided reports.

During the site visit, the Qualified Person observed several drill collars, took photographs of the drill collars, and recorded the geographic coordinates. The measured coordinates were compared with those reported in the provided database. The difference between the measured and reported coordinates were within acceptable limits.

From 2 to 4 April 2017, Dmitry Pertel visited the Sahel Laboratory in Niamey, and had an opportunity to interview the personnel there. The laboratory was in the middle of the relocation process, and therefore it was not possible to observe the working equipment which was all dismantled at the time of the inspection.

The Qualified Person has reviewed the drill logs, cross sections, plan maps for the Dasa geological database.

The Qualified Person checked the analytical and geological database using macros and processes designed to detect the following errors as described in Section 14.3. Data Validation.

All work relating to geological exploration and leach testing was found to be of a high quality. The data is considered suitable for Mineral Resource estimation.

Dmitry Pertel considers his 2017 site visit to be current under Section 6.2 of NI 43-101 and no additional site visit was deemed necessary. Another visit was not warranted as the same procedures were in place and there was no material changes or new areas under investigation.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1. Overview

In 2020 Global Atomic Corporation initiated a Feasibility Study test work program for the Dasa Uranium Project located in Niger. The objectives of the test work were to develop a process for the recovery of uranium from the high-grade Flank Zone of the Dasa deposit and to provide detailed information for the processing plant design. This work followed on from the Preliminary Economic Assessment (PEA) completed in 2020 on this orebody.

Between 2011 and 2018, various metallurgical test work programs were completed investigating potential uranium recovery methods from the near surface resource of this deposit. The historical test work programs investigated opportunities for scrubbing, heap, and tank leach recovery routes as reported in the previous PEA and technical reports. However, the limited overall uranium recoveries of 85% were seen as suboptimal as other mines in the area have reported achieving higher recoveries using a different process of pugging and curing.

Processing test work during the current phase of the Dasa Project focused on a deeper higher-grade resource of the deposit known as the Flank Zone whose mineralisation is similar to other operations (Somair and Cominak) in Niger. During 2019, the laboratory at Somair in Niger and Insight R&D in Toronto, Canada, conducted test programs to provide information for preliminary flowsheet development based on pugging and curing principles as applied at the Somair and Cominak operations. The objective of that test work was to obtain metallurgical response of the mineralised material from the Flank Zone using pugging and curing process route. The recent feasibility test work was conducted to provide design information for the Flank Zone.

Pugging and curing bench scale test work achieved improved uranium extraction efficiencies from the tank and heap leach test work of 85% to between 89% and 97% for the new pugging and curing process. Thus, the pugging and curing process, which adds a strong acid to the “dry” mill feed, formed the basis of the Dasa Project’s PEA process flowsheet design (CSA Global, 2020).

Following a favourable response of the Flank Zone ores to pugging and curing, further flowsheet development test work programs were designed to advance the Dasa Project to achieve Feasibility Study level design requirements focused on in this report.

The Feasibility Study test work was conducted during 2020 and 2021 and its interpretation for flowsheet design is discussed in this section. Various optimisation tests were conducted during the feasibility study and implemented into the final flowsheet design.

The feasibility study update has not had any additional significant test work completed. As such this section remains unaltered from the 2023 feasibility details.

13.2. Previous Metallurgical Test work (2011 – 2018)

Previous quantitative mineralogy and hydrometallurgical testing programs were conducted by SGS and Mintek for the Dasa Project. Relevant technical data generated as part of previous programs has been reported in the previous PEA (CSA Global, 2018).

Characterization Studies (2011 – 2018)

Characterization tests performed at SGS on Dasa 1 and Dasa 3 ores included mineralogy, comminution, leach tests, precipitation and uranium recovery using conventional recovery processes. The key factors from these tests are as follows:

1. The head grade was approximately 600 ppm U_3O_8.
2. Bottle roll leaches on 600 ppm U_3O_8 feed samples gave uranium extraction of 78% to 86% for tests run at 20 g/L free acid.
3. Carbonate leaching achieved 68% uranium extraction.
4. Comminution tests indicated that the mineralized material was suitable for semi-autogenous grinding as noted in the grindability parameters in Table 13-1: Summary of selected comminution test results.

Table 13-1: Summary of selected comminution test results

Parameter	Unit	Value
CWI	kWh/t	11.5
BWI	kWh/t	16.1
Ai	g	0.096
JK Parameters A × b		118
JK Parameters Ta		1.32
Relative Density		2.31

Near Surface Low-Grade Samples Test Work (For Heap Leaching)

Low-grade samples were subjected to mineralogical analysis, leach tests including scrubbing, bottle roll and column leach test work under both acid and alkaline conditions at Mintek. The head analysis of the samples was approximately 250 ppm U_3O_8. The following results were achieved:

- Bottle roll leach tests showed that acid-in-agglomeration and curing had a positive effect on kinetics and overall uranium dissolution.
- Agitated leach tests at 250 ppm U3O8 achieved uranium recoveries around 93% at a grind P80 170 microns after 24 hours and 530 kg/t sulphuric acid addition.
- Alkaline leach tests achieved uranium recoveries around 62% after 73 days of leaching.
- Stacking tests and hydraulic conductivity test work showed that stacking heights of the leach material of 5–7 m was achievable.
- The samples were not amenable to scrubbing.

Agitated Leach Testing

Medium-grade mineralization samples from the Dasa 3 area were provided to SGS for additional leach tests including comminution test work to confirm data from previous studies. The head analysis of these samples ranged between 1,200 ppm and 1,900 ppm U_3O_8.

Historical Test work Outcomes

The key outcomes which can be drawn from historical test work are:

- Limited uranium recovery at 85%.
- High sulphuric acid consumption of up to 530 kg/t ore.
- The lower recovery and high acid consumption pushed the test work towards alternative recovery processes like pugging and curing.

13.3. PEA and Feasibility Study Sample Selection (2019 – 2020)

A new test work program focussed on the Flank Zone of the Dasa orebody was initiated in 2019 as part of the PEA (CSA Global, 2020) which was further advanced into the current Feasibility Study.

Sample Selection

Samples for test work were selected by site geologists to be representative of the Flank Zone resource with respect to ore type, and locations. These samples were then prepared and shipped to test facilities. Each sample bag was labelled with a client sample identification number, drill hole number and interval depth.

The sample selection was initially focussed on suitable PEA samples and was later extended to cover Feasibility Study samples upon the successful application of the pugging and curing process on the Flank Zone ores.

The facilities where the metallurgical test work programs were completed included:

- Somair Laboratory, Niger - PEA test work.
- Insight R&D, Canada – PEA and Feasibility Study test work.
- SGS Canada – Feasibility Study comminution test work.

The drill holes sampled provided a good geographical coverage of the defined Flank Zone ore resource. The samples covered a range of mineralisation styles for both the PEA and Feasibility test work programs.

Table 13-2 and Table 13-3 summarise the test samples selected and shipped to the test facilities for the PEA and Feasibility studies.

Table 13-2: PEA Study Flank Zone Drill Hole Samples.

PEA – Somair Laboratory		**PEA – Insight R&D**	
Drill Hole – Flank Zone	Grade, U ppm	Drill Hole - ASDH538	Grade, U ppm
Sample 21344	1 068	Sample 15975	2 414
Sample 21345	3 106	Sample 16098	6 953
Sample 21346	3 081	Sample 15978	6 486
Sample 21347	22 348		
Sample 21348	35 210		
Sample 21349	2 420		

The preliminary pugging and curing test work for the Dasa Project were completed by the Somair laboratory in Niger. Further pugging and curing tests and the rest of the Dasa flowsheet process development was completed in Canada by Insight R&D.

Table 13-3: Feasibility Study Drill Hole Samples.

Borehole	**Average Grade, U_3O_8 ppm (by XRF)**	**Mass, kg**
ASDH538	6 263	52.3
ASDH541	6 957	15.0
ASDH543	5 848	79.7
ASDH552	6 115	5.2
ASDH563	6 677	98.4
ASDH565	8 962	1.5
ASDH570	5 690	3.1
TOTAL		**301**

Table 13-4 summarises the lithologies covered by the various samples provided for the Feasibility study and indicates that most of the ore (75.1%) consists of a medium to very coarse-grained sandstone.

Table 13-4: Lithology Summary of Flank Zone Samples for Pilot Plant Test Work.

Type	Lithology	Mass (kg)	%
1	Very Coarse Grain Sandstone	119.3	39.6
2	Coarse Grain Sandstone	65.4	21.7
3	Medium Grain Sandstone	41.6	13.8
4	Analcimolitic Sandstone	27.4	9.1
5	Organic Matter Sandstone	16.8	5.6
6	Analcimolite	12.2	4.0
7	Microcinglomerate	7.2	2.4
8	Fine Sandstone	6.3	2.1
9	Siltstone	5.3	1.7
10	Arkosic Sandstone	0.0	0.0
11	**TOTAL**	**301.1**	**100**

13.4. PEA and Bench Scale Test Work Summary (2019)

The PEA and bench scale test work review was based on the following reports:

1. Insight R&D, Dasa Leach and SX Test work, Project Number 1910-01-R0, February 3, 2020.
2. Insight R&D, Modernizing the Cominak Process for Dasa Ore, Draft Report Number 2003-01-R1, August 24, 2020.
3. Insight R&D, Optimising Oxidant Concentration for Dasa Ore, Project Report Number 2006-02-R1, September 3, 2020.
4. Insight R&D, Somair Laboratory Leach Test Work Results spreadsheets.

Pug Leaching and Curing Process Tests

Due to suboptimal recoveries and high acid consumption from the previous tank and heap leaching flowsheet approaches, an alternative process was required. Cominak and Somair are operations in Niger that have historically processed uranium-bearing sandstone deposits similar to the Dasa Project. A review of published literature and limited discussions with operational teams provided insights into the optimal performances achieved at these existing operations. The Cominak plant was designed to process feed head grades averaging 4,500 ppm U_3O_8 and the operations team reported that they consistently achieved uranium recoveries above 90%. This provided motivation to pursue the pugging and curing process for the Dasa Project and to define a framework for:

- Process confirmation.
- Process development.

The PEA test work was specifically focused on duplicating the fundamentals of the pugging and curing process as applied at the Somair and Cominak process plants.

The objective of the PEA test work was to obtain the metallurgical response of mineralised material from the high-grade Flank Zone of the Dasa deposit to pugging and curing and to provide preliminary design information for the PEA Study. This involved head assay, pug leach and curing, oxidation reagents screening, solvent extraction, and product precipitation reagents screening, and drying. Optimum conditions were then used in locked cycle pilot plant tests to approximate plant metallurgical results.

Somair Laboratory Test Work

Preliminary pug leaching and curing tests on six samples from the Flank Zone ore resource were tested at the Somair laboratory in 2019. The test results achieved are summarised in Table 13-5. Three samples achieved approximately 97% uranium extraction into solution, while the other 3 samples had high-clay (~21%) and achieved lower recoveries ranging from 57% to 81%. The clay content differences in these test samples were due to the drillhole location, which was adjacent to the footwall contact and fault zone, whose occasional tuffs had been altered to clays. This level of alteration is not interpreted within the main Flank Zone and recoveries above 90% are anticipated.

Table 13-5: Somair Laboratory Summary of Uranium Extraction Results.

Sample	Unit	21344	21345	21346	21347	21348	21349
Head grade	ppm	1 068	3 106	3 081	22 348	35 210	2 420
Clay Content	%	<2	<2	23.4	21.6	18.8	6.7
Acid	kg/t	50	50	80	80	80	100
Oxidant	kg/t	2	2	2	2	2	2
L/S	l/kg	0.12	0.12	0.16	0.18	0.16	0.12
Residue grade	ppm	35	99	1 302	9 608	6 696	75
Extraction efficiency	%	96.7	96.8	57.7	57.0	80.9	96.9

Insight R&D Test Work

New samples of the Flank Zone from borehole ASDH538 (samples 15975, 16098 and 15978) were provided to Insight R&D in December 2019 for more detailed test work following pugging and curing principles utilising a similar "recipe" as used in Orano's Cominak plant flowsheet. The work was conducted at Process Research Ortech Inc. facilities at Mississauga, Canada and covered the following:

- Pug leaching and curing.
- Solvent extraction.
- Stripping.
- Precipitation and drying.

High-Grade Sample Preparation

Samples from borehole ASDH 538 (samples 15975, 16098 and 15978) were prepared by crushing to less than 0.8 mm top size in a hammer mill. The particle size distribution for sample 15975 and sample 16098 was 69% -210 µm while for sample 15978 was 57% -210 µm.

For the pug leach test work, the samples were mixed with 80 kg/t sulphuric acid, 10 kg/t nitric acid, 2.5 kg/t sodium nitrate and water to achieve 16% moisture. The mixing was carried out for 10 minutes following the Cominak plant operating procedures. The mixture was then cured for 180 minutes at room temperature. The pugged solids were re-pulped to 50% solids at 60 °C for one hour. The solids were then filtered and washed with slightly acidified water to recover entrained solution. The solid residues, pregnant leach solution (PLS) and wash solution streams were then assayed to enable calculation of a material balance.

High-Grade Sample Test Results

The average uranium leach and wash recovery for the three samples was 95%. The results are summarized in Table 13-6 below.

Table 13-6: Insight R&D Summary of Uranium Extraction Results.

Low-Grade Sample Test and Results.

Test	Samples Class	U in Feed (ppm)	U in Residue (ppm)	U Extraction (%)
1	Low Grade	343	111	68
2	Low Grade	243	73	70
3	Low Grade	624	228	65
4	Low Grade	210	71	67
5	Medium Grade	2 414	60	98
6	High Grade	6 953	170	98
7	High Grade	6 486	775	89

In addition to the high-grade samples tested, additional lower grade samples were tested. These originate from outside of the Flank Zone which is the focus of this study. These low-grade ore tests evaluated the impact of different particle size distributions, variation of oxidation reagents, and influence of feed grade upgrading. Finer grind did not provide any recovery improvement, and neither did the variation in oxidant type. The average head grade and recovery of the four samples was 355 ppm U_3O_8 and 68% respectively.

Solvent Extraction

Insight R&D conducted test work to interpret the solvent extraction characteristics of the PLS generated from the high-grade leach tests. Equilibrium loading isotherms were generated utilising a primary amine solvent to determine its loading capacity and the likely number of equilibrium stages required to extract the uranium from the aqueous PLS into the organic solvent. The solvent extraction organic consisted of 8.7% amine 336 extractant and 10% tridecanol modifier in a kerosene solvent at a pH between 1.1 and 2.1.

The results from the uranium recovery may be summarized as follows:

- Extraction was achieved using an aqueous to organic ratio of 1.54 and four extraction stages.
- Stripping was achieved with 1.5 M NaCl aqueous solution at an aqueous to organic ratio of 1:7 producing a strip solution containing 26,887 ppm U.
- The organic composition which provided the most favourable conditions for extraction were 8.7% amine 336 extractant and 10% tridecanol modifier in kerosene.

Precipitation and Drying

Insight R&D conducted precipitation tests on the pregnant strip solution (OK liquor) from solvent extraction. They duplicated the precipitant used by Cominak namely milk of magnesium solution to produce a slurry of magnesium di-uranate, which was filtered and air dried to successfully produce a cake with a uranium concentration of 52.1 wt%, equivalent to 61% U_3O_8.

Bench Scale Test Work Outcomes

Bench scale testing led to the following optimised parameters for pugging, curing, and repulping:

- Grind to P_{80} ~ 600 microns in a dry mill.
- Pug drum mixing time: 10 minutes.
- Curing time: 180 minutes.
- Sulphuric acid addition: 80 kg/t.
- Nitric acid addition: 10 kg/t.
- Sodium nitrate addition: 2.5 kg/t.
- Water addition: 103 kg/t.
- Target moisture content: 15% m/m.
- Repulping time: 60 minutes.
- Repulp slurry density: 50% m/m.
- Uranium solution concentration: 6.9 g/L.

On successful testing at bench scale, a pilot plant scale flowsheet evaluation using the selected parameters was conducted.

13.5. Feasibility Study Test Work (2020 – 2021)

The feasibility study test work review was based on the following reports:

- Insight R&D, Dasa Pilot Plant Campaign 1, Project Report Number 2006-03-1-R0, December 7, 2020.
- Insight R&D, Dasa Pilot Plant Campaign 2, Project Report Number 2006-03-2-R0, January 15, 2021.
- Insight R&D, Dasa Pilot Plant Campaign 3, Project Report Number 2006-03-3-R0, February 4, 2021.
- SGS Minerals Services, SMC Test Report for Global Atomic Corporation, JKTech Job No: 21007/P1, December 2020, SGS, Ontario, Canada.
- Insight R&D, Tailings Density Report Jan 20, 2021, Spreadsheet Report.
- Insight R&D, SDU Filtration Test Results, June 29, 2021, Spreadsheet Report.
- Insight R&D, Yellow Cake Settling Density Sieve Composite Feb 28 21 Data Report.

The Feasibility Study test work was motivated based on the demonstrated PEA bench scale process viability. The Feasibility Study test work programs focussed on the following:

- Variability testing to demonstrate the processing characteristics of different ore zones.
- Testing of process circuits to obtain engineering design data for equipment sizing and selection.
- Mass balancing data to confirm reagent consumptions and metal recoveries.
- Solid liquid separation test work for sizing and selection of process equipment.
- Particle size data for pumping and agitation specifications.
- Neutralisation of raffinate solution.

Feasibility Study Sample Composites

The variability samples were representative of the first 5-years of Flank Zone mining. The ore samples were sorted into 3 pilot plant campaigns, that represented the first, second and third, 20-month periods of the first 5-years of mining. Table 13-7 summarises the lithologies of each Campaign as per the sub-division. The sample mass for each Campaign was approximately 80 kg.

Table 13-7: Lithology Summary of Pilot Plant Campaign Test Work Samples.

Type	Lithology	Campaign 1 (%)	Campaign 2 (%)	Campaign 3 (%)
1	Very Coarse Grain Sandstone	21.9	54.0	0.0
2	Coarse Grain Sandstone	25.0	18.1	2.3
3	Medium Grain Sandstone	4.0	14.0	5.7
4	Analcimolitic Sandstone	25.9	5.8	6.7
5	Organic Matter Sandstone	14.4	0.0	0.0
6	Analcimolite	8.8	4.0	4.4
7	Microcinglomerate	0.0	0.0	0.0
8	Fine Sandstone	0.0	0.0	0.0
9	Siltstone	0.0	4.1	21.1

Type	Lithology	Campaign 1 (%)	Campaign 2 (%)	Campaign 3 (%)
10	Arkosic Sandstone	0.0	0.0	56.7
	Total	**100**	**100**	**100**

The Campaign 1 composite sample was approximately 51% of medium to very coarse-grained sandstone with a higher content of analcimolitic and organic matter sandstones.

The Campaign 2 composite sample was approximately 86.0% of medium to very coarse-grained sandstone.

The Campaign 3 composite sample was approximately 81% of medium to very coarse-grained sandstone.

A composite sample (~ 20 kg) was created from the bulk 301 kg sample for comminution test work.

Comminution Characterisation Results

SGS Canada completed the grindability characterisation of composite samples from the Flank Zone of the Dasa ore body. The test work included:

- SMC Test.
- Bond Ball Mill Work Index.
- Abrasion Index.

The test work results are summarised Table 13-8. The complete grindability characterisation test work results are provided in SGS Project 18226-01 Final Report.

SMC Test Results

The SMC test was performed on Composite 1 and Composite 2. The test results are summarized in Table 13-8 and detailed in the JKTech report, along with the test procedure, calibration, and test details.

Table 13-8: Summary of SMC Test Results.

Parameter	Unit	Composite 1	Composite 2
A		85.0	88.8
b		2.40	2.09
A × b		204	186
DWi	kWh/m^3	1.14	1.26
Mia	kWh/t	5.5	5.9
Mih	kWh/t	2.8	3.1
Mic	kWh/t	1.5	1.6

Parameter	Unit	Composite 1	Composite 2
SCSE	kWh/t	5.85	5.95
Relative Density		2.33	2.35

Both Composite 1 and Composite 2 were categorized as very soft in terms of resistance to impact breakage, with A × b values of 204 and 186, respectively. The average relative densities varied from 2.33 to 2.35.

It should be noted that the milling required for liberation of uranium is of lower intensity and limited to the removal of uranium bearing compounds that coat the quartz particles. The milling process should minimize fractures or breakage of quartz grains as these increases undesirable leach reactions and quartz-based acid consumption that results in more silica in solution which can cause unwanted crud formation in the SX plant.

Bond Ball Mill Grindability Test Results

Composite 1 and Composite 2 samples were submitted for the Bond ball Work Index grindability test, and the results are summarized in Table 13-9.

With BWI values of 12.6 kWh/t for Composite 1 and 12.7 kWh/t for Composite 2, both samples were categorized as moderately soft. The attained P_{80} values were 357 and 358 microns.

Table 13-9: Summary of Bond Ball Mill Grindability Test Results.

Parameter	Unit	Composite 1	Composite 2
Closing Screen	µm	425	425
Feed, F_{80}	µm	2244	2075
Product, P_{80}	µm	358	357
Gram per Mill Revolution	g	3.89	3.89
Ball Bond Work Index	kWh/t	12.6	12.7
Hardness Percentile		31	32

Bond Abrasion Test Results

Composite 1 and Composite 2 were also submitted for the Bond abrasion test. The test results are summarized in Table 13-10.

With an Ai value of 0.210 grams, Composite 1 was categorized as slightly abrasive while an Ai value of 0.256 grams means that Composite 2 falls in the medium range of abrasivity of the SGS database.

Table 13-10: Summary of Bond Abrasion Test Results.

Parameter	Unit	Composite 1	Composite 2
Ai	g	0.210	0.256
Percentile Abrasivity		37	45

13.6. Pilot Plant Flowsheet

The pilot plant flowsheet is represented in Figure 13-1. The front end (leach section) of the pilot plant flowsheet was identical for all the Campaigns 1, 2, and 3. The overall flowsheet was operated in an integrated manner. The pilot plant campaign was long enough to enable the effects of recycling liquors like the raffinate solution for the team to evaluate SX recovery and subsequent uranium recovery process.

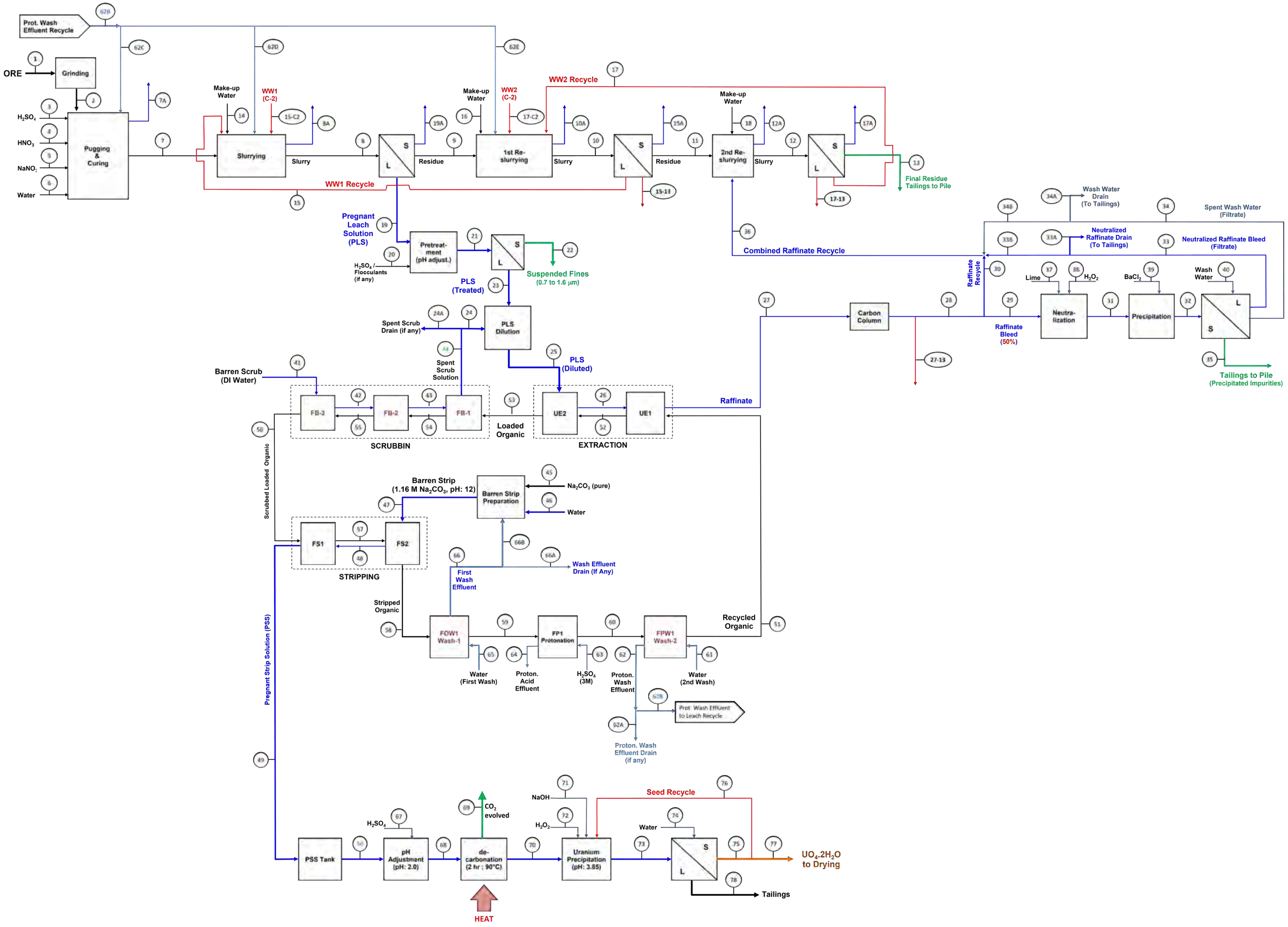


Figure 13-1: Dasa Pilot Plant Flowsheet

Any improvements identified in a Campaign were carried on to the next Campaign which enabled continuous improvement of the flowsheet and resulted in an optimized final flowsheet that was used for the parallel development of the plant design.

13.7. Pilot Plant Metallurgical Test Work Results

Sample Composition

The composition of the samples used in the pilot plant Campaign programs are shown in Table 13-11. Samples from milled and homogenized material were taken and analysed by Inductively Coupled Plasma Optical Emissions Spectroscopy (ICP-OES) to produce data on the ore composition.

Table 13-11: Average Head Grade of Campaign Composites (wt.%).

Element	Campaign 1	Campaign 2	Campaign 3
U	0.427	0.426	0.374
Al	4.635	4.558	3.218
Ca	0.164	0.134	0.132
Cr	0.015	0.0188	
Cu	0.088	0.044	0.036
Fe	1.756	1.411	0.921
K	1.498	1.526	1.498
Mg	0.480	0.406	0.168
Mn	0.023	0.033	0.014
Na	1.039	2.109	1.329
Ti	0.140	0.114	0.095
V	0.012	0.012	0.030
Zn	0.009	0.006	0.008

The average uranium grades for Campaigns 1 and 2 were identical whilst Campaign 3 was slightly lower than the first two Campaigns.

Particle Size Distribution

The ore samples received had been previously crushed to less than 10 mm top size. The sorted samples were ground in a controlled manner to achieve 92% passing 600 µm (known Somair particle size distribution). Figure 13-2 summarises the particle size distribution of the milled ores for Campaign 1, 2 and 3 and compares them to the known Somair particle size distribution.

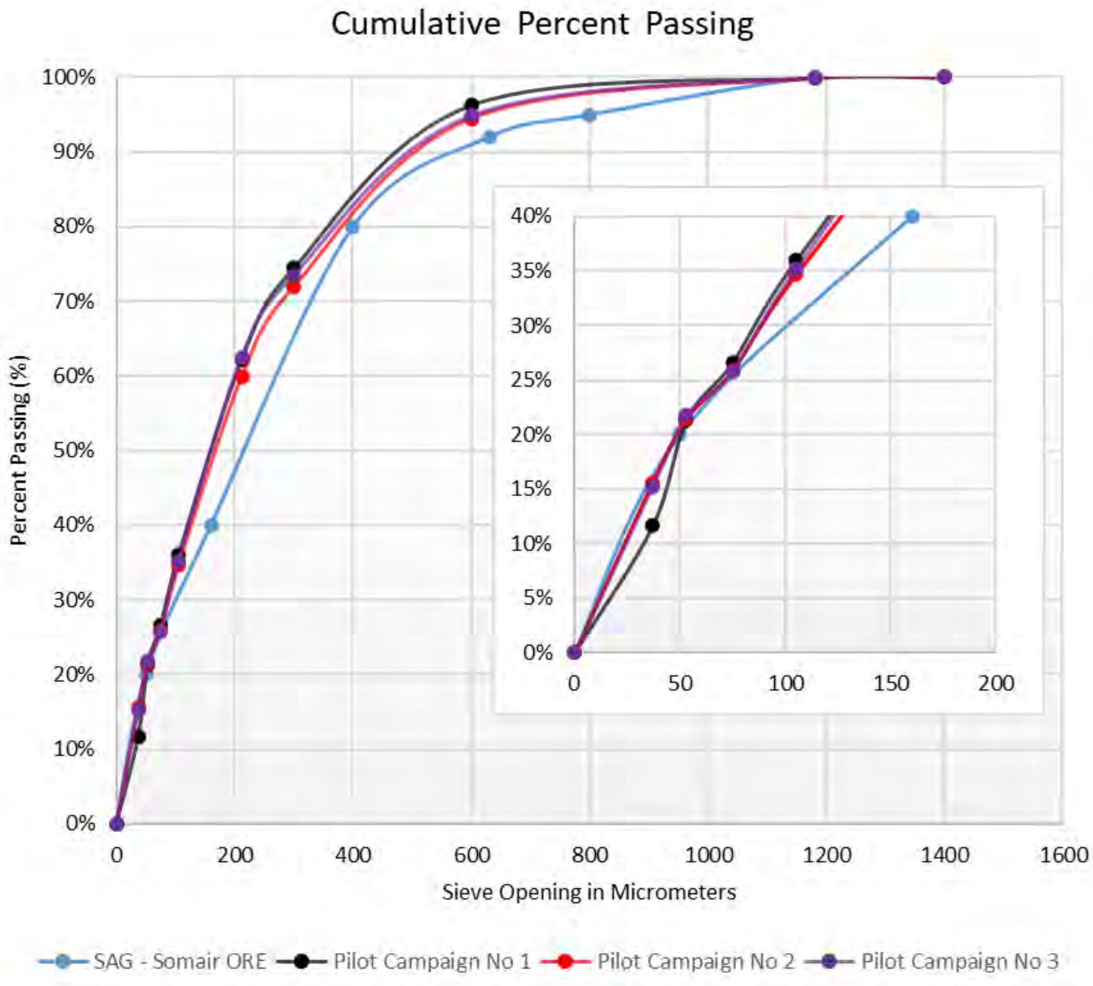


Figure 13-2: Campaign 1, 2 and 3 Particle Size Distributions vs. Somair Reference.

As can be seen from Figure 13-2, the ores from the first 60 months of mining in the Flank Zone responded similarly to the milling process.

Leach Tests

Ore samples from the Dasa Flank Zone's first 20 months of mining were processed through the first pilot plant as Campaign 1, followed by Campaign 2 (21 – 40 months) and Campaign 3 (41 – 60 months). The tests were conducted using the following reagents consumptions as determined by the PEA bench scale test work: 80 kg/t sulphuric acid, 10 kg/t nitric acid, 2.5 kg/t sodium nitrate, 103 kg/t deionised water.

Pugging was completed in 10 minutes followed by curing for 180 minutes. At the end of the curing period, the leach solids were repulped to 50% m/m solids and agitated for 60 minutes.

Table 13-12: Pregnant Leach Solution Average Grade Post Filtration.

Test Campaign	U	Na	Al	Fe	K	Mg
Campaign 1 Solution (mg/L)	6 325	9 100	12 630	11 420	259	4 813
Campaign 2 Solution (mg/L)	7 608	13 180	21 030	15 270	299	6 373
Campaign 3 Solution (mg/L)	6 753	15 019	19 628	9 831	169	2 529

Table 13-13 summarises the average extraction of uranium into pregnant leach solution. The uranium remaining in the leach residues includes refractory uranium which could not be leached as well as solution losses resulting from leach solids wash inefficiencies.

Table 13-13: : Recovery of Uranium from Ore into Pregnant Leach Solution.

Test Campaign	Units (%)	Recovery (%)
Pilot Plant Campaign 1	%	92.8
Pilot Plant Campaign 2	%	94.6
Pilot Plant Campaign 3	%	97.8

The average uranium recovery to pregnant solution was estimated at 95.1%.

Because of lower impurities in the ore sample for Campaign 3, the team investigated the possibility of lowering acid addition to reduce acid consumption and thus reduce the operating costs. However, in the evaluation of data, acid consumption was found to be directly proportional to recovery (decreasing acid addition lowered recovery) and thus acid addition was maintained at 80 kg/t for Campaign 3.

Uranium Solvent Extraction Circuit

Uranium SX used an organic extractant dissolved in an organic diluent carrier to selectively extract uranium from the uranium-rich solution, while leaving impurities behind. The organic solution used in the tests was calculated to require 8.65% by volume Alamine 336 extractant in 84.35% by volume diluent and 7.00% Exxal 13 modifier.

Table 13-14: Summary of SX Operating Conditions.

Pilot Plant	Campaign 1		Campaign 2		Campaign 3	
SX Stages/O:A Ratio	Stages	O:A	Stage	O:A	Stages	O:A
Extraction	2	2.2	2	2.3	2	2.1
Scrubbing	2	6.0	3	9.6	2	8.1
Stripping	6	6.4	2	6.7	2	5.1
Regeneration	1	4.0	n/a		n/a	
Pre-protonation Wash	1	5.3	1	9.3	1	8.3
Protonation	1	8.7	1	5.8	1	5.2
Post-Protonation Wash	1	5.0	1	8.5	1	7.2

The loaded organic from the extraction tests were scrubbed using de-ionised water to remove impurities (principally iron, alumina, silica) that were extracted to the organic phase with minimal loss of uranium.

Campaign 1 used a 1.5 M sodium chloride solution for stripping at a pH of 1.5 to 2.2. However, sodium chloride could not strip all the uranium from the loaded organic despite the use of up to six strip stages. Sodium carbonate was tested and proved to be a more effective stripping agent.

In Campaigns 2 and 3, 1.16 M sodium carbonate solution required only two stages to completely strip the loaded organic phase.

Table 13-15 summarises the efficacy of the stripping reagents used during the pilot plant Campaigns.

Table 13-15: Stripping Agent Efficiencies.

Campaign	Strip Agent	Loaded Organic (mg/L U)	Stripped Organic (mg/L U)	Stripping Efficiency (%)
Campaign 1	1.50 M NaCl	2 651	246	90
Campaign 2	1.16 M Na_2CO_3	2 537	0	100
Campaign 3	1.16 M Na_2CO_3	2 779	1	100

Raffinate Neutralisation

The raffinate solution leaving the extraction circuit was recycled and used for leach residue washing. It had been observed during bench scale test work that the density of the raffinate solution increased up to 1.3 g/ml through repeated recycling which had a negative impact on the associated filtration process. Thus, a portion of the raffinate (up to 52%) was bled through a neutralization circuit where it was treated with lime (a 20 wt% $Ca(OH)_2$) solution to precipitate unwanted metals before being recombined with the balance of the raffinate.

Uranyl Peroxide Production

The pregnant strip solutions obtained from the SX were combined and used to conduct bench scale uranyl peroxide precipitation tests. The precipitation process developed utilised the following recipe:

- De-carbonation of the feed solution by reducing pH to ~2.0 with the addition of H_2SO_4.
- Boiling of the solution at 90 °C to produce a uranyl sulphate solution.
- Caustic addition to adjust the pH of the feed solution between 3.7 to 4.8.
- Addition of 200% excess of the stoichiometric amount of hydrogen peroxide.

This process was successfully demonstrated but had the following shortcomings:

- Extreme operating conditions.
- High operating costs due to high reagent consumptions i.e., sulphuric acid and sodium carbonate.
- Operated in open circuit creating a water balance issue for the process.

Additional test work was conducted to assess optimised design parameters and to reduce future operating costs. This is described in section 13.6 below.

UP Thickening Test Work

Static settling tests were performed at room temperature in 2 litre graduated cylinders. The test results are summarised in Table 13-16. No flocculant was used during the test work.

Table 13-16: Summary of Uranyl Peroxide Free Settling Tests.

Description	Campaign 1	Campaign 3
Test Type	Free Settling	Free Settling
Average Feed Solids, % m/m	8.52%	4.50%
Free Settling Rates, m/h	0.76	1.04
Flux, t/m^2/h	1.66	1.16
Underflow Density, % m/m	43.08	30.6

Reasonable thickener fluxes were achieved.

Ore Variability

The pilot plant test work indicated that the ore's leachable impurities decreased in Campaign 3 (months 41 to 60) and therefore an opportunity to use less acid could be achieved in later years. However, subsequent test work indicated that by maintaining 80 kg/t H_2SO_4, the uranium recovery could be further enhanced.

Pilot Plant Campaign Conclusions

The pilot plant Campaigns demonstrated the flowsheet to be robust for the high-grade Flank Zone ores of the Dasa deposit. However, several areas were identified for optimisation as follows:

- Slow filtration of leach residue solids which was observed in all the test work Campaigns.
- High acid consumption, and severe process conditions required for the destruction of the intermediate uranyl carbonate product.
- Raffinate solution density which increases with recycling – observed from test work Campaigns.
- Water balance issues resulting from the open circuit stripping.

13.8. Flowsheet Optimisation Test Work

The flowsheet optimisation test work review was based on the following reports:

- Insight R&D, Dasa Pilot Plant 1st Filtration Tests, Project Report Number 2010-01-1-R0, April 8, 2021.
- Insight R&D, Dasa Leach Filtration 2nd Test Campaign, Project Report Number 2103-01-1-R0, July 8, 2021.
- Insight R&D, Yellow Cake Production Route, Project Report Number 2103-01-2-R0, July 28, 2021.

Leach Residue Filtration Test Work

Following slow filtration rates during pilot plant Campaigns, optimisation test work focussed on the application of coagulation and flocculation to enhance filtration performance.

The following outcomes were identified from bench scale test work:

- Sequential addition of a cationic coagulant and flocculant enhanced filtration performance.
- Dosage rates of 25 g/t and 150 g/t for coagulant and flocculant respectively achieved benchmark rates.
- Correct on-belt washing configuration achieved effective filtration fluxes of 0.64 to 0.88 t/h/m^2.

The filtration optimisation test work results are summarised in the Insight R&D, Dasa Leach Filtration 2nd Test Campaign Project Report No. 2103-01-1-R0.

Sodium Di-uranate Precipitation Tests

Though the carbonate destruction route for the production was successfully demonstrated, some improvements were required due to high acid consumption, the open circuit nature of the precipitation flowsheet which created water balance issues, and extreme temperature and acidic operating conditions required for the process. To resolve these issues, the precipitation of an intermediate Sodium Di-Uranate product (SDU) from the carbonate based OK liquor was thought to offer some improvements. This SDU approach was demonstrated by test work to be viable and produced a high-quality UP yellowcake that met ASTM C967-20 sodium content standards. This process route solved the water balance issue, and significantly reduced reagent consumption costs.

However, settling and filtration rates of the intermediate SDU precipitate were found to be low as summarised in Table 13-17.

Table 13-17: Sodium Di-uranate Thickener and Filter Fluxes.

Description	Units	Free Settling Test	Filtration Test
Feed Solids	% m/m	3.1%	3.1%
Flux	t/m^2/d	0.015	0.084

This was attributed to very small SDU particle sizes which do not settle easily and offer a high resistance to filtration. New test work has been completed using fluid bed reactor technology by insight R&D. They developed a precipitation reactor and separation process which allows larger product particles which exhibit a significantly better filtration rate.

This is summarised in Table 13-18 showing previous and revised filtration rates.

Table 13-18: Filtration Rates.

Filtration Rate (tonnes/m^2/h)				
Description	**Original Work**	**New Test Results 1**	**New Test Results 2**	**Equipment Specification**
SDU Yellowcake	0.003	0.3 to 0.5	0.05 to 0.2	0.1
UP Yellowcake	0.16	0.3 to 0.6	NA	0.1

This test work has derisked the applicable section of the SDU and UP yellowcake allowing smaller equipment to be specified.

Raffinate Neutralisation

Pilot plant test work indicated high dissolution of various impurity metals resulting in an elevated raffinate solution density. This high-density characteristic of the raffinate reduces its washing effectiveness on the belt filters and therefore treatment of the raffinate solution to reduce density is required. Neutralisation of the solution to precipitate dissolved metals used extensive amounts of reagents that will result in high operating costs in future operations. Furthermore, the water recovery from the gypsum-based solids was poor which along with water of hydration results in the neutralised precipitated solids having a significant volume in the disposal facility. Therefore, an alternative flowsheet in which a bleed stream of raffinate solution without neutralisation is pumped to an evaporation pond to reduce overall volumes of process solution was incorporated into the flowsheet design. The raffinate bled to an evaporation pond will be replaced with raw water in the plant design.

13.9. Summary

Core samples from the various ore lithologies and mineralisation styles were selected to represent the expected LoM average grades.

The comminution test work programme produced characteristic data for the Flank Zone ore lithology. The uranium minerals exist in the cement holding the silica particles together within a sandstone matrix. Thus, a target grind of P_{95} 600 µm was found to be sufficient for the liberation of the quartz particles without overgrinding. A grind size of P_{95} 600 µm was selected for design.

A detailed pilot plant campaign represented mine production for the first 5-years of operation and provided a view with regards to ore variability. The pilot plant indicated that that there were fewer impurity metals being dissolved with increasing ore depth, which presented an opportunity to lower acid addition. However, subsequent test work indicated that better recovery could be achieved by retaining the acid addition at 80 kg/t.

The SDU intermediate process route was developed to replace the uranyl carbonate destruction process route developed during the pilot plant trials. The SDU precipitation conditions were accepted as improving the flowsheet robustness and the base design conditions were modified to suit, resulting in conservative operating cost inputs due to reagents cost savings.

Leach residue filtration was improved by the sequential application of cationic coagulants (25 g/t) and flocculants (150 g/t).

13.10. Conclusions

The validity of the Dasa Project flowsheet and the projected uranium recovery were confirmed by the pilot plant test work and is in line with the process as utilised at Cominak for treating similar ore. The conclusions from the pilot plant test work were:

- The flowsheet developed was successfully applied on all the ores evaluated on the pilot plant test work program.
- The number of impurities in the ore reduce with depth, thus becomes easier to process.
- Sodium carbonate was identified as more effective stripping agent than sodium chloride and was amenable to the production of an intermediate SDU product with lower operating cost and easier operation.
- A raffinate solution bleed was required to control the wash solution density and maintain its washing effectives on the belt filters.

13.11. Process Interpretation

The laboratory test work was based on the flowsheet from the Cominak mine operation at Arlit treating similar ore type, so the process is understood.

Test work has proven that simple crushing, dry SAG milling, screening to P_{95} 600 µm with heating of ore to ensure the moisture of feed to the screens is less than 0.5% m/m are effective measures.

Pugging and curing were shown to work well. The pilot plant test work was carried out using trays for curing after leach. The practical application of using conveyors on the plant for curing is well understood and utilised at the neighbouring Cominak operation. Repulping and mixing of the cured solids with belt filtration to recover PLS and washing of the leach tail solids is understood.

Three pilot plant campaigns were conducted in order to demonstrate the integrated operation of the major components of the process circuit and to determine the design requirements for specific items of process equipment.

Uranium Solvent Extraction

The feasibility study SX flowsheet is a simplified version of the pilot plant flowsheet as summarised in Table 13-19. The simplification of the test work SX circuit was motivated by comparisons with other uranium SX circuits and discussions with uranium SX experts including reagent suppliers, other operators, and information from existing operations.

Table 13-19: Summary of SX Test Work Setup Versus Feasibility Design.

SX Stage	Pilot Plant Campaign 2 & 3	Feasibility Study Process Design	Comment
Extraction	2	4	Two stages were sufficient for the pilot plant scale. 4 stages for feasibility design to allow for operational flexibility and variability management.
Scrubbing	3	3	
Stripping	2	3	To design the strip with pH control with concentrated Na_2CO_3 solution to avoid precipitation in the mixer settlers; also target high O/A ratio.
Pre-protonation Wash	1	0	Not seen as beneficial in feasibility design
Protonation	1	1	In line mixers used for protonation instead of mixer-settlers as protonation is a fast process. Require at least pH 1.5 to be effective and above pH 1.8 protonation drops.
Post-protonation Wash	1	0	Not seen as beneficial in feasibility design as returning acid back to extraction is not seen as a problem

Raffinate Neutralisation

Test work indicated that the build-up of metals in solution create filtration problems on the belt filter and loss of uranium. Neutralisation treatment of a raffinate bleed stream was found to be expensive in terms of operating cost and only achieved limited water recovery. Therefore, a bleed stream of raffinate solution to an evaporation pond was incorporated into the flowsheet design. The bled raffinate solution is then replaced with top-up raw water.

The solids precipitated during the evaporation process will be retained within the ponds, until the ponds are full.

Sodium Di-uranate Precipitation

The pilot plant test work proposed a uranium carbonate strip for the maximisation of uranium recovery from the organic solution. A sodium di-uranate (SDU) precipitation step has been incorporated in the feasibility design for the recovery of uranium from the carbonate solution. The SDU precipitation step is a proven process circuit currently in use at Somair. The SDU precipitation test work was demonstrated on Dasa ores and has solved the water balance issue and reduced reagent consumption rates.

The production of SDU has slow settlement and filtration rates due to fine particle sizes. Particle size can be enhanced with a 60 °C precipitation temperature along with multi-stage caustic addition.

This problem has been alleviated with the use of fluid bed reactor technology which now has a significantly improved filtration rate being achieved which derisks this section.

Metallurgical Recovery

The recoveries from pilot plant test work provided uranium recoveries as summarised in Table 13-19. The pilot plant operated in close to ideal operating conditions with detailed monitoring of all operational parameters. However, with the commercial plant operation inefficiencies and higher metal in solution losses will occur and need to be discounted from the pilot plant overall recovery.

Table 13-20: Dasa Pilot Plant Test Work Overall Uranium Recovery Summary at 80 kg/t H_2SO_4 and 2.5 kg/t $NaNO_3$.

Campaign	Units	Value
Pilot Plant Campaign 1	%	92.8
Pilot Plant Campaign 2	%	94.6
Pilot Plant Campaign 3	%	97.8
Overall Average Recovery	%	95.1
Feasibility Design Recovery	%	93.4

Ore to be processed later in mine operations has better recoveries as seen in Campaign 3 due to reduced dissolution of other metals. The recoveries presented in Table 13-20 were achieved after washing leached ore residue.

Although the pilot plant achieved negligible soluble losses, a factor has been allocated for this in the recovery analysis in line with expected plant operating conditions as additional losses are expected to be of a soluble nature.

The recommended plant recovery model allocates an overall recovery of 93.4% for the processed uranium. This is based on a determined 95.1% average uranium recovery identified in the recovery analysis report and 1.8% attributed to multiple soluble uranium losses.

Subsequent Processes

Cominak produced a Magnesium Di-uranate (MDU) product which is not ideal for best product price and not common in recent uranium process flowsheets. The Dasa plant will produce a superior uranyl peroxide product ($UO_4.2H_2O$) using hydrogen peroxide precipitation. This is a proven and more profitable product.

Feasibility Design Recovery	%	94.15

Ore to be processed later in mine operations has better recoveries as seen in Campaign 3 due to reduced dissolution of other metals. The recoveries presented in Table 13-19 were achieved after washing leached ore residue. Although the pilot plant achieved negligible soluble losses, a factor has been allocated for this in the recovery analysis in line with expected plant operating conditions as additional losses are expected to be of a soluble nature.

The recommended plant recovery model allocates an overall recovery of 94.15% for the processed uranium. This is based on a determined 95.1% average uranium recovery identified in the recovery analysis report and 0.95% attributed to multiple soluble uranium losses.

14. MINERAL RESOURCE ESTIMATES

Section 14 was previously reported in a NI 43-101 Technical Report with an effective date of 1 June 2019 (CSA Global, 2019). It has subsequently revised by AMC Consultants based on the 2021-2022 drill campaign and reported on in a press release dated May 23, 2023, with an effective date of May 12, 2023. The MRE has been prepared in accordance with CIM Definition Standards for Mineral Resources and Mineral Reserves (10 May 2014).

No new exploration work has been completed at the Project since the MRE's effective date of May 12, 2023.

14.1. Software Used

The Dasa uranium deposit Mineral Resources were updated by the Qualified Person and other AMC geologists under supervision of the Qualified Person, using Micromine version 2023 (23.0.381.4) software.

14.2. Database Compilation

GAC supplied AMC with the database including the new 28 drillholes that were drilled in 2021-2022 in CSV TEXT format. Results for previous exploration programmes were available in Micromine format.

The combined analytical database comprises estimated uranium equivalent grades (eU_3O_8) based on the gamma-logging of the drillholes. A separate file was supplied with all results of chemical assays resulting from XRF analysis.

The uranium oxide equivalent grades were calculated by the client's gamma probe operator from the LAS files (gamma-logging results). LAS files included CPS values, which were converted to uranium oxide grades using standard corrections and coefficients that account for the probe type (K-factor), casing steel thickness, presence of water and other factors.

The eU_3O_8 grades were calculated for each 10 cm interval using the LAS files. Some historical holes were not gamma-logged to the total depth, but which had results of the chemical assays. In those instances where gamma data did not exist and chemical assay data did exist, the chemical assays were used for interpretation and modelling. The available data is summarized in Table 14-1.

Table 14-1: Summary of Supplied and Used Data.

Category	Pre-2022 data			2022 data		Used for MRE
	Supplied	Gamma-Logged	Chemical assays	Gamma-logged	XRF	
Drill hole collars	1,028					
Drill holes used for MRE		1,017	93	28	24	1,046
Metres drilled	150,399	138,230	6,333	16,369		155,213
Survey records	11,622			1,709		13,331
Records in assay data file, including		1,138,299	9,784	158,920	4,773	
Assayed/probed intervals for U_3O_8		1,382,299	9,772	158,920		1,541,919 (incl. 700 chemical assays and 1,541,219 probed intervals)
Records in geology logging file	9,039			10,695		19,734

The databases consisted of several parts:

- Analytical database, including:
  - Drillhole collar coordinates.
  - Drillhole survey data.
  - Drillhole sampling database (results of the chemical assays).
  - Drillhole gamma-ray logging database (results with eU_3O_8 calculation).
  - Drillhole geological logging and codes.
- Topography data in the form of a DTM (supplied as a Micromine file).
- Bulk density data measurements for 3,594 core samples.

Import of the various updated datasets into Micromine software proceeded without errors.

The provided data for the MRE update also included the following files that were developed in 2019 for the previous MRE update. All these files were used for the current MRE update:

- Interpreted strings for mineralized bodies using 100 ppm eU_3O_8 cut-off.
- Wireframe models for all mineralized bodies.
- Wireframe models for full lithological model of the deposit.

14.3 Data Validation

The analytical and geological databases provided by GAC were checked using macros and processes designed to detect the following errors:

- Duplicate drillhole names.
- One or more drillhole collar coordinates missing in the collar file.

- FROM or TO missing or absent in the assay file.
- FROM > TO in the assay file.
- Sample intervals are not contiguous in the assay file (gaps exist between the assays).
- Sample intervals overlap in the assay file.
- First sample is not equal to 0 m in the assay file.
- First depth is not equal to 0 m in the survey file.
- Several downhole survey records exist for the same depth.
- Azimuth is not between 0° and 360° in the survey file.
- Dip is not between 0 and 90 degrees in the survey file.
- Azimuth or dip is missing in survey file.
- Total depth of the holes is less than the depth of the last sample.

It was found that 10 of GAC's earlier holes do not have analytical information or probe data. All of these holes were excluded from the resource estimation update process.

No other errors or omissions have been identified in the databases, and no corrections were introduced to the databases by AMC Consultants. Those intervals that were not gamma-logged, but assayed, were combined in a data file having the combined uranium data.

14.4 Exploratory Data Analysis – Statistical Analysis

Classical statistical analysis was updated twice for the deposit. The first study was carried out to determine the distribution characteristics of the uranium grades.

Figure 14-1 summarizes the statistical properties of the unrestricted assay databases for uranium. The statistical parameters for all uranium grades are shown in Table 14-2.

The histogram for unrestricted uranium grade population has a positively skewed log distribution and demonstrates that there is no apparent mixing of grade populations. The histogram does not indicate an obvious cut-off grade that could be used for interpretation of uranium mineralization. A decision was made to use the same nominal cut-off grade of 100 ppm for the subsequent update of interpretation of mineralized bodies which was used for the previous MRE update in 2019. The adoption of 100 ppm cut-off grade also reduces the residual effect of any radium halos by their exclusion, as gamma data is not accurate or precise below this level.

Once the uranium mineralization interpretation was updated for all new drillholes and wireframed, classical statistical analysis was repeated for the composited samples within the interpreted envelopes to meet the following objectives:

- To estimate the mixing effect of grade populations for uranium within the interpreted mineralized bodies.
- To estimate the necessity of separation of grade populations if more than one population was observed.
- To reveal the possible top-cut grades for uranium for grade interpolation.

The input sample file was flagged to exclude those intervals that appeared outside the wireframed mineralized envelopes for uranium. The modelled histogram for the uranium grades restricted within mineralized envelopes does not demonstrate apparent mixing of grade populations for uranium (Figure 14-1).

The lognormal histograms and cumulative probability plots were analyzed to determine the top-cut grades to be applied to the input analytical data before the geostatistical analysis. The majority of the input intervals with uranium grades were determined from the gamma logging results for 10 cm intervals. Thus, a decision was made that no top-cut grade values are applied on the analyzed intervals because the process of deconvolving of uranium grades from gamma-logging results usually is somewhat self-limiting for abnormally high grades. It was determined that top cutting is not required.

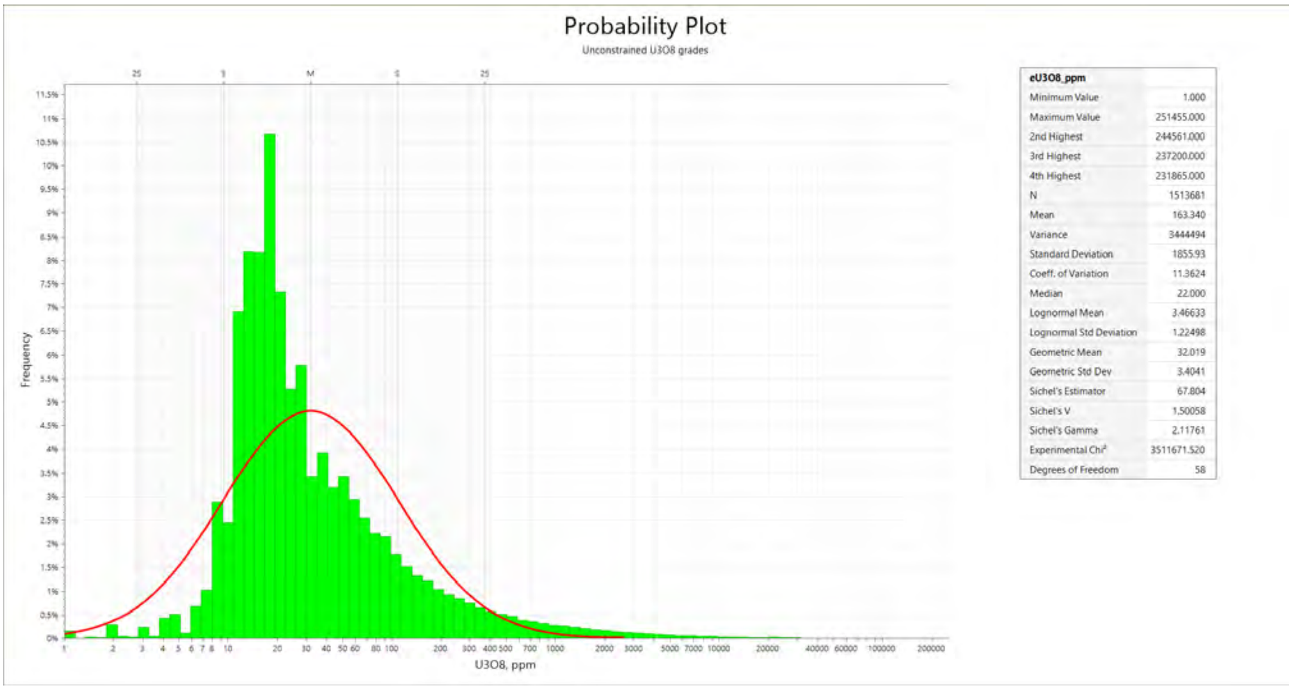


Figure 14-1: Log Histogram for Unrestricted Uranium Grades.

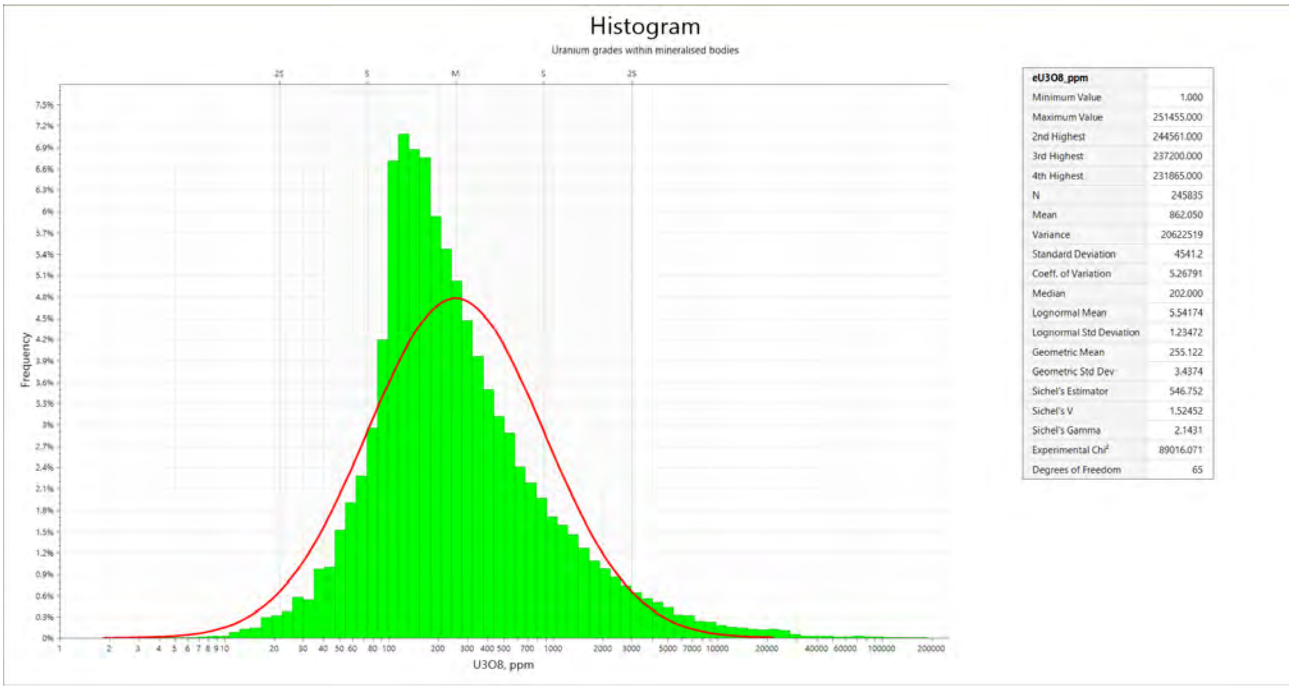


Figure 14-2: Log Histogram for Uranium Grades Within Mineralized Envelopes.

The coefficient of variation for the composited uranium grades is relatively high, which indicates that the possibility of modelling robust semi-variograms is relatively poor.

Table 14-2: Classical Statistics for Uranium Grades (Weighted on Length).

Minimum (ppm)	Maximum (ppm)	No of points	Mean (ppm)	Variance	Standard deviation	Coefficient of variation	Median (ppm)
Unrestricted Sample Intervals							
0	251,455	1,542,061	161	3,372,198	1,836	11.5	22
Intervals Within Mineralized Bodies							
0	251,455	245,957	859	20,459,029	4,523	5.3	202
0.5 m Composites Within Mineralized Intervals							
0	218,487	50,485	860	19,490,679	4,415	5.2	202

14.5 Lithological Model

A full lithological model was developed for the DASA deposit using logged lithological codes in the supplied database by CSA Global in 2019. AMC Consultants did not update the lithological model with the new drillholes potential differences were considered immaterial.

For the 2019 model, the geological data and all codes were imported into the Leapfrog software, where each main lithological unit was modelled along with the major known and logged faults. Firstly, four major fault planes were modelled, and subsequently the deposit was subdivided into five main fault blocks. All lithological units were modelled within each fault blocks separately, limited by the fault planes.

Example of the modelled lithology for one of the cross sections is shown in Figure 14-3.

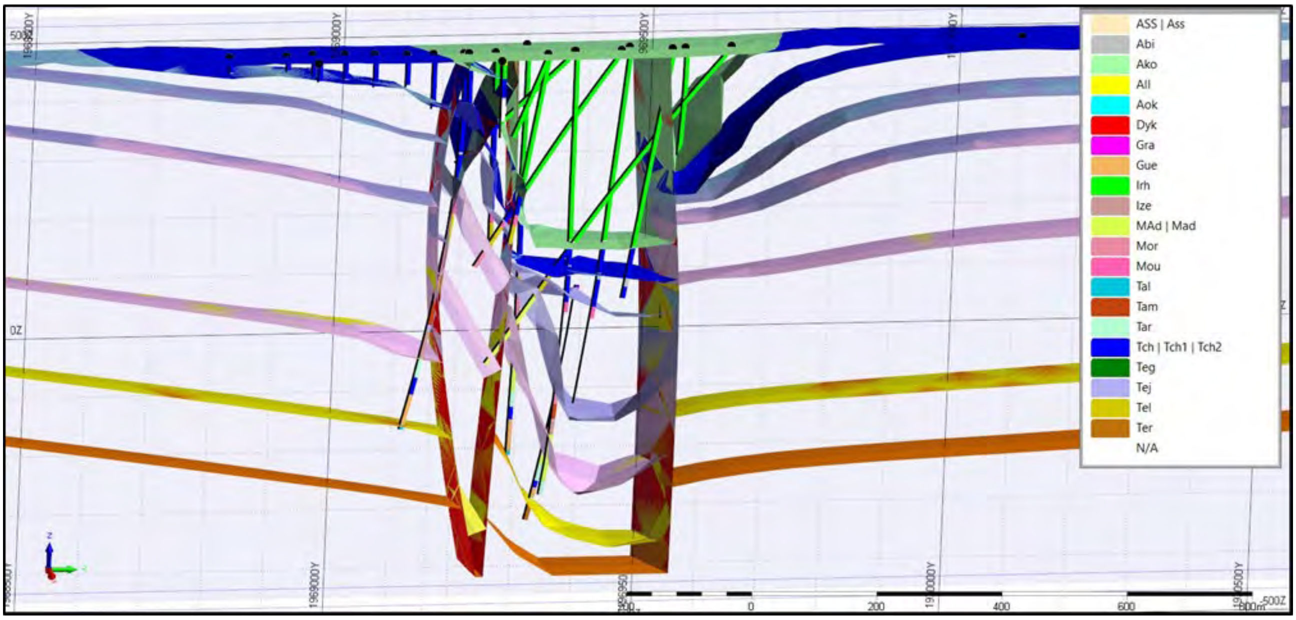


Figure 14-3: Lithological Model and Faults, Section 23 (359,900mE).

Source: CSA Global, 2019.

The developed lithological model was subsequently used to control the interpretation and wireframing of all mineralized bodies, which were digitized in line with the modelled lithological units and clipped to the modelled faults.

14.6 Interpretation of Mineralized Bodies

The grade compositing process was used to calculate the mineralized intervals using a 100 ppm eU_3O_8 cut-off grade. The calculated grade composites were displayed along the drillhole traces to assist with interpretation only. The interpretation process involved correlation of identified mineralized intervals between the holes along exploration lines and between the sections to make sure that the correct lens numbers would subsequently be assigned to the analytical data file.

The grade compositing process was used with the following input parameters:

- Cut-off value: 100 ppm eU_3O_8.
- Minimum composite length: 1 m.
- Minimum grade of final composite: 100 ppm eU_3O_8.
- Maximum consecutive length of internal waste: 0.5 m.
- Minimum grade * length: 200 ppm*m eU_3O_8.

Interpretation was updated interactively for all 56 north-south cross sections. The north-south sections were 50 m apart with some infilled holes within the Flank Zone in the central part of the deposit. When the interpretation of uranium grades was updated, each section was displayed in Micromine's Vizex display environment together with drillhole traces, grade composites, interval grade values and a slice through the lithological model. A total of 356 individual mineralized lenses were interpreted and modelled for the deposit. The following techniques were employed while interpreting and updating the uranium mineralization:

- Each cross section was displayed on screen with a clipping window equal to a half distance from the adjacent sections.
- All interpreted strings were snapped to the corresponding drillhole composited intervals, i.e. the interpretation was constrained in the third dimension.
- Internal waste within the mineralized envelopes was not interpreted and modelled separately. It was included in the composited grade intervals used for the resource estimation.
- The interpretation was extended perpendicular to the corresponding first and last interpreted cross section to the distance equal to half of the distance between the adjacent exploration lines, i.e. nominally 25 m or less; In this case, the interpretation honoured the general direction of the structure and the tendency for changes of the form of the geological body.
- If a mineralized envelope did not extend to the adjacent drillhole section, it was projected halfway to the next section keeping its thickness and terminated. The general direction and dip of the envelopes was maintained.
- All interpreted strings were clipped to the fault planes, and all interpreted envelopes were digitized in line with the main lithological structures of the deposit.
- If a mineralized envelope did not extend to the next drillhole within the interpreted exploration line, it was extrapolated halfway to the next drillhole keeping its thickness and terminated. The general direction and dip of the envelopes was maintained.
- If a mineralized envelope was at the topographic surface, it was extended above the topographic base. This was done to make sure there would be no gaps between the block model and the topographic base when the block model was built.

An example of an interpreted and updated section is shown in Figure 14-4, where thick red lines along drillhole traces are grade composites, purple hatched lines are slices through modelled wireframes, and hatched areas are different lithological units.

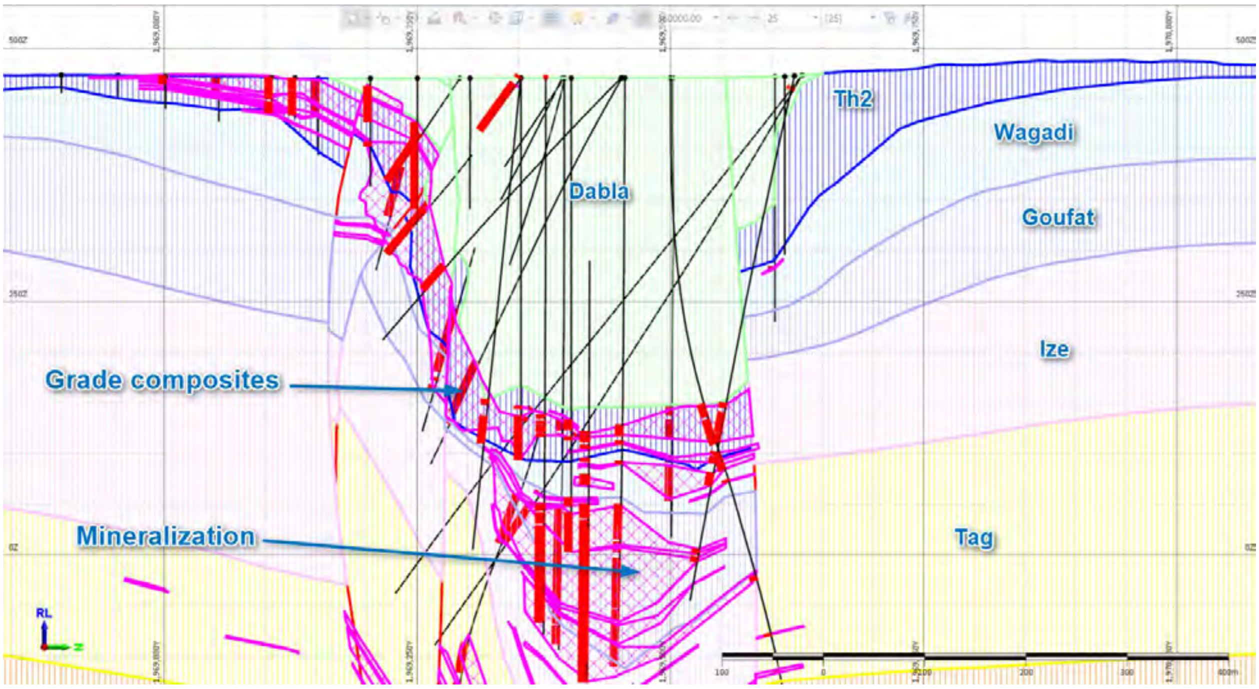


Figure 14-4: Example of Interpretation of DASA deposit – Section 360,000mE.

14.7 Wireframing

The interpreted strings were used to update three-dimensional (3-D) solid wireframes for the mineralized envelopes. Every cross section was displayed on the screen along with the closest interpreted section. If the corresponding envelope did not appear on the next cross section, the former was projected halfway to the next section, where it was terminated. Every mineralized envelope was reviewed and, where necessary, wireframed or updated separately and individually. Mineralized bodies were extended and projected to the interpreted sub-vertical fault planes, where it was possible, and then terminated. Internal waste was included within the interpretations where continuity would be improved by doing so.

Figure 14-5 is a 3-D view of the modelled mineralized bodies where red collars are the drillholes drilled in 2022. A total of 356 mineralized wireframes were modelled or updated for the deposit. The modelled mineralized bodies between the faults or steeply dipping bodies generally represent the graben structure, while all other bodies outside the graben are generally flat and relatively shallow mineralized lenses.

All wireframe models were validated so that they are all solids (closed) and that they do not contain intersecting triangles. Their total volume was 77 million m^3 (Table 14-3).

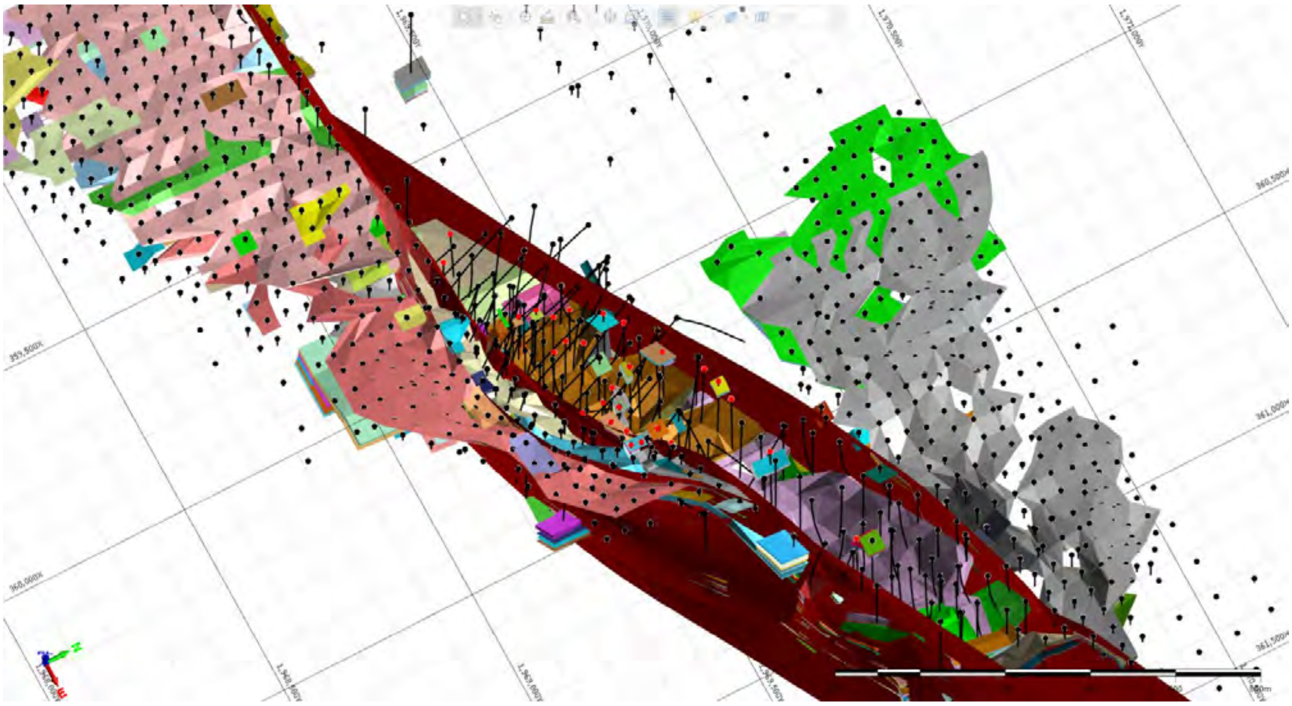


Figure 14-5: Oblique View of the Wireframed Uranium Mineralized Envelopes and Fault Planes (Looking Northwest).

Table 14-3: Number of Mineralization Envelope Wireframes and their Total Volume.

Number of	Volume (m^3)
356	77,143,398

14.8 Drillhole Data Selection and Compositing

Drillhole data selection is a standard procedure which ensures that the correct samples are used in classical statistical and geostatistical analyses and grade interpolation processes. For this purpose, the solid wireframes for each mineralized envelope were subsequently used to select and flag the drillhole sample intervals for each of the modelled mineralization envelopes.

Visual validation of the flagged samples was carried out to make sure the correct samples were selected by the wireframes.

Classical statistical analysis was then repeated for those uranium grades within the mineralization envelopes.

The majority of intervals in the raw analytical data file were 10 cm based on gamma-logging. The data was composited from raw intervals again, this time utilizing the mineralization flagging. It was decided to composite all intervals to 0.5 m. Thus, the selected samples within each mineralized envelope were separately composited over 0.5 m intervals, starting at the drillhole collar and progressing downhole.

Compositing was stopped and restarted at all boundaries between mineralized envelopes and waste material.

14.9 Dynamic Search

The 2018 to 2022 drilling programmes confirmed that the Flank Zone is a set of steeply dipping mineralized bodies with variable northeast dip without any apparent "steps" and separated by fault planes. This improved the understanding of the deposit geology and helped to develop a more robust interpretation of mineralized lenses in that area to update the model.

It was decided that a dynamic search would deliver the most robust results for the deposit. To set up the dynamic search, it was necessary to assign azimuth, plunge, and dip values to each cell in the block model. That was achieved using the following methodology:

- Strings were digitized along the deposit strike in plan-view as shown in Figure 14-6 (purple lines).
- Strings was digitized for every 50 m spaced section approximately parallel to the deposit strike. All strings were digitized through the central parts of the mineralization wireframe slices. These strings (Figure 14-7) represented an approximate plunge for each modelled lens (green lines).
- A set of strings was digitized for every 50 m spaced section approximately perpendicular to the deposit strike. All strings were digitized through the central parts of the mineralization wireframe slices. These strings (Figure 14-8) represented an approximate dip for each modelled lens (red lines).
- When all strings for azimuth, plunge and dip were digitized, they were "conditioned", i.e., points were inserted in such a way that the distance between points along strings would not be greater than 10 m.
- Azimuth for each pair of points along strings was calculated and recorded in the string file which was digitized for the deposit strike.
- Inclination for each pair of points along strings was calculated and recorded in the string file which was digitized for the plunge of the lenses.
- Inclination for each pair of points along strings was calculated and recorded in the string file which was digitized for the dip of the lenses.
- All calculated values for strings were checked to have correct positive or negative values and corrected for consistency if it was necessary.
- The resultant strings were used to interpolate azimuth, plunge, and dip values into each cell of the block model. A spherical search was employed for this process.

The resultant block model had assigned azimuth, dip, and plunge values for each model cell, representing the general directions of the mineralized lenses.

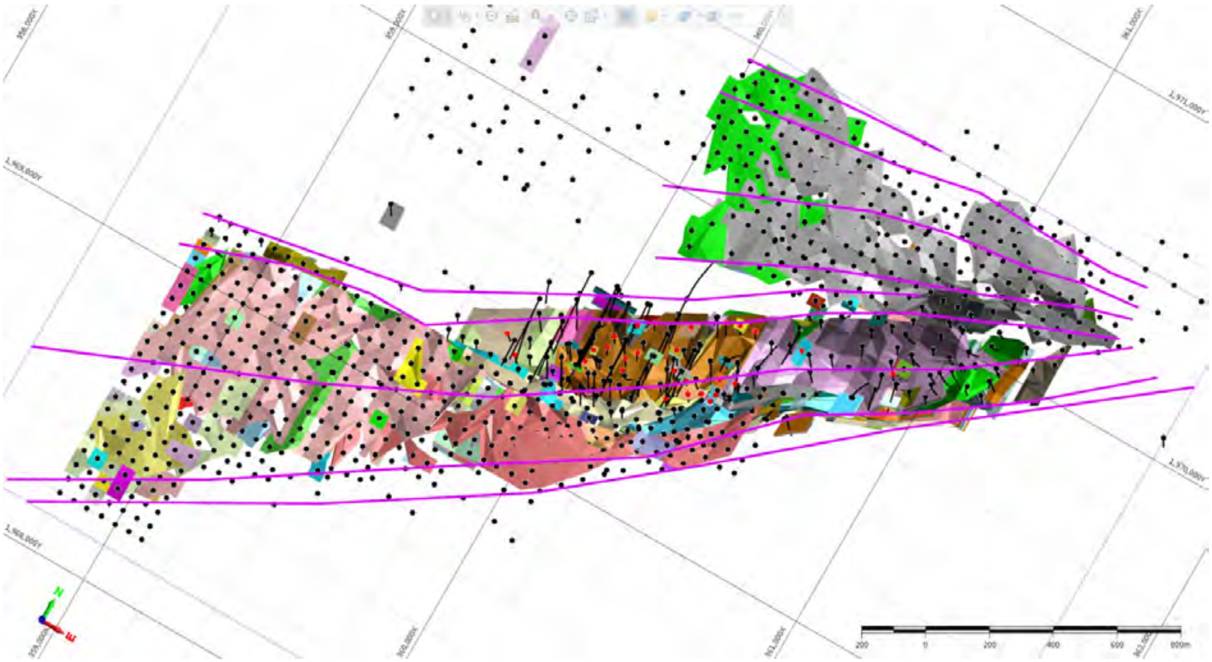


Figure 14-6: Strings for the Strike of Mineralization (Plan View).

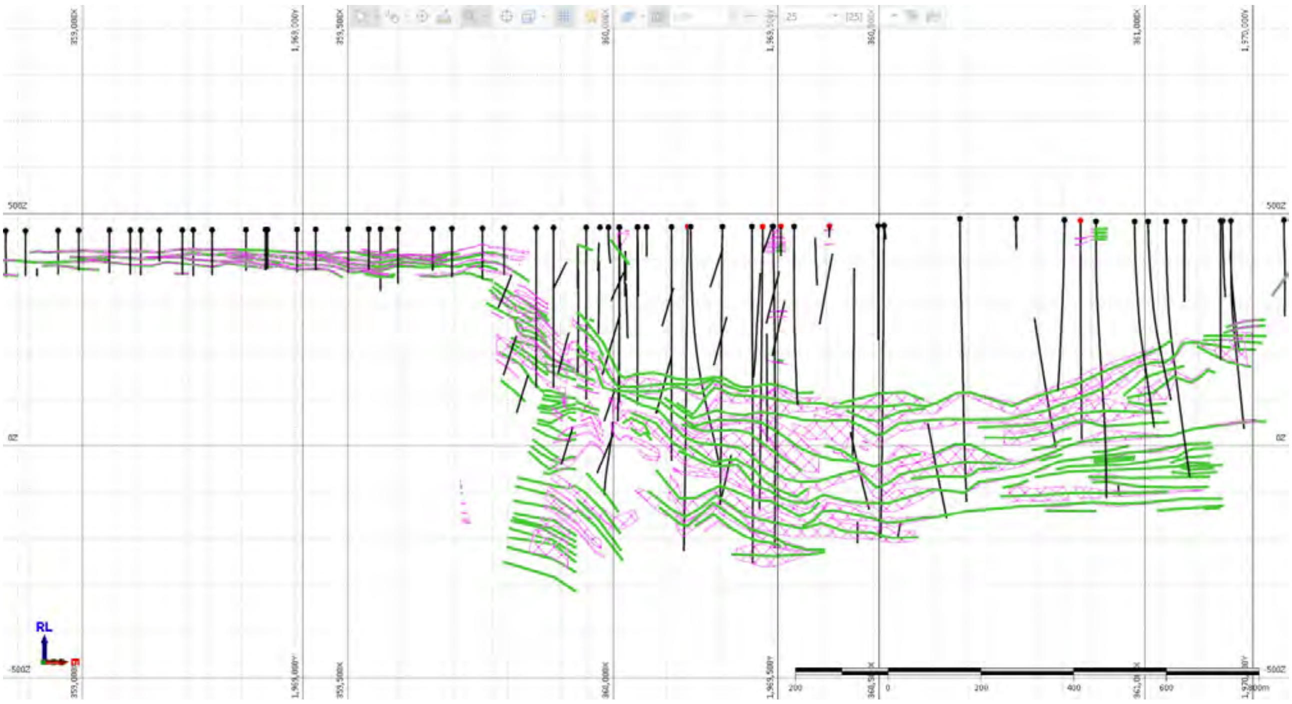


Figure 14-7: Strings for the Plunge of Mineralization (Section View Looking Northwest).

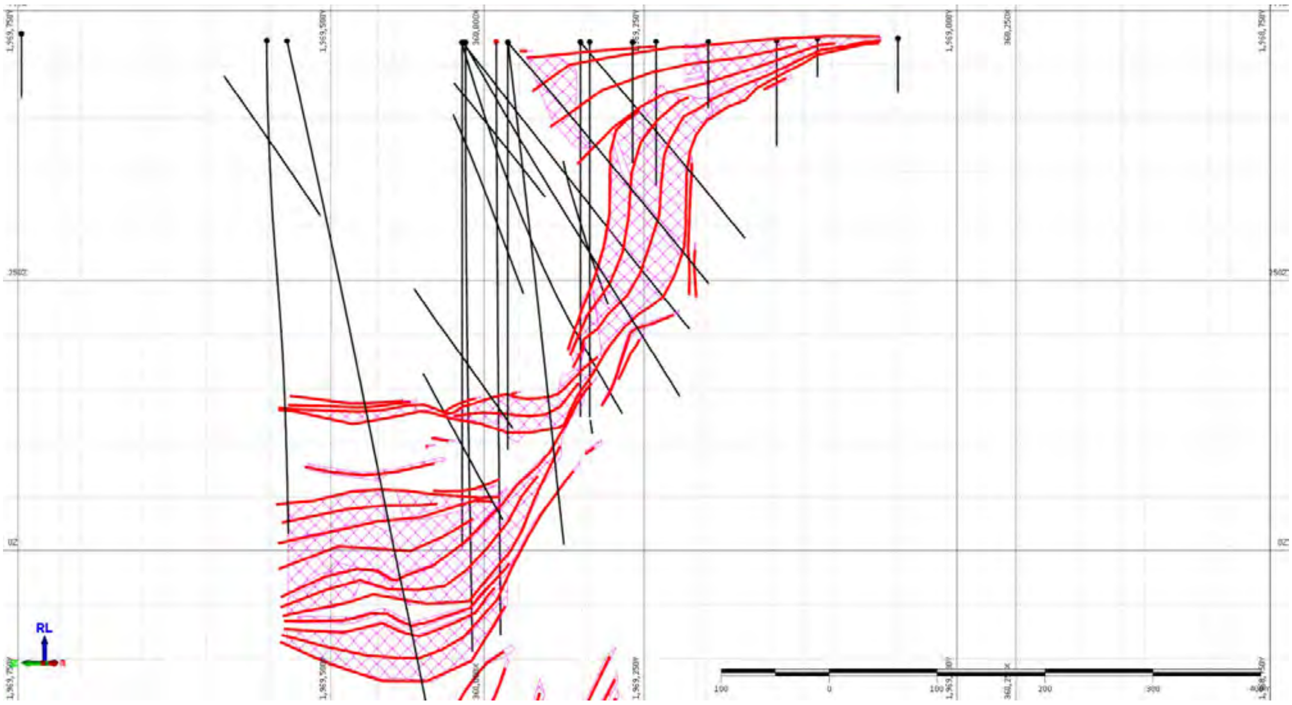


Figure 14-8: Strings for The Dip of The Mineralization (Section View Looking Northeast).

14.10 Variography

The geostatistical analysis was repeated for the updated composite data. It was found that the modelled variograms were very similar to those used for the 2019 Mineral Resource.

Downhole experimental variogram was modelled to estimate the expected nugget variance for uranium grades (Figure 14.10). The estimated nugget variance was then used to model directional variogram models.

A variogram map was then generated in plan view to establish the direction of maximum grade continuity. The map clearly demonstrated that the azimuth of maximum continuity is 55°, which generally matches with the overall strike of the mineralized bodies (Figure 14-9). The directions for variogram models for the major, semi-major and minor axes were established as 55° azimuth, 0° dip; 145° azimuth, 0° dip; and vertical respectively. However, it should be noted that the grade interpolator with dynamic search applied variable directions to the variogram models in accordance with the orientation parameters estimated for each model cell.

It was found that robust absolute variograms are difficult to model most likely due to the high coefficient of variation of uranium grades. Therefore, pairwise relative variogram models were calculated and modelled for the composited uranium sample file without applied top-cut grades (Figure 14-11 to Figure 14-13).

All modelled experimental variograms (Table 14-4) were exponential and spherical with three nested structures. The obtained variogram ranges were used to determine the search radii parameters. The latter were used in the grade interpolation processes.

Table 14-4: Modelled Variograms Characteristics.

Type	Axis	Azimuth	Dip	Nugget (%)	Partial sills (%)			Ranges (m)		
				C_0	C_1	C_2	C_3	A_1	A_2	A_3
Rel. exp. and Spherical	Major	55	0	8	12	50	30	1	6	100
	Semi-major	145	0					1	28	85
	Minor	145	90					1	9	46

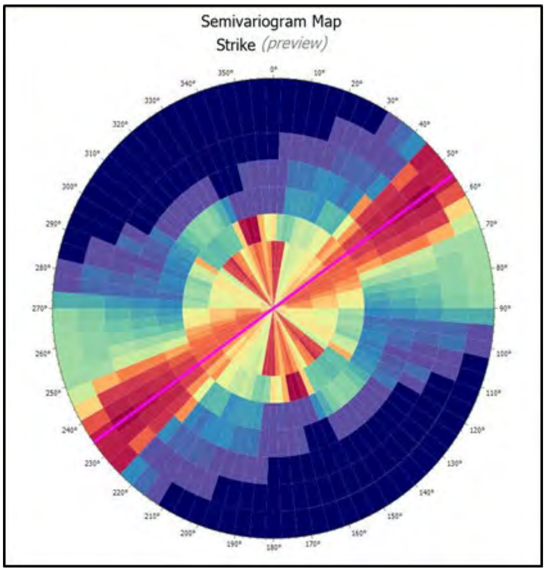


Figure 14-9: Variogram Model Map in Plan View.

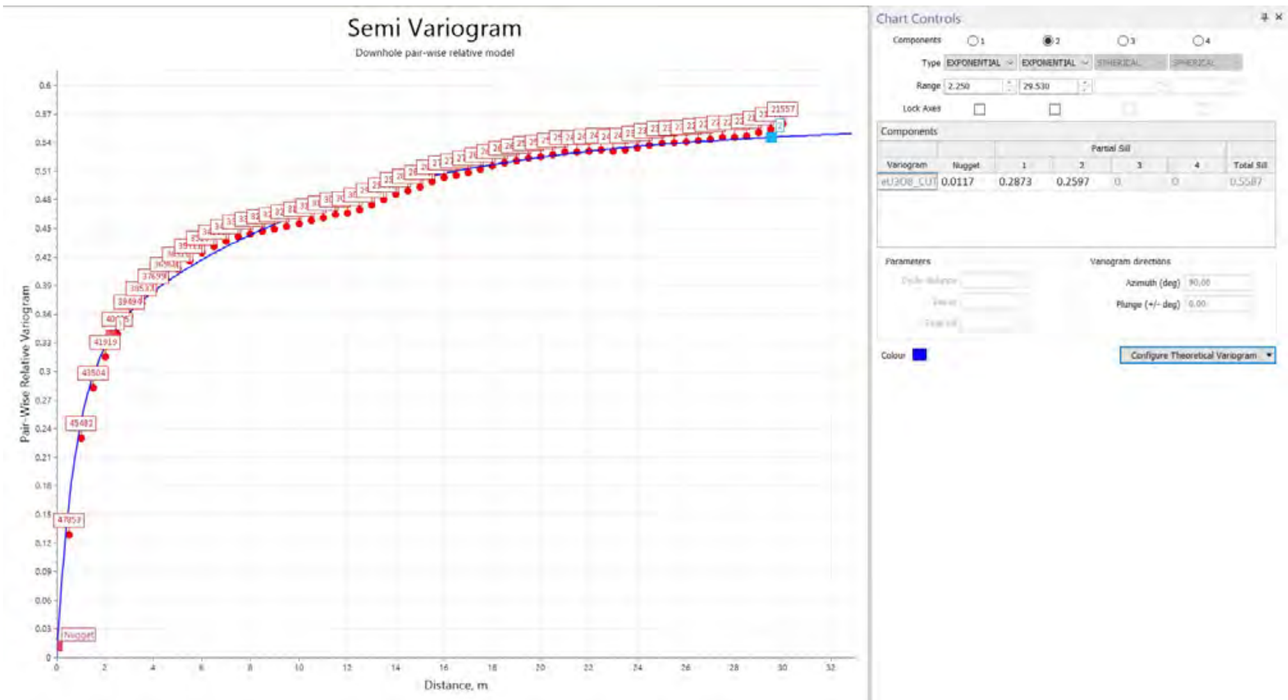


Figure 14-10: Downhole Pairwise Relative Variogram Model.

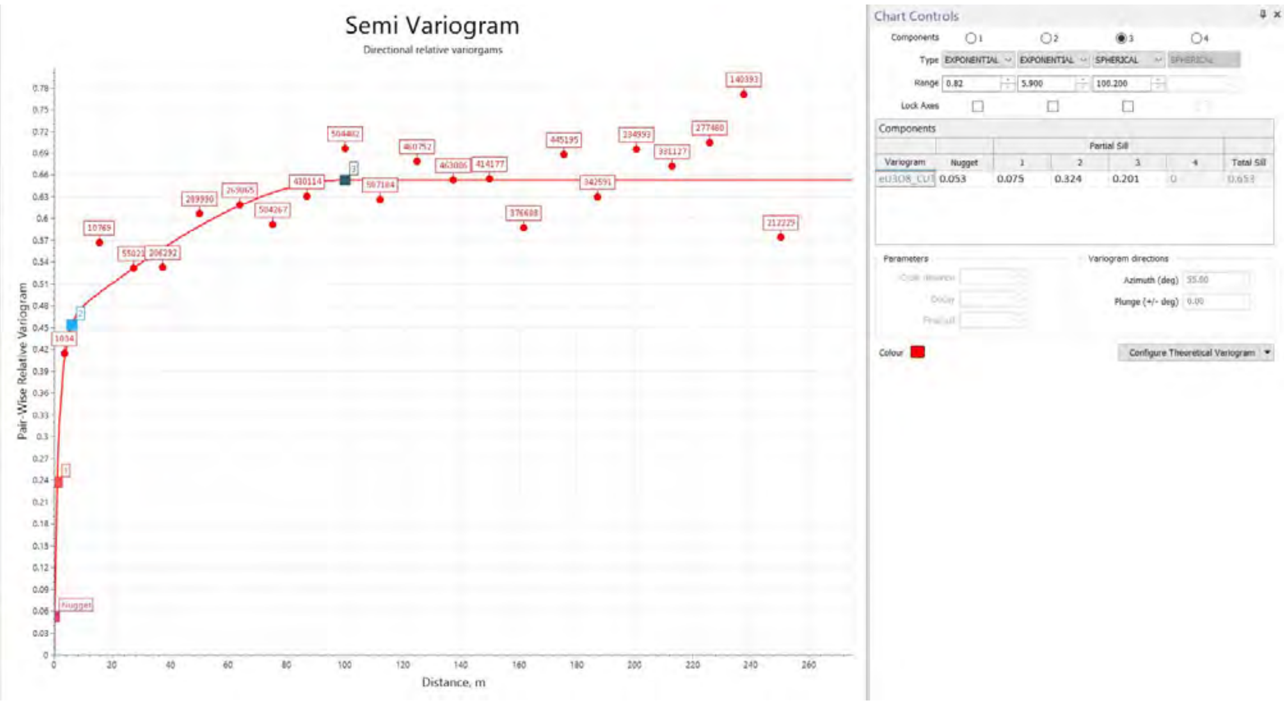


Figure 14-11: Pairwise Relative Variogram Model for The Major Axis.

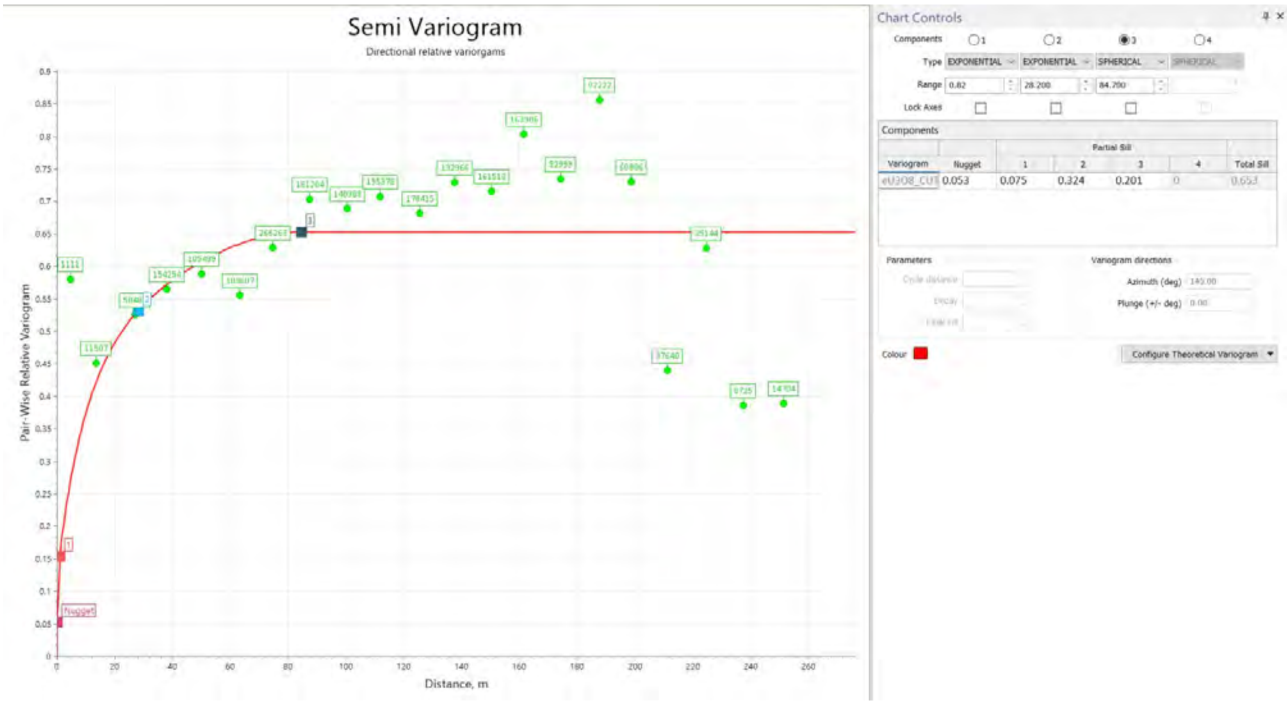


Figure 14-12: Pairwise Relative Variogram Model for The Semi-Major Axis.

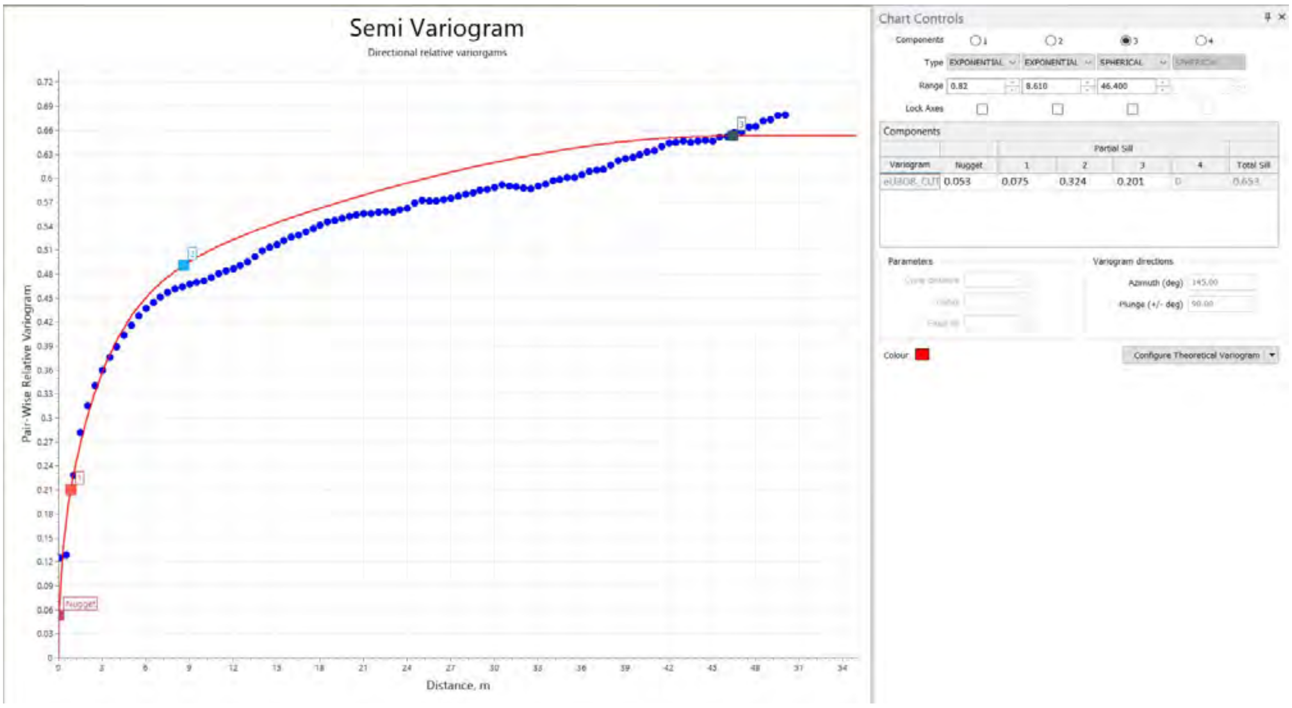


Figure 14-13: Pairwise Relative Variogram Model for The Minor Axis.

14.11 Block Modelling

An empty block model was created within the closed wireframe models flagged for each of the mineralized envelopes. The block model was then restricted to blocks below the topography surface (i.e. all the model cells above the surface were deleted from the model file).

Block model parameters are shown in Table 14-5

Table 14-5: Block model parameters

Axis	Extent (m)		Block size (m)	Maximum sub-celling (m)	Number of parent blocks
	Minimum	Maximum			
Easting	358,435	361,615	10	1	318
Northing	1,968,065	1,970,755	10	1	269
RL	-486	502	4	1	247

The initial filling with a corresponding parent cell size was followed by sub-celling where necessary. The sub-celling occurred near the boundaries of the mineralization or where the model was truncated with the topographic surface. The parent cell size was selected on the basis of the exploration grid and general morphology of the mineralized bodies and to avoid the generation of excessively large block model. The sub-celling size was chosen to maintain the resolution of the mineralized bodies. The sub-cells were optimized in the model where possible to form larger cells.

The resultant block model attributes are listed in Table 14-6

Table 14-6: Block Model Attributes.

Field	Description
X	Easting, m
Y	Northing, m
Z	Elevation
_X	Easting block size, m
_Y	Northing block size, m
_Z	Elevation block size, m
DENSITY	Density values (dry), t/m3
WF	Modelled wireframe name (BODY1 to BODY356)
MRE	Code for MRE limits: 0 – not in the MRE area, 1 – inside the MRE area
CLASS	Classification: 2 – Indicated, 3 - Inferred
AZIMUTH	Azimuth for dynamic search, degrees
PLUNGE	Plunge for dynamic search, degrees
DIP	Dip for dynamic search, degrees
eU_3O_8 ppm	U_3O_8 interpolated grades, ppm

14.12 Grade Interpolation

Uranium equivalent (eU_3O_8) grades were interpolated into the empty block model using ordinary kriging (OK). The search ellipse and variogram models were dynamically oriented by the interpolator using the azimuth, plunge, and dip values from each model cell.

The OK process was performed at different search radii until all model cells were interpolated. The search radii were determined by means of the evaluation of the modelled variogram parameters. Each mineralized lens was estimated separately.

The first search radii for all lenses were selected to be equal to one third of the modelled variogram long ranges in all directions. Model cells that did not receive a grade estimate from the first interpolation run were used in the next interpolation with greater search radii equal to two thirds of modelled variogram long ranges in all directions. The third interpolation run employed radii equal to full modelled variogram ranges. The model cells that did not receive grades from the first three interpolation runs were then estimated using radii incremented by the full variogram ranges until all model cells were informed with uranium grade.

When model cells were estimated using radii not exceeding full modelled variogram ranges, a restriction of at least three samples from at least two drillholes was applied to increase the reliability of the estimates.

"Parent cell estimates" were employed (i.e. all sub-cells within each parent cell were informed by the same grade).

Each modelled lens was estimated individually without mixing of data points between lenses. The vertical search was limited to 10 m to honour better the vertical variability of grades.

De-clustering was performed during the interpolation process by using four sectors within the search neighbourhood. Each sector was restricted to a maximum of four points for all the lenses, and the search neighbourhood was restricted to an overall minimum of three points from at least two drillholes for the interpolation runs using radii within the semi-variogram ranges. The maximum combined number of points allowable for the interpolation was therefore 16. Discretization utilized 5-points by 5-points by 5-points. The discretization point estimates are then simply averaged for the final block grade estimates. The general parameters for the interpolation strategy are presented in Table 14-7

Table 14-7: Interpolation Parameters.

Interpolation Method	**Ordinary Kriging**			
Pass	**1**	**2**	**3**	**4**
Major, semi-major and minor search radii	33 by 28 by 3 m	67 by 57 by 7 m	100 by 85 by 10 m	200 by 170 by 20 m
Minimum number of points	4	4	4	1
Maximum number of points per sector	4	4	4	4
Number of sectors	4	4	4	4
Maximum number of points	16	16	16	16
Minimum number of drillholes	3	3	3	1

14.13 Bulk Density Values

Dry bulk density values were obtained during previous and recent exploration programmes on the deposit. Direct measurements of 3,594 core samples were taken and processed by GAC.

The density values were assigned to each model cell based on the average value from the density dataset collected and provided by GAC. Each model cell was assigned a density value of 2.36 t/m^3.

14.14 Mineral Resource Classification Strategy

The classification of the model was updated for those areas of the deposit where the confidence in geological continuity of the mineralized bodies was upgraded with additional drilling, where the exploration grid density was increased, and where the understanding of the deposit geology was supported by the structural and lithological model.

The Mineral Resource classification strategy utilized in this report is based primarily on geological confidence, search and interpolation parameters, and exploration drillhole density. Kriging variance was also used to assist with the classification. The specific requirements concerning the minimum number of samples and minimum number of drillholes used for grade interpolation for each block were applied and are tabulated in Table 14-7

The block model was displayed in Micromine's Vizex environment and colour coded according to interpolation runs. After visual inspection, it was decided that the classification of Mineral Resources could be based on exploration drillhole density and interpolation runs which were based on modelled variogram ranges. It was decided that the exploration grid of at least 50 m by 50 m would support the Indicated Resource category if blocks were estimated from at least two drillholes by search ellipse not exceeding the semi-variogram ranges. All the remaining model cells were classified as Inferred. No Measured Mineral Resource category has been applied to the DASA model.

The resource classification strategy is illustrated below (colours: green – Indicated, blue – Inferred) in Figure 14-14.

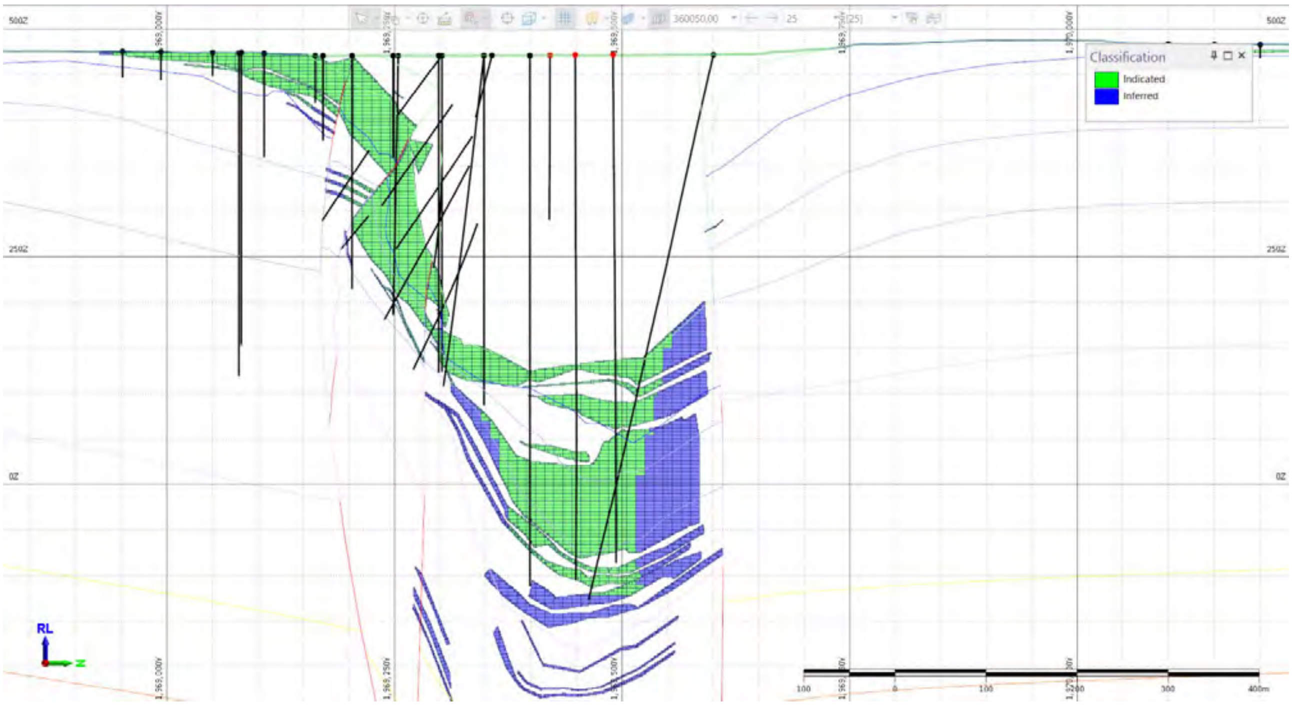


Figure 14-14: Resource Classification Strategy, Section 360,050mE.

14.15 Block Model Validation

Validation of the grade estimates was completed by:

- Visual checks on screen in cross section and plan view to ensure that block model grades honour the grade of sample composites.
- Statistical comparison of sample and block grades.
- Alternative interpolation using IDW methods, which returned very close results.
- Generation of swath plots to compare input and output grades by easting, northing and elevation.

14.16 Visual Validation

The block model with interpolated grades was displayed on screen along with the sample grades and colour coded. Visual validation demonstrated close correlation between modelled grades and composited samples (Figure 14-15).

14.17 Statistical Validation

The average eU_3O_8 grades in the model were compared with the average grades in the composited sample files. It was found that the global modelled grades were 17% relative lower than the grades in the composites (710 ppm eU_3O_8 in the block model versus 860 ppm eU_3O_8 in the composite file) for the combined categories. This difference is acceptable as it was expected due to the relative clustering of high-grade samples and smoothing or change of support related to the estimation process.

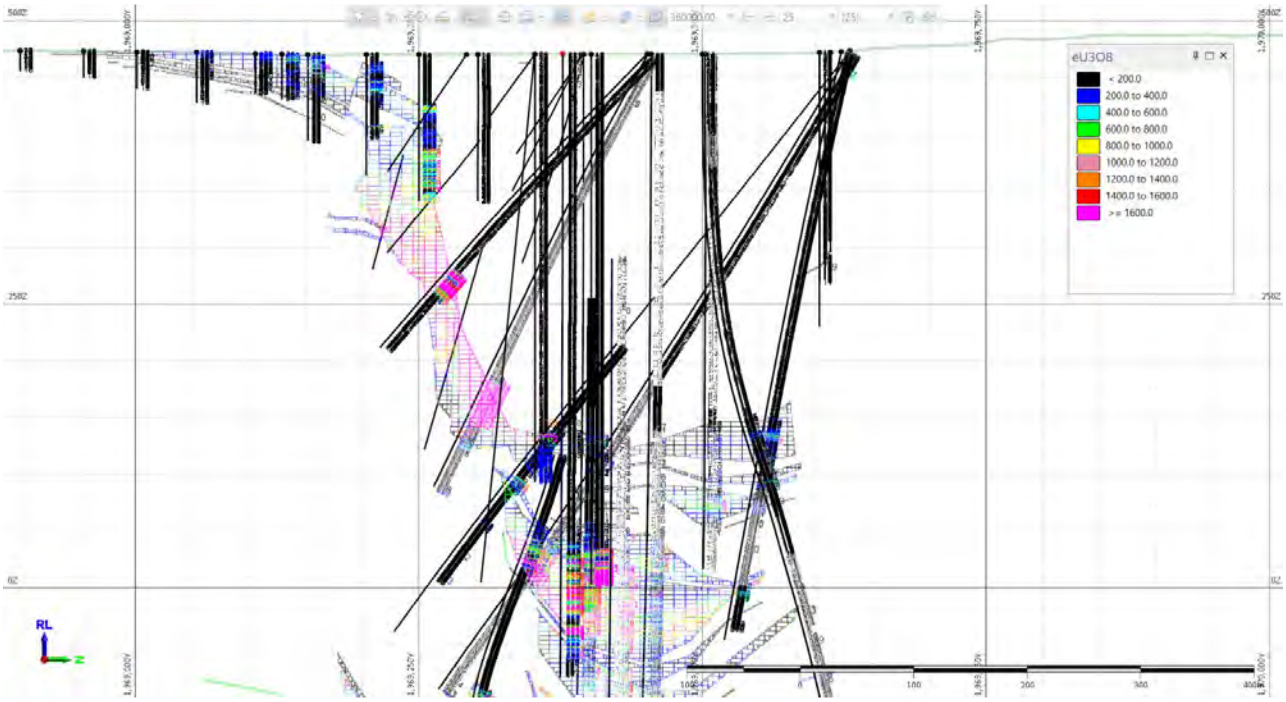


Figure 14-15: Visual Comparison of eU3O8 Grades in The Model vs. 0.5 m Composites (Section 260,000me, Looking West).

14.18 Swath Plots

Swath plots were generated for each 20 m bench and each 50 m vertical section in east-west and north-south directions. The results of this validation are shown from Figure 14-16 to Figure 14-18. The plots demonstrate close correlation between the modelled uranium grades and sample composites. It is apparent that the model has smoothed the composite grades to some extent, which is to be expected due to the volume variance effect. The Qualified Person was satisfied with the results of the model validation.

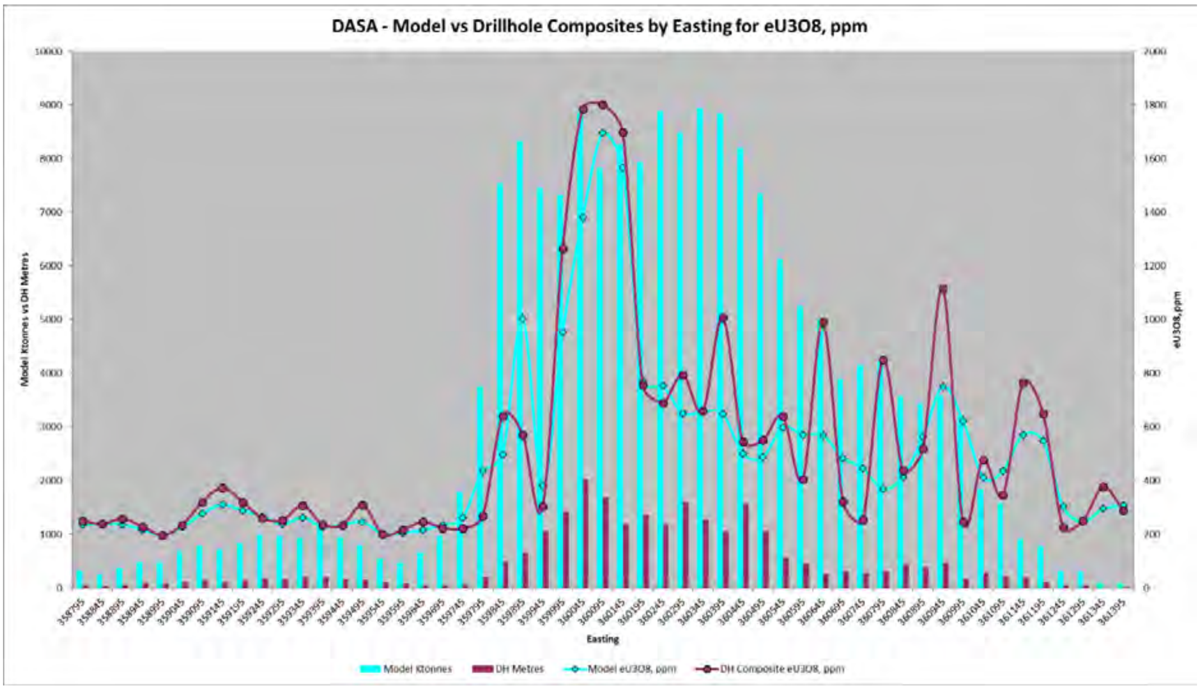


Figure 14-16: Swath Plot for 50 m Easting Sections.

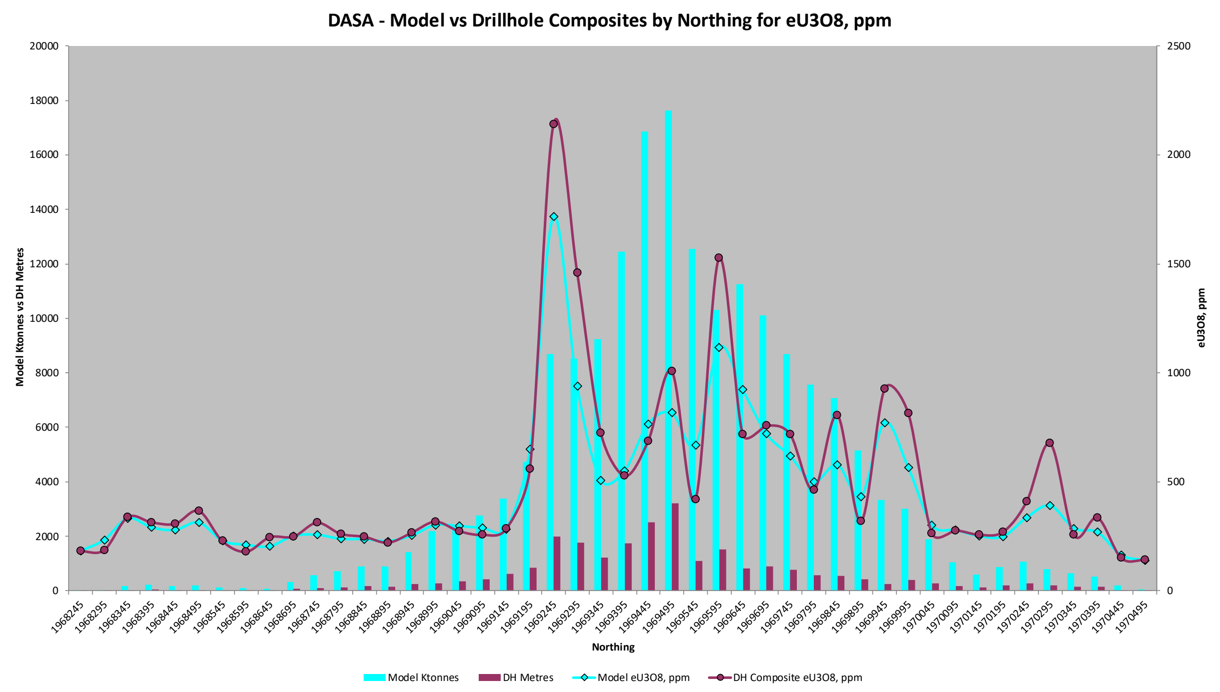


Figure 14-17: Swath Plot for 50 m Northing Sections.

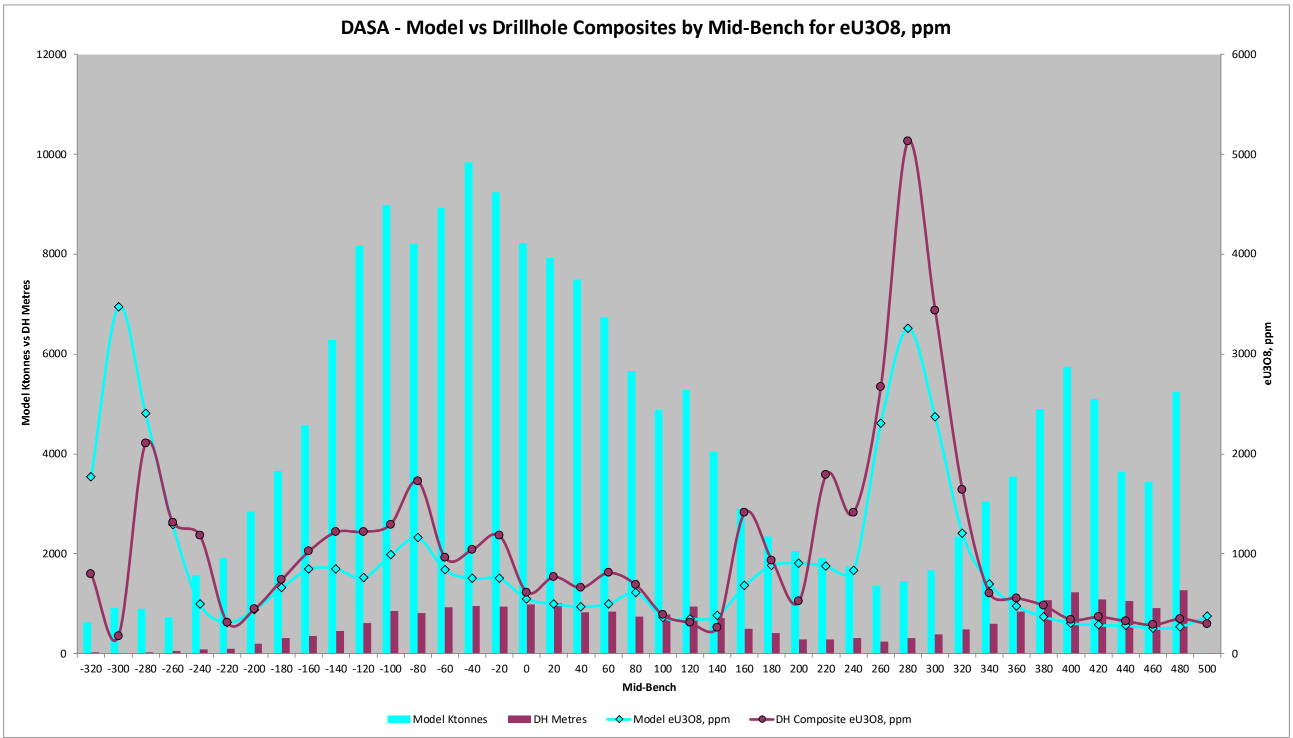


Figure 14-18: Swath Plot for 20 m Flitches.

14.19 Reasonable Prospects for Economic Extraction

CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM Council, 2014) require that resources have "reasonable prospects for economic extraction". This generally implies that the quantity and grade estimates meet certain economic thresholds and that the Mineral Resources are reported at an appropriate cut-off grade considering possible extraction scenarios and processing recoveries.

To ensure that reported resources have a reasonable prospect of economic extraction, a potential marginal economic cut-off grade was calculated using input economic parameters provided by GAC (Table 14-8, all costs are in USD).

Table 14-8: Economic Parameters for Underground Mining.

Parameter	Units	Value
Mining method		Underground
U_3O_8 price	$/lb	50
Dilution	%	2.1
Mining recovery	%	95
Mining cost	$/t	65
Processing cost	$/t	62
Processing recovery	%	94
G&A	$/t	36.9

These parameters resulted in estimated marginal economic cut-off of 1,487 ppm U_3O_8. Therefore, it was decided that a cut-off of 1,480 ppm U_3O_8 is reasonable for reporting of Mineral Resources for underground mining.

The block model was colour coded using the selected cut-off grade and limiting outlines were digitized on each section, that captured blocks above the cut-off which are potential for underground mining method. All blocks that occurred outside of the limiting boundaries were excluded from the report.

The author and Qualified Person deems that there are reasonable prospects for eventual economic extraction on the following basis:

- There is substantial mineralization with continuous high-grade zones that have potential for underground mining.
- The cut-off grades adopted for reporting (1,480 ppm for underground method mining) is considered reasonable, given the Mineral Resource is likely to be exploited by underground mining and processed using leaching techniques.
- The calculated marginal cut-off grade is based on the uranium price of $50/lb, while the forecasts of most markets reach $60/lb in 2024 (www.cmegroup.com).

14.20 Mineral Resource statement

Mineral Resources for the DASA deposit were estimated assuming that the deposit would be exploited by underground mining.

Mineral Resources for underground mining were estimated within the central part of the deposit only and above the cut-off grade of 1,480 ppm eU_3O_8. The cut-off grade was calculated using the economic parameters in the Table 14-8 and U_3O_8 price of $50/lb.

The Mineral Resources for underground mining are shown in Table 14-9 below.

Table 14-9: DASA Mineral Resources for Underground Mining as at May 12, 2023.

Category	Tonnes	eU_3O_8	Contained Uranium Metal
	Mt	ppm	Mlb
Indicated	10.1	4,913	109.3
Inferred	4.5	5,243	51.4

Notes:

Mineral Resources are classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (10 May 2014).

The MRE was prepared by Dmitry Pertel, MAIG, (AMC Consultants).

The effective date of the MRE is 12 May 2023.

A cut-off grade of 1,480 ppm eU_3O_8 has been applied for underground resources, assuming all resources to be mined from an underground decline access.

A bulk density of 2.36 t/m^3 has been applied for all model cells.

Rows and columns may not add up exactly due to rounding.

No Measured Mineral Resource or Mineral Reserves of any category were identified.

Mineral Resources are not Mineral Reserves and by definition do not demonstrate economic viability. This MRE includes inferred Mineral Resources that are normally considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves.

14.21. Factors that May Affect the Mineral Resource estimate.

The Qualified Person is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant issues that could potentially affect this MRE.

Additional technical factors which may affect the MREs include:

- Potential future conceptual study assessments of mining, processing, and other factors.
- eU_3O_8 price and valuation assumptions.
- Changes to the assumptions used to estimate eU_3O_8 content (e.g. bulk density estimation, cut-off grade selection for interpretation of mineralized zones, grade modelling methodology).
- The uranium disequilibrium factor was defined based on comparison of chemical assays with gamma logging. There is no investigation of radon degassing factor which may influence the gamma activity to some extent. The effect of this issue on the entire project is not likely to be material to the Project but may have localized effects. The available number of chemical assays and results of XRF for uranium grades support the reliable calculation of eU_3O_8 grades that were the basis of the modelled grades. The comparison of chemical assay and gamma data suggests a disequilibrium factor which is believed to be close to 1.0. Comparison of gamma logging with radium assays in closed cans as well as radium assays in closed cans with uranium assays could assist to define reliably the radiological factors.
- Geological interpretation (revision of lithological contacts, mineralization domains, modelling of internal waste domains, etc.).
- Changes to design parameter assumptions that pertain to the resource constraining conceptual underground stopes design, including mining recovery.
- Changes to geotechnical and mining assumptions; or the identification of alternative mining methods.
- Changes to process recovery estimates if the metallurgical recovery in certain domains is less or greater than currently assumed.
- Operation in Niger could have some political risks, which could potentially be mitigated by being a strategic commodity in a mining friendly jurisdiction.

14.22. Difference from Previous Resource Estimate

The Mineral Resource reported herein differs from the previous resource estimate completed by CSA Global in 2019 as the previous estimate was reported assuming combining open cut and underground mining methods with corresponding reporting cut-offs of 320 and 1,200 ppm eU_3O_8. A conceptual open pit was not used to constrain the model in the current estimate as the deposit is now assumed by GAC to be mined only by underground methods.

The reporting cut-off grade was also updated using revised input economic parameters, and it now 1,480 ppm eU_3O_8 instead of 1,200 ppm that was used in 2019.

The lithological model that controls mineralized bodies was not updated in this study which is not considered to be material as all new drillholes confirmed the validity of the lithological model developed in 2019.

The global difference with varying reporting cut-off grades is shown in Table 14-10. It is apparent from the table that additional drilling completed in 2022 resulted in updated classification of Mineral Resources and, therefore, also in more metal that was classified as Indicated. The global average grades were similar, being slightly lower for the Indicated category, but higher for the Inferred category in 2023 comparing with 2019 estimate.

Table 14-10: Global Difference Between 2023 and 2019 Estimates.

Cut-Off, eU_3O_8, ppm	Category	May 2023 revised estimate			July 2019 estimate			% Change
		Tonnes (Mt)	eU_3O_8 (ppm)	Contained Metal (Mlb)	Tonnes (Mt)	eU_3O_8 (ppm)	Contained Metal (Mlb)	Contained Metal (Mlb)
100	Indicated	103.6	803	183.5	81.6	718	129.1	42
	Inferred	71.0	636	99.5	96.1	606	128.4	-23
320	Indicated	44.9	1,602	158.5	32.0	1,530	108.0	47
	Inferred	25.4	1,435	80.4	35.0	1,333	102.7	-22
1,200	Indicated	12.6	4,201	117.1	7.9	4,483	78.0	50
	Inferred	5.9	4,320	56.1	8.4	3,783	69.9	-20
1,500	Indicated	10.1	4,926	109.6	6.2	5,328	73.1	50
	Inferred	4.4	5,349	51.5	6.3	4,563	63.7	-19
2,500	Indicated	5.7	7,258	91.0	3.6	7,849	61.9	47
	Inferred	2.4	8,211	43.2	3.4	6,838	51.4	-16
10,000	Indicated	0.9	22,185	43.5	0.6	24,401	31.1	40

Cut-Off, eU_3O_8, ppm	Category	May 2023 revised estimate			July 2019 estimate			% Change
		Tonnes (Mt)	eU_3O_8 (ppm)	Contained Metal (Mlb)	Tonnes (Mt)	eU_3O_8 (ppm)	Contained Metal (Mlb)	Contained Metal (Mlb)
	Inferred	0.6	18,362	25.3	0.8	14,598	25.3	0

Additionally, the significant sampling programme with XRF analysis of core samples resulted in the more accurate estimation of the uranium disequilibrium factor and in a more robust calculation and justification of the eU_3O_8 grades. The Qualified Person is satisfied with the quality of the provided information related to both the grades calculations, and justification of the disequilibrium factor.

15. MINERAL RESERVE ESTIMATES

15.1. Overview

Mineral Reserves for the Dasa Uranium Project have been estimated based on the geology and Mineral Resource Estimate discussed in the preceding chapters of this report. An engineering design and costing exercise has been undertaken to Definitive Feasibility Study (DFS) levels of accuracy which will support the Mineral Reserve Estimate subject to a positive economic outcome of the evaluation. The DFS has addressed all required aspects of the project to enable the estimation of Mineral Reserves and is discussed in detail in the following report chapters, and the sub-sections below summarise this work undertaken to motivate the Mineral Reserve Estimate.

15.2. Mineral Reserve Estimation Approach

Mining and Mining Modifying Factors

The selected mining method for the Dasa orebody is a transverse Long Hole Open Stoping (LHOS) method with a cemented hydraulic fill, it is noted that this method is a similar method proposed in the previous study work undertaken on the Dasa Project. The method is fully mechanised, and an appropriate fleet of mining equipment has been included in the design.

The identified mining areas will be accessed by a single decline developed from surface and a gradient of 8 degrees in the footwall of the orebody. Access to the stoping blocks will be at 22.5 m vertical intervals with a footwall drive developed along strike 20 m from the stopes. Stope access crosscuts will be developed at 16.5 m intervals off the footwall drive. Table 15-1 below shows the key geotechnical design criteria used for the selected mining method while Table 15-2 shows the key modifying factors used in the mining design.

Table 15-1: Geotechnical Design Criteria.

Item	unit	Value
Maximum stope span	metres	16.5
Sublevel spacing	metres	22.5
Maximum length of stope	metres	90
Minimum, backfill strength (vertical exposure)	kPa	400
Minimum stand-off distance between footwall drive and orebody	metres	20

Table 15-2: : Modifying Factors.

Factor	Unit	Value	Comment
Minimum mining width	m	4.5	Minimum width for LHD
Pillar loss	%	2	60 % extraction applied to sill pillar recovery
Mining Dilution	%	10	Zero Grade
Ore recovery (from mining)	%	95	
Cut-off grade	ppm	1500	Applied in MSO for stope optimisation

The mine ventilation system is a key aspect of the mine design due to the presence of radioactive elements in the air. The ventilation system designed is a once through system (no recirculation of air) and will replace the volume of air in the mine on average every 15 minutes. Ventilation of excavations within the orebody where radiation risk is higher is by use of an exhaust system which removes contaminated air from the workings immediately into the return airway system, ensuring that risk relating to exposure to radiation is always minimised.

Process and Metal Recovery

The proposed process solution for the Dasa Project is an acid leach followed by solvent extraction of the uranium from solution. The recommended plant recovery model allocates an overall recovery of 94.15% for the processed uranium ore. This is based on a determined 95.1% average uranium recovery identified in the test work recovery analysis report and 0.95% attributed to multiple soluble uranium losses.

The Dasa Plant has a capacity to treat 365,000 t/a of run-of-mine (RoM) at a head grade of 4,113 ppm U_3O_8. Thus, annual production (U_3O_8 equivalent) is approximately 2,868,000 lb/a over 23.75 years (3,647,000 kb/a during the first 12 years).

Mine Infrastructure and Services

The requirements and costs for all surface and underground infrastructure to support the mining of the Dasa orebody have been included into the DFS evaluation, including:

Surface

- Access roads and security.
- Bulk supplies for power and water.
- All buildings including offices, change houses and workshops.
- Tailing storage facility.
- Surface reticulation of power and water.

Underground

- Service water reticulation.
- Return water and pumping reticulation (including groundwater).
- Electrical reticulation.
- Backfill reticulation.
- Fixed underground infrastructure.

Costs

Capital and operating costs have been generated for the mine, process plant and mine support infrastructure described above.

Table 15-3 below provides a summary of the capital costs while Table 15-4 provides a summary of the operating costs.

Table 15-3: Capital Costs.

Description	Initial ($millions)	Sustaining ($millions)	Total ($millions)
Mining	58.8	218.7	277.5
Processing	83.2	38.9	122.1
Infrastructure	68.2	5.2	73.4
Total Direct Capital Costs	210.2	262.8	473.0
Indirect and Owner's Costs	60.9	30.0	90.9
Total (including Indirect Costs)	271.1	292.8	563.9
Contingency	37.2	29.9	67.1
Reclamation	0	15.9	15.9
Total Capital	**308.3**	**338.6**	**646.9**

Table 15-4: Operating Costs.

Cost Item	\$/lb U_3O_8 Recovered	\$/tonne of Feed
Mining cost	9.10	77
Processing cost	10.00	85
Overhead cost	6.51	55
Cash costs before royalties	25.62	217
Royalties	5.11	43
Total cash costs	30.73	260
Sustaining capital	4.74	40
AISC	35.47	300

Metal Prices

The metal price used in the DFS evaluation of the Dasa Project is \$75 per lb of U_3O_8.

Payable Mineral Resource Included in Mine Layout

To determine the optimum stope shapes, considering the design input parameters as defined in the mine design criteria DeswikSO® was used. This is a mineable shape optimizer which seeks to create stope shapes, based on pre-set input parameters, while optimising the extraction of ore from the resource.

A primary input to the DeswikSO® process is specification of the cut-off grade to apply. DeswikSO® will attempt to produce stopes that have a grade higher than the specified cut-off grade. Table 15-5 below shows the cut-off grade estimate used in the tope optimisation process.

Table 15-5: Cut-off Grade Calculation.

Item	Value
U_30_8 Price (\$/lb)	70
lb/kg	2,204
U_30_8 Price (\$ /kg)	154.32
Operating cost (\$/t milled)	
Mining	83

Processing	77
Overheads (G&A)	40
Operating cost ($/t milled)	**200**
Breakeven recovered grade (kg/t)	1.30
Metallurgical recovery (%)	92.45
Breakeven RoM grade (ppm) before royalty	1,402
Royalties (%)	7
Breakeven after royalty	**1,500**
Note: Operating costs used in stope optimisation may not be the same as those reported in the final operating costs as this process is based on preliminary data available at the start of the project.	

Other inputs into the stope optimisation include the stope dimensions and pillar locations, Table 15-6 below shows the input parameters used in the Dasa stope optimisation.

Table 15-6: Stope Optimisation Input Criteria.

Item	Unit	LHOS	Comment
Stope height (sub-level spacing)	m	22.5	Geotech guideline
Stope strike length	m	Min = 5, Max=100	
Minimum mining width	m	5	
Maximum stope width	m	16.5	Geotech guideline
Minimum stope dip	Degree	60	
Rib pillar width	m	5	One rib pillar in Block C at Y=360290
Sill pillar thickness	m	10	One sill pillar in Block A at approximately Z=300 m
Stope direction	azi º	325	Transverse stopes
Cut-off Grade	ppm	1,500	RoM grade from Stopes

Inferred resources were not targeted in the stope optimisation process. Only Inferred material which was included in a stope which contains mostly indicated material was included in the mining inventory. Where Inferred material was included in a mined stope, it was assigned a zero grade and treated as waste material. For a stope to be included in the mine design it needed to contain a minimum of 1000 tonnes of indicated resources. Figure 15-1 below shows the stope shapes produced by DeswikSO®.

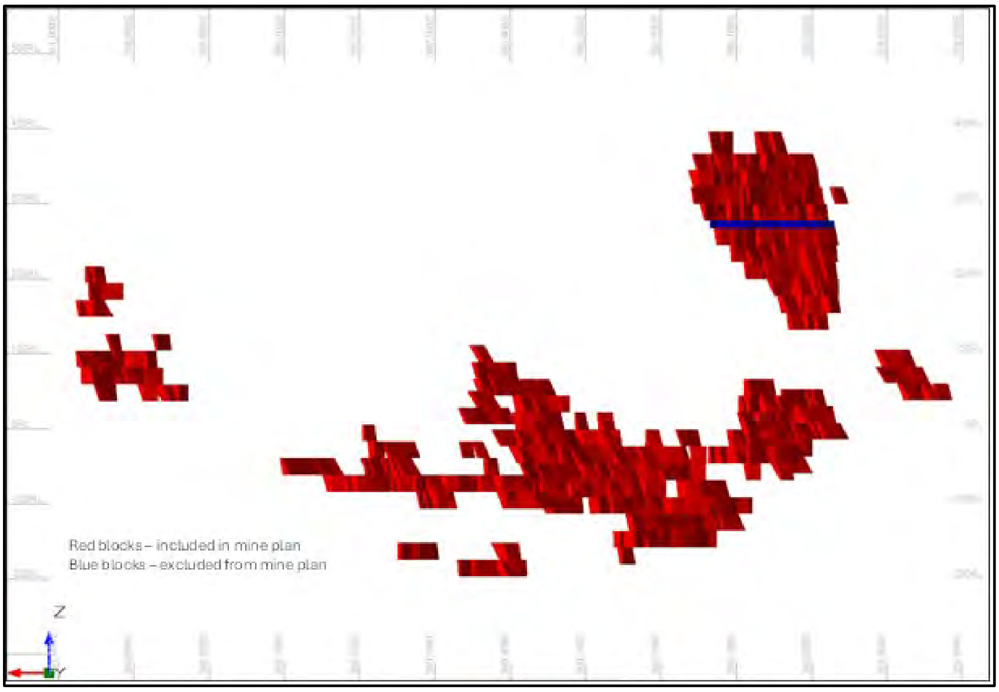


Figure 15-1: Stope Shapes Produced by DeswikSO®.

15.3. Mine Layout and Scheduling

Based on the mining method descriptions discussed above and the portions of the Mineral Reserve identified as payable, a mine layout has been generated in the Deswik® suite of mining software. The layout includes the development excavations and stopes for the LoM. The mine layout is shown in Figure 15-2 below.

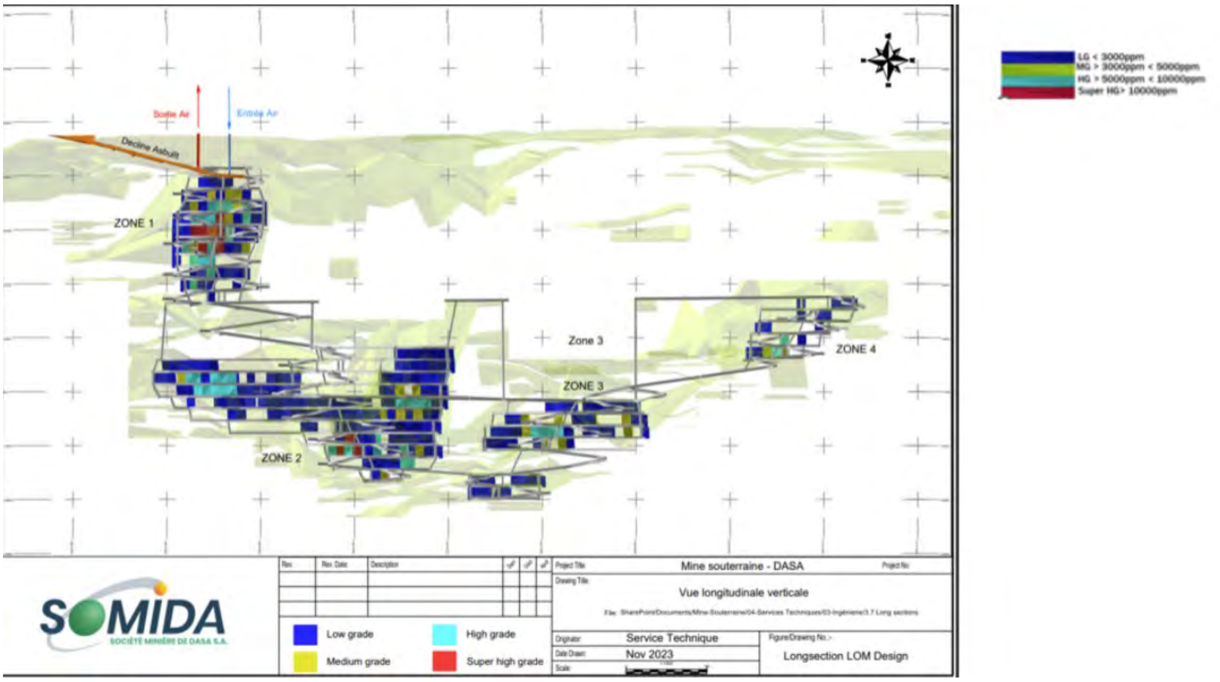


Figure 15-2: View of Underground Mine Looking North.

The mine layout was then exported to DeswikSched®, the mine scheduling module of the Deswik® suite. The activities making up the mine schedule were sequenced into a logical sequence. Mining productivities were estimated for each excavation type and used in the production of the mining schedule. The productivities by excavation type are shown in Table 15-7 below.

Table 15-7: Mine Scheduling Parameters.

Item	Unit	Value
Stope Productivity		
Long hole drilling.	m/month	5700
Stope Loading (LHD).	t/day	900
Backfilling.	m^3/day	400 (minimum)
Backfill curing time.	days	14
Development Advance		
Ramp.		75
Level Access.		60
Footwall drive.	m/month	60
Stope crosscut – Waste.		60
Stope crosscut – Ore.		50
Drop raising.		24

The activities were then scheduled considering the productivities detailed in Table 15-7 above, as well as the availability of resources (mining equipment) allocated to each activity.

The key aspects in the mining schedule are:

- SOMIDA had completed the boxcut excavation and 526 m of the ramp access development end December 2023. Scheduling of development started from survey positions of as-built development as at end December 2023.
- The Upcast ventilation system is established by the completion of ventilation raise in August 2024.
- Level development commences on the 5367 Level in September 2024.
- First ore is produced from development in September 2024.
- Stoping commences in October 2025 on the 5323 Level.
- Steady state production of 32,000 tonnes per month of ore is achieved in January 2027.
- The mine has a life of just over 26-years.

A graphic production profile for the full mining schedule for the underground project is presented in Figure 15-3 below.

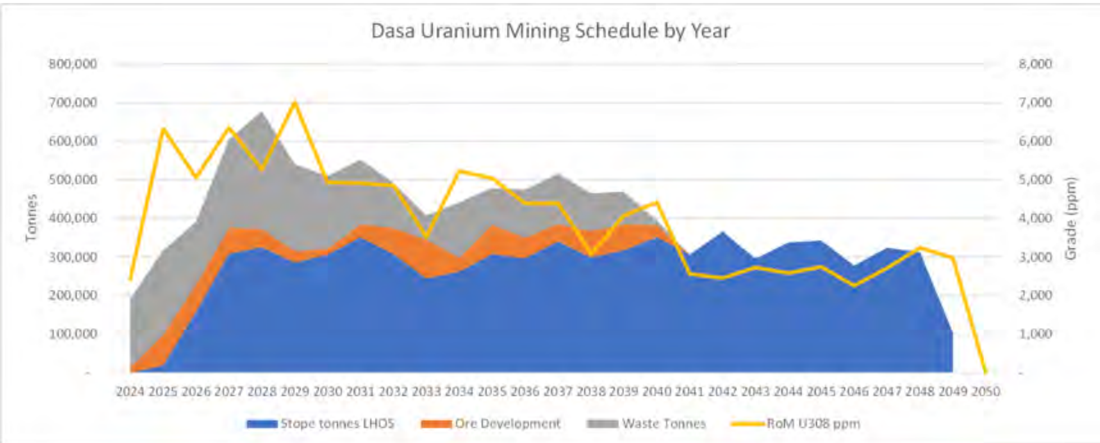


Figure 15-3: Dasa Phase 1 Underground Production Profile.

15.4. Permitting

The Dasa Mining Permit and an Environmental Certificate of Conformity has been issued by the Republic of Niger, the Dasa Project is therefore fully permitted for commercial production.

15.5. Mineral Reserve Estimate

Based on the techno-economic evaluation described above, a Discounted Cashflow (DCF) analysis was undertaken, the outcome of this DCF analysis was positive and it is therefore considered appropriate to declare a Mineral Reserve Estimate. Table 15-8 below shows the mining inventory generated by the mine planning exercise. The limited quantity of inferred resources is a result of inferred resources occurring within the stope shapes resulting from MSO evaluation described above. The inferred material amounts to 0.13% of the RoM tonnes. For purposes of the DCF analysis, the inferred material was assigned a zero grade and treated as waste material.

Table 15-8: Mining Inventory by Mineral Resource Class

Mineral Reserve Category	RoM tonnes	U_3O_8 ppm	U_3O_8 (t)	U_3O_8 (Million lbs)
Measured	-	-	-	
Indicated	8,035,902	4,119	33,097	72.964
Inferred	10,708	2,819	30	0.066
Total Mining Inventory	**8,046,610**	**4,117**	**33,127**	**73.031**

Based on the above analysis a Mineral Reserve Estimate has been made as shown in Table 15-9 below, the date of the Mineral Reserve is 28th February 2024. The Measured and Indicated Mineral Resources are inclusive of those Mineral Resources modified to produce the Mineral Reserves. As there are no Measured Mineral Resources included in the mine plan, all Mineral Reserves Estimated are in the Probable category. The Mineral reserve is based on the mining inventory above, with the inferred material in the mining inventory excluded from the Mineral Reserve.

Table 15-9: Mineral Reserve Estimate for Dasa Uranium Project (28th February 2024)

Mineral Reserve Category	RoM (Mt)	U_3O_8 (ppm)	U_3O_8 (kt)	U_3O_8 (Million lbs)
Proven Mineral Reserve	-	-	-	
Probable Mineral Reserve	8.05	4,113	33.1	73.0
Total Mineral Reserve	**8.05**	**4,113**	**33.1**	**73.0**

Notes on Mineral Reserves:

Mineral Reserves are reported with an effective date of 28th February 2024.

Mineral Reserves are reported using the 2014 CIM Definition Standards.
Inferred Mineral Resources, although included in the mining inventory, are excluded from the Mineral Reserve Estimate.
Measured and Indicated Mineral Resources are inclusive of those Mineral Resources modified to produce the Mineral Reserves
The Qualified Person responsible for Mineral Reserve Estimate is A D Pooley
Differences between tonnes, grade and contained metal content are due to rounding.

15.6. Factors Which May Impact Mineral Reserve Estimates

Factors that may materially impact the Mineral Reserve estimates are listed below. The technical risks listed have been mitigated as far as possible based on the site-specific data provided by Global Atomic Corporation and appropriate test work, other risks relating to the financial aspects of the project are generic to all mining projects.

- Changes to geotechnical conditions that have been modelled for the site resulting in an impact to the mining design and subsequent impact on mining costs, grade, and extraction ratios.
- Increase in the estimated groundwater inflows into the mine resulting in the requirement to increase water handling and disposal capacity at increased cost.
- Changes to input assumptions for the cut-off grade calculation and stope optimiser software which may impact on the volumes of payable Mineral Resources.
- Changes to the metallurgical recovery when scaled up to full size plant - impacting on revenue.
- Changes in economic assumptions over time relating to uranium price and exchange rates.
- Country risk including changes to tax laws and permitting requirements.

16. MINING METHODS

16.1. Introduction

Access to the underground workings will be via a single access decline developed from surface. The decline will serve as an intake airway and conduits for the transport of rock, personnel, and material. The decline will be developed at a nominal inclination of -8°, or 1:7 and will be sized to accommodate the selected mining equipment as well as the required intake ventilation at suitable airflow velocities. The decline system will be located in the footwall of the orebody.

Access to the orebody will be established at selected elevations by developing a crosscut from the decline towards the orebody, in a direction approximately perpendicular to the strike of the orebody. A footwall drive will be developed approximately 20 m from the orebody, along strike. The stope access crosscuts spaced at 16.5 metre centres will be developed from the footwall drive across the orebody. The stoping method proposed for Dasa is Long-hole Open Stoping (LHOS) with cemented backfill.

The level spacing is nominally 22.5 vertical metres. The general sequence of stoping will be bottom-up. Mining will commence on the lowest sublevel level in a mining block, progressing upwards towards a sill pillar, separating the mining block from the one above. Stopes will be filled with cemented hydraulic fill (CHF). The use of the cemented backfill will allow the mining of the sill pillars once the mining of a stope panel is completed.

The mine has been divided into 5 zones which are formed by areas of higher grade, which are targeted in the mine plan.

Figure 16-1 shows a vertical projection isometric view of the mine layout with the mining zones identified.

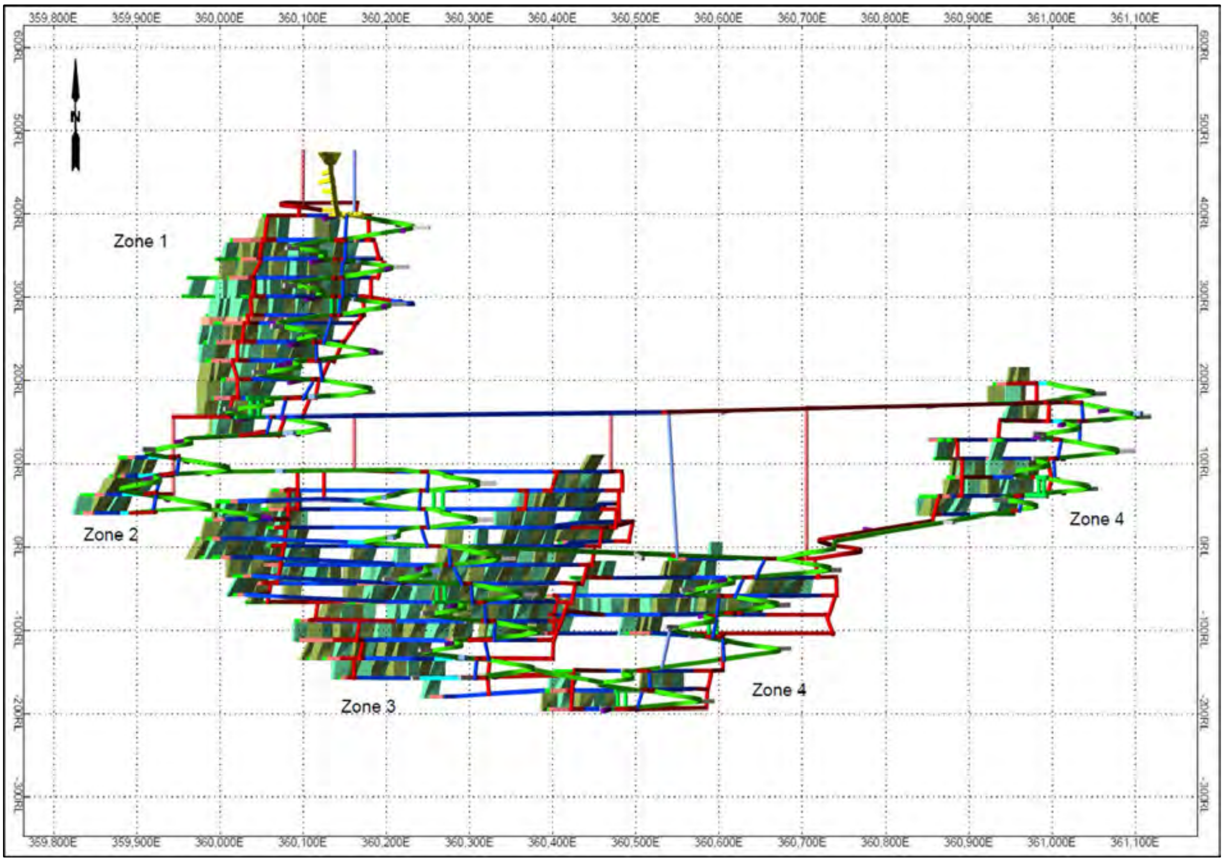


Figure 16-1: View of Underground Mine Looking North.

Mining will commence above the sill pillar on the 5277.5 m elevation in Zone 1 and progress upwards which the development of the access to the bottom of Zone 1 and the other zones continues.

Access to the top of Zone 5 is established via a haulage on the 5 152.5 m elevation. The haulage is twinned part of the way to provide an intake airway for Zones 4 and 5 and a return airway for Zone 5.

The mine is designed to produce approximately 1000 tonnes per day of Run of Mine (RoM) feed to the plant.

Ore will be trucked to surface using underground articulated dump trucks and tipped directly onto the RoM pad adjacent to the plant. Stockpiles will also be created adjacent to the RoM pad, which will assist in ore drying and ore blending. Waste will initially be trucked to surface, where it will be used for surface infrastructure earthworks. Thereafter, any waste trucked to surface will be placed on the waste dump. Once sufficient stope voids are available underground waste rock will be incorporated into the stope backfill material.

16.2. Geological Considerations for Mining Design

Key geological points which may affect the mining at the project are listed below:

- The orebody dip varies from 60 degrees in the Flank Zone to flat and 40 degrees in a South Easterly direction in the Graben Zone.
- The orebody in both the Flank Zone and Graben Zone is massive in nature with a width up to 70 m and a strike length of up to 140 m.
- Figure 16-2 and Figure 16-3 show a plan view and cross section through the Flank Zone which makes up a large proportion of the mine plan in the scoping study.
- Figure 16-4 shows a 3-D view of the orebody wireframes looking in a North-Westerly direction.

The selection of ore to be mined and definition of stopes is likely to be based on a grade cut-off. The orebody is not geologically constrained and there is unlikely to be a visual difference between ore and waste.

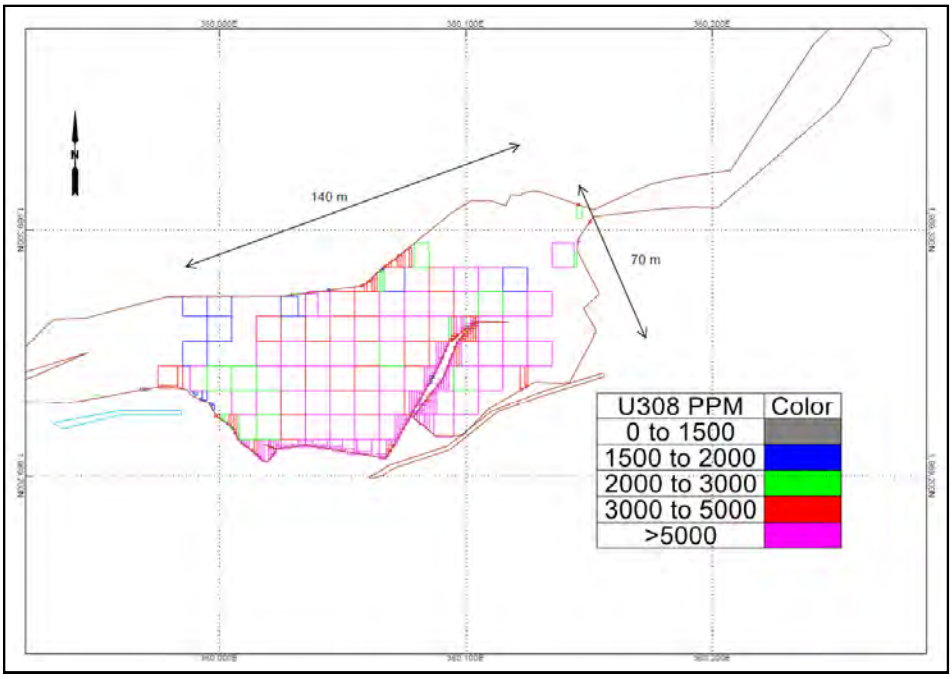


Figure 16-2: Plan View on 260 RL.

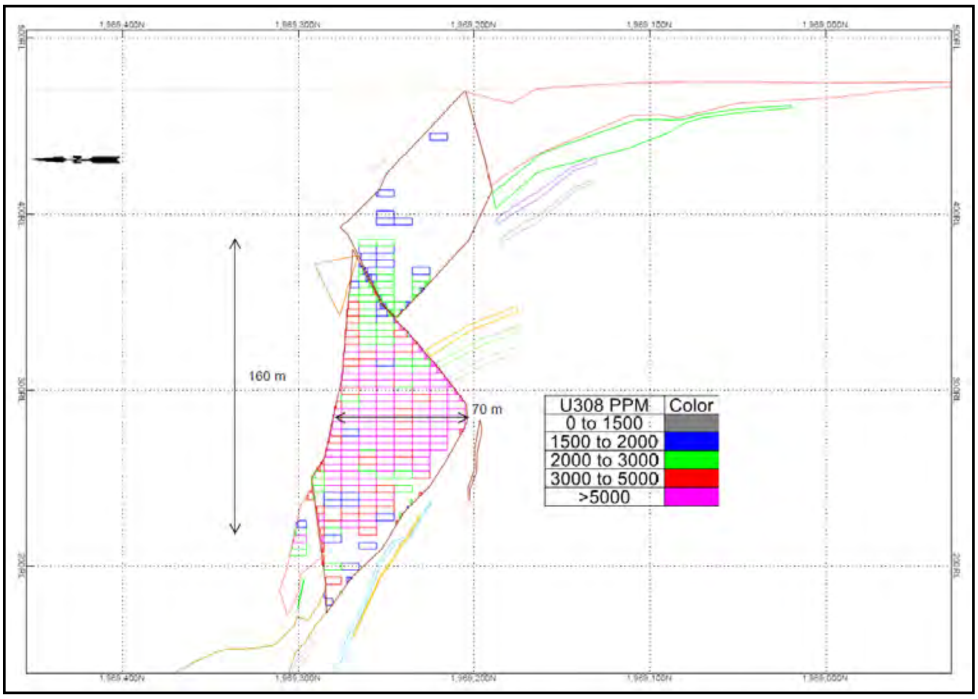


Figure 16-3: Cross Section Through Flank Zone.

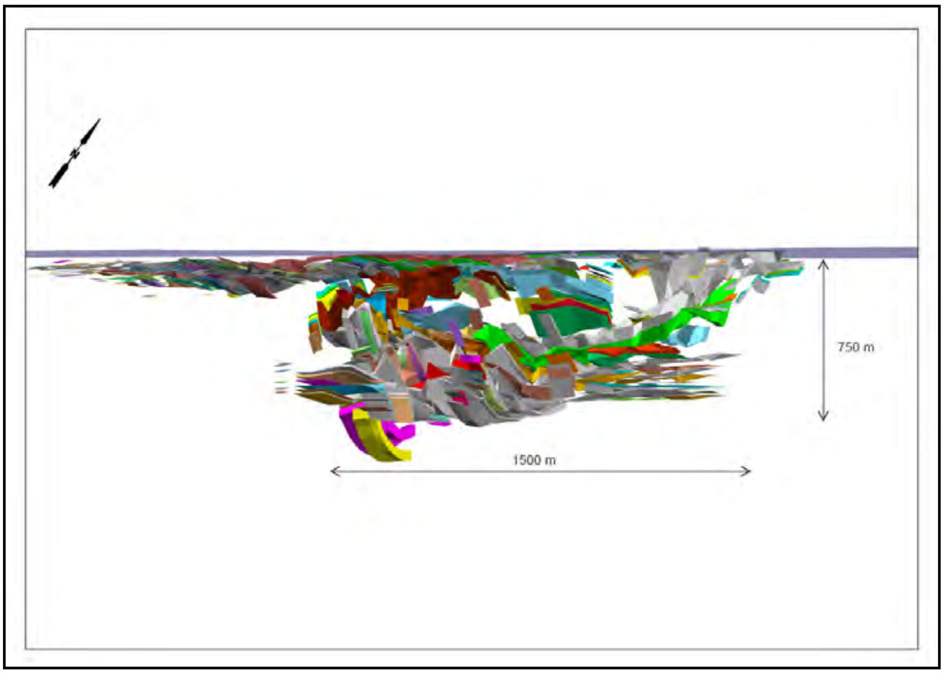


Figure 16-4: 3-D View of Orebody wireframes (Looking North-West)

The geological data was provided to Bara by GAC in the form of orebody wireframes and a grade block model. The orebody wireframes were provided in DXF format in a file named ORE_VER7.dxf which contained layers for each individual wireframe as illustrated in Figure 16-4 above.

The block model was provided in both text and Datamine format in a file entitled Model_Krig_2023.dm. The block model has a parent cell size of 10 m × 10 m × 4 m but contains sub cells of 1 m × 1 m ×1 m. This block size is considered appropriate for mine design considering the bulk nature of the orebody. No further manipulation of the block model was undertaken by Bara for mine design purposes.

16.3. Geotechnical Evaluation

Geotechnical Data Collection

The geotechnical data required for the project was obtained from geotechnical core logging and laboratory strength testing of core. The geotechnical logging data enabled the derivation of rock mass quality parameters such as:

- Rock Quality Designation (RQD).
- Rock Mass Rating (RMR89).
- Geological Strength Index (GSI).
- Tunnelling Quality Index (Q-Index and Q'-Index).
- Stability Numbers (N).
- Discontinuity orientations (Dip and Dip direction).

The data was obtained through the drilling of boreholes. Figure 16-5 below illustrates the locations of the nine main boreholes that were used in the geotechnical evaluation and design work.

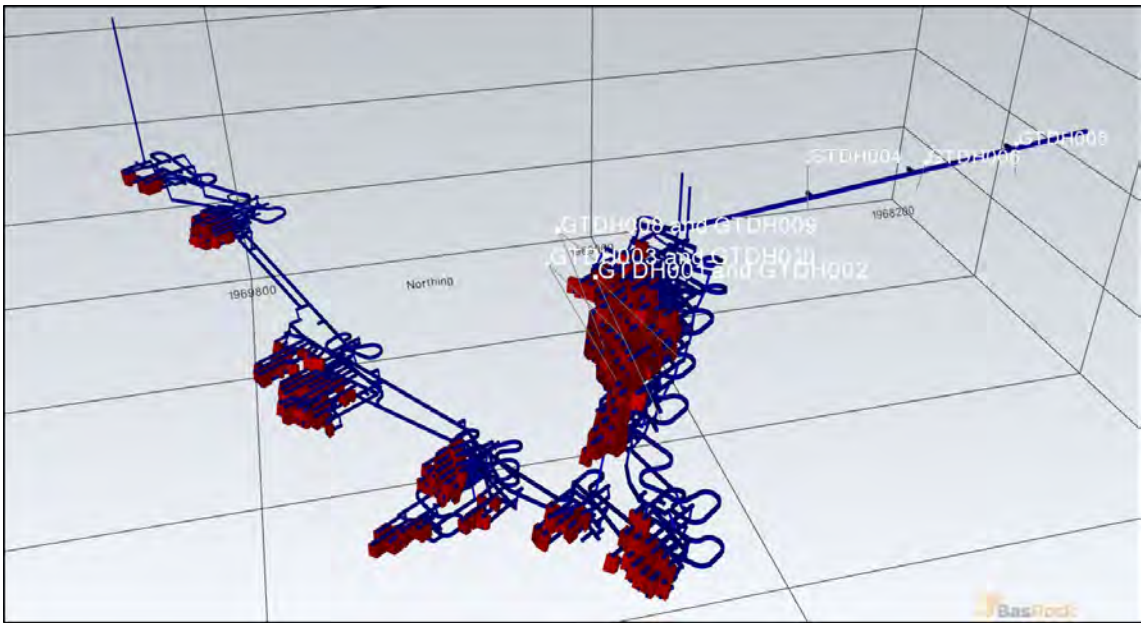


Figure 16-5: Locations of Borehole used in Geotechnical Evaluation and Design Work.

The strength of the materials within the rock mass was determined from the laboratory testing of core, these tests included:

- Uniaxial Compressive Strength with Elastic Moduli (UCM).
- Triaxial Compressive Strength (TCS).
- Splitting Tensile Strength Tests (Brazilian).

- Direct Shear Strength Tests.

The modelling input parameters were determined from analysis of the rock test results together with rock quality parameters by using the Rocscience software, RocLab.

The following rock types were intersected in the project area (based on total length of drill core):

- Sandstones 35.27%.
- Mudstones 30.54%.
- Siltstones 26.7%.
- Analcimolite 2.97%.
- Conglomerate 2.49%.
- Other 2.03%.

A summary of the rock mass qualities obtained through geotechnical logging and laboratory testing are presented in the tables below based on the formations that intersect the proposed mining areas and development. Rock Quality Designation (RQD), Rock Mass Rating (RMR), Rock Tunnel Quality Index (Q), modified Rock Tunnelling Quality Index (Q') and the Mining Rock Mass Rating (MRMR). The mean, minimum, maximum, standard deviation, 25th percentile, 50th percentile and 75th percentile of each rock mass classification system is provided.

An initial conceptual level determination was made for use in the mining method trade off study which was then validated and advanced to feasibility study levels of accuracy through further borehole drilling and rock laboratory testing. The anticipated groundwater conditions and potential influx of water into the underground excavations has resulted in a further refinement of rock mass qualities as provided in Table 16-1 to Table 16-4 below

Table 16-1: Irhazer Formation properties (Hanging Wall).

	RQD (%)	RMR	Q	Q'	MRMR	RQD (%)	RMR	Q	Q'	MRMR	RQD (%)	RMR	Q	Q'	MRMR
	Initial determination					Validation					Validation (Jw = 0.5)				
Mean	54	60	4	13	23	65	63	8	21	32	75	54	5	23	28
Minimum	15	48	1	2	14	31	50	1	3	19	39	41	1	5	16
Maximum	72	72	162	33	38	79	76	129	43	52	89	68	68	48	46
Standard deviation	12	5	12	17	4	10	6	12	21	7	11	7	8	21	7
25^{th} %	47	57	1	5	20	60	59	3	8	27	69	50	2	11	24
50^{th} %	53	59	2	8	22	65	62	5	15	31	75	53	3	17	27
75^{th} %	63	64	3	13	25	73	67	12	32	37	83	59	6	29	32

Table 16-2: Tchirozerine Formation Properties (Ore Zone).

	RQD (%)	RMR	Q	Q'	MRMR	RQD (%)	RMR	Q	Q'	MRMR	RQD (%)	RMR	Q	Q'	MRMR
	Initial determination					Validation					Validation (Jw = 0.5)				
Mean	81	72	25	34	34	86	75	24	39	48	86	66	11	38	42
Minimum	23	54	1	2	14	34	58	1	3	27	34	49	1	3	23
Maximum	98	86	186	210	48	98	88	147	177	64	98	80	72	177	56
Standard deviation	16	7	37	44	8	14	7	31	40	10	14	7	15	39	9
25^{th} %	82	68	4	9	29	86	71	5	14	41	86	63	2	13	36
50^{th} %	86	72	7	15	35	90	75	11	25	50	90	67	5	25	43
75^{th} %	89	78	31	44	40	92	80	31	52	55	92	71	14	52	48

Table 16-3: Teloua Formation Properties (Ore Zone).

	RQD (%)	RMR	Q	Q'	MRMR	RQD (%)	RMR	Q	Q'	MRMR	RQD (%)	RMR	Q	Q'	MRMR
	Initial determination					Validation					Validation (Jw = 0.5)				
Mean	79	73	22	29	36	84	76	22	34	49	84	67	11	37	43
Minimum	29	58	1	2	19	38	60	2	4	27	38	52	1	5	25
Maximum	100	84	243	258	49	100	86	190	212	64	100	77	94	212	56
Standard deviation	18	6	35	40	7	16	6	29	36	9	16	6	14	36	8
25^{th} %	77	68	4	6	32	82	72	7	15	44	82	63	4	16	38
50^{th} %	86	73	9	14	37	90	77	13	26	51	90	68	6	28	44
75^{th} %	89	78	20	28	41	92	80	21	37	55	92	72	11	45	48

Table 16-4: : Dajy Formation Properties (Foot Wall).

	RQD (%)	RMR	Q	Q'	MRMR	RQD (%)	RMR	Q	Q'	MRMR	RQD (%)	RMR	Q	Q'	MRMR
	Initial determination					**Validation**					**Validation (Jw = 0.5)**				
Mean	76	70	23	29	34	82	73	21	32	47	82	65	11	35	41
Minimum	14	51	3	3	16	30	55	2	3	27	30	46	2	3	23
Maximum	110	84	263	291	49	108	87	205	237	64	108	78	102	237	56
Standard deviation	23	8	37	44	7	19	8	31	41	8	19	8	15	42	8
25th %	73	66	5	7	29	80	69	5	8	41	80	60	4	8	36
50th %	85	71	10	14	34	89	74	12	21	48	89	66	6	22	41
75th %	90	76	22	28	40	93	79	24	40	53	93	70	13	47	46

Rock strength tests were conducted on borehole core which were sampled during the geotechnical data gathering phase. A summary of the test results is provided below:

- Fifty-three uniaxial compressive strength tests were undertaken and average values of 39.71 MPa for all rock units, 47.30 MPa for the sandstone, and 19.11 MPa for the mudstone were obtained.
- Twenty-two triaxial compressive strength tests were undertaken TCS results were utilised to determine the Mohr-Coulomb (MC) and Hoek-Brown (HB) parameters for use in numerical analysis programmes. Six failures occurred partially along pre-existing foliations.
- Thirty-six splitting tensile strength tests were undertaken. The average tensile strength value is 3.62 MPa all rock units, 3.83 MPa for the sandstone, and 3.14 MPa for the mudstone.
- Four direct shear strength tests were carried out to determine the maximum resistance that a material can withstand when subjected to shearing. Failure occurred generally smooth and flat pre-existing sliding surfaces.

16.4. Design of Stope Excavations

Stope span designs were undertaken and were based on the critical hydraulic radius for various dip angles of the orebody as well as varied stoped widths. The backfill free standing ability was determined empirically and was based on the maximum stope heights and maximum strike lengths. Four options are provided for the open stopes, summarised as follows, and illustrated below:

- Unsupported, top-down drilling, parallel blast holes, 16.5 m wide drift.
- Cable bolted, top-down drilling, parallel blast holes, 16.5 m wide drift.
- Unsupported, top-down drilling, fan blast holes, 5 m drift.
- Cable bolted, top-down drilling, fan blast holes, 5 m drift.

Figure 16-6 below shows the four options considered for stope design.

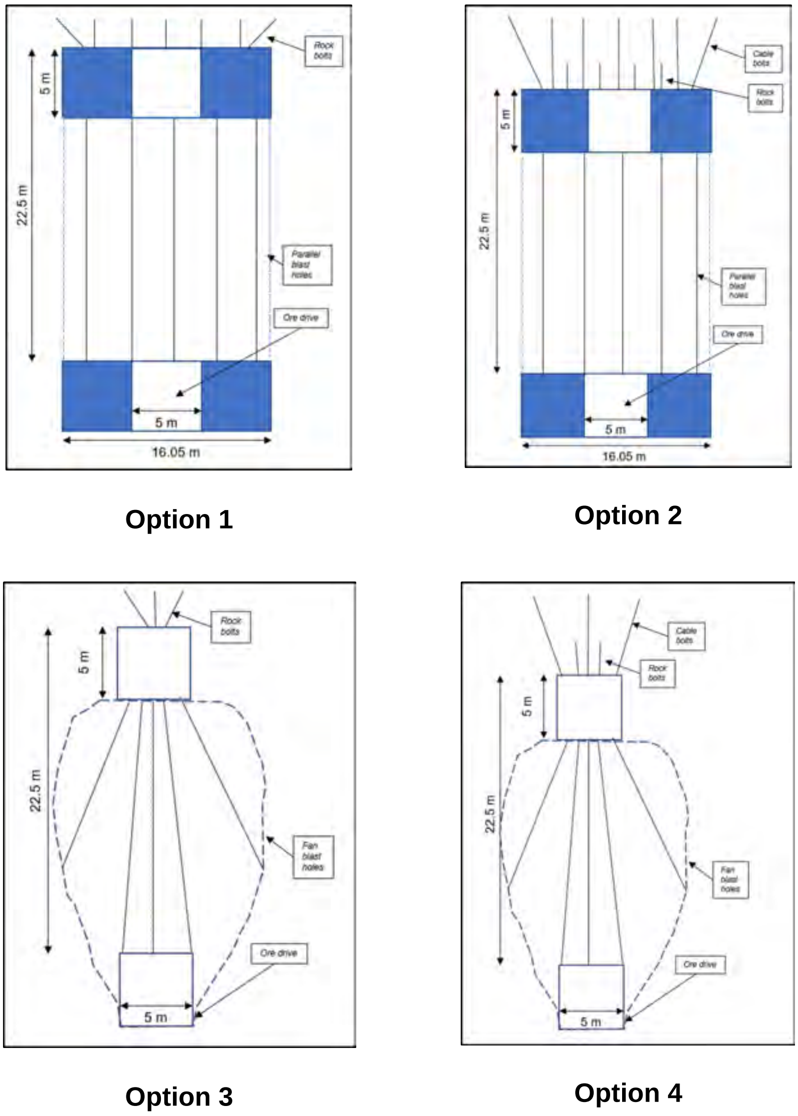


Figure 16-6: Four Options Considered for Open Stope Design.

The Modified Stability Graph Method developed by Matthews et al, 1981, and later modified by Potvin, Nickson, Mawdesley and others, provides an empirical method for estimating the dimensions of open stopes based on the relationship between rock mass quality and mining span. This method involves the calculation of a stability number (N') for each stope wall and relating this to a critical hydraulic radius.

The stability number N' provides an indication of the geotechnical factors that may affect stability. The larger the N' value, the more stable the excavation. The hydraulic radius, HR, is a measure of the effect of size and shape of a stope surface and is proportional to the N'. The results are presented in Table 16-5 and Table 16-6.

As a result of this analysis, stope design Option 4 (cable bolted top sills with down-hole drilling) was selected as the preferred stope design.

Table 16-5: Stability Number and Critical Hydraulic Radii per Stope Wall for Each Option.

ITEM	ZONE	WALL	OPTION 1 UNSUPPORTED, 16.05 m		OPTION 2 SUPPORTED, 16.05 m		OPTION 3 UNSUPPORTED, 5 m		OPTION 4 SUPPORTED, 5 m	
			25th %	MEAN	25th %	MEAN	25th %	MEAN	25th %	MEAN
Q'	1	SW	9.56	22.97	9.56	22.97	9.56	22.97	9.56	22.97
		C	9.56	22.97	9.56	22.97	9.56	22.97	9.56	22.97
		EW	9.56	22.97	9.56	22.97	9.56	22.97	9.56	22.97
	2	SW	7.00	17.00	7.00	17.00	7.00	17.00	7.00	17.00
		C	7.00	17.00	7.00	17.00	7.00	17.00	7.00	17.00
		EW	7.00	17.00	7.00	17.00	7.00	17.00	7.00	17.00
	3, 4 & 5	SW	7.00	17.00	7.00	17.00	7.00	17.00	7.00	17.00
		C	7.00	17.00	7.00	17.00	7.00	17.00	7.00	17.00
		EW	7.00	17.00	7.00	17.00	7.00	17.00	7.00	17.00
STABILITY NUMBER	1	SW	39.77	95.56	39.77	95.56	39.77	95.56	39.77	95.56
		C	1.89	4.55	1.89	4.55	2.29	5.51	2.29	5.51
		EW	3.98	9.56	3.98	9.56	6.12	14.70	6.12	14.70
	2	SW	28.25	68.60	28.25	68.60	29.12	70.72	29.12	70.72
		C	0.42	1.02	0.42	1.02	0.59	1.43	0.59	1.43
		EW	1.12	2.72	1.12	2.72	1.57	3.81	1.57	3.81
	3, 4 & 5	SW	21.84	53.04	21.84	53.04	29.12	70.72	29.12	70.72
		C	0.42	1.02	0.42	1.02	0.42	1.02	0.42	1.02
		EW	1.12	2.72	1.12	2.72	1.12	2.72	1.12	2.72
			POT	MAW	POT	MAW	POT	MAW	POT	MAW
CRITICAL HYDRAULIC RADIUS	1	SW	12.76	22.12	15.20	32.62	12.76	22.12	15.20	32.62
		C	4.15	4.23	8.87	6.28	4.45	4.70	9.18	6.97
		EW	5.46	6.33	10.12	9.38	6.40	8.00	10.92	11.85
	2	SW	11.29	18.47	14.34	27.26	11.42	18.78	14.42	27.71
		C	2.39	1.88	6.81	2.80	2.71	2.25	7.23	3.35
		EW	3.43	3.20	8.10	4.75	3.89	3.84	8.60	5.70
	3, 4 & 5	SW	10.27	16.06	13.70	23.72	11.42	18.78	14.42	27.71
		C	2.39	1.88	6.81	2.80	2.39	1.88	6.81	2.80
		EW	3.43	3.20	8.10	4.75	3.43	3.20	8.10	4.75

Table 16-6: Stope Details for Each Option.

ITEM	ZONE	DIMENSION	CALCULATED			
			OPTION 1 UNSUPPORTED, 16.05 m	OPTION 2 SUPPORTED, 16.05 m	OPTION 3 UNSUPPORTED, 5 m	OPTION 4 SUPPORTED, 5 m
STOPE DIMENSION	1	SPAN (m)	16.05	16.05	5.00	5.00
		HEIGHT (m)	22.50	22.50	22.50	22.50
		LENGTH BEFORE BACKFILL (m)	18.0	120 (limit of ore body)	Limit of ore body	Limit of ore body
	2	SPAN (m)	16.05	16.05	5.00	5.00
		HEIGHT (m)	22.50	22.50	22.50	22.50
		LENGTH BEFORE BACKFILL (m)	8.0	90 (limit of ore body)	Limit of ore body	Limit of ore body
	3, 4 & 5	SPAN (m)	16.05	16.05	5.00	5.00
		HEIGHT (m)	22.50	22.50	22.50	22.50
		LENGTH BEFORE BACKFILL (m)	8.0	90 (limit of ore body)	100 (limit of ore body)	Limit of ore body
CABLE BOLTS	ALL	LENGTH (m)	Not applicable	4 - 6	Not applicable	4 - 6
		SPACING (m)	Not applicable	2 - 2.5	Not applicable	2 - 2.5
ROCK BOLTS	ALL	LENGTH (m)	2.4	2.4	1.5	1.5
		SPACING (m)	2.0	2.0	2.0	2.0
BACKFILL UCS	ALL	HARD CAPPING (kPa)	600	600	600	600
		PRIMARY (kPa)	400	400	400	400
		SECONDARY (kPa)	200 (min 150)	200 (min 150)	200 (min 150)	200 (min 150)
PILLARS	1	RIB PILLAR DESIGN WIDTH (m)	Not applicable	Not applicable	Not applicable	Not applicable
	2		Not applicable	Not applicable	Not applicable	Not applicable
	3, 4 & 5		5	5	5	5
	1	SILL PILLAR DESIGN HEIGHT (m)	8.5	8.5	8.5	8.5
	2		Not applicable	Not applicable	Not applicable	Not applicable
	3, 4 & 5		Not applicable	Not applicable	Not applicable	Not applicable
GLOBAL EXTRACTION RATIO	1	%	91%	91%	91%	91%
	2		100%	100%	100%	100%
	3, 4 & 5		93%	93%	93%	93%

16.5. Pillar Design and Stand Off Distance

Crown pillar thickness was analysed using numerical techniques. A safety factor of 1.8 – 2.0 was utilised to determine the required crown pillar thickness between surface and the underground workings.

Figure 16-7 below illustrates the constructed model modelled in RS2.

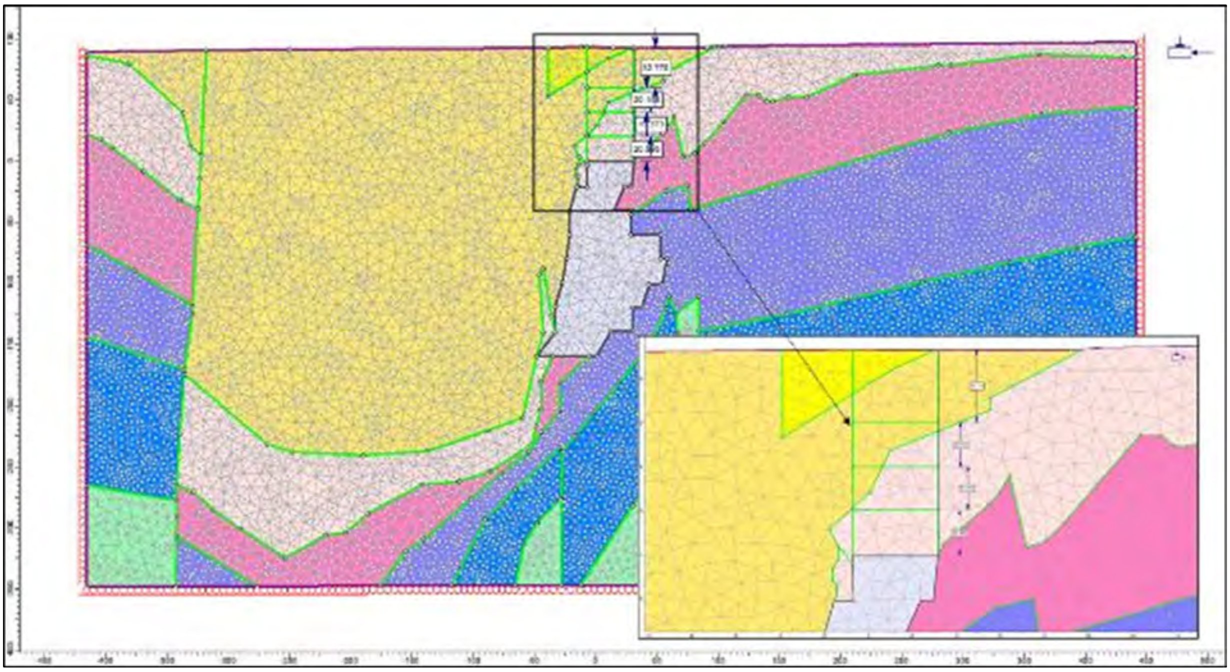


Figure 16-7: Modelling of Crown Pillar Requirements.

The suggested crown thickness of 67.5 m is required as determined by the modelled results.

Sill pillar design was also undertaken where the confinement formula developed by Lunder (1994) for the calculation of pillar strength was used, and the required sill pillar thickness was determined to be 8.5 m for Zone 1 as shown in Figure 16-8 (indicated by yellow circle).

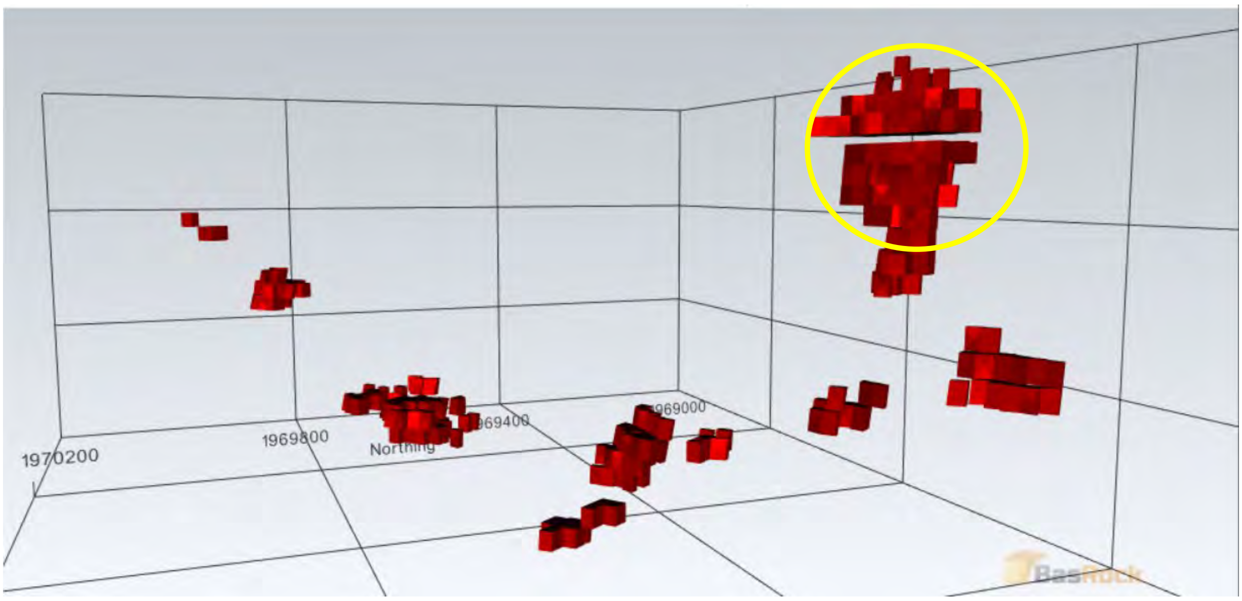


Figure 16-8: Sill Pillar Requirements.

The placement of access development was numerically analysed where the stress effects of the stoping excavations (when backfilled) on the access excavations (footwall drive) were determined. In summary, the safe standoff distances determined through the analysis is as follows:

- Zone 1 = 20 m.
- Zone 2 = 15 m.
- Zone 3, 4 & 5 = 20 m.

16.6. Boxcut Design

The stability of the boxcut slopes have been assessed using the SLIDE slope stability software. The slope configuration at its deepest positions is recommended as illustrated in Figure 16-9 below.

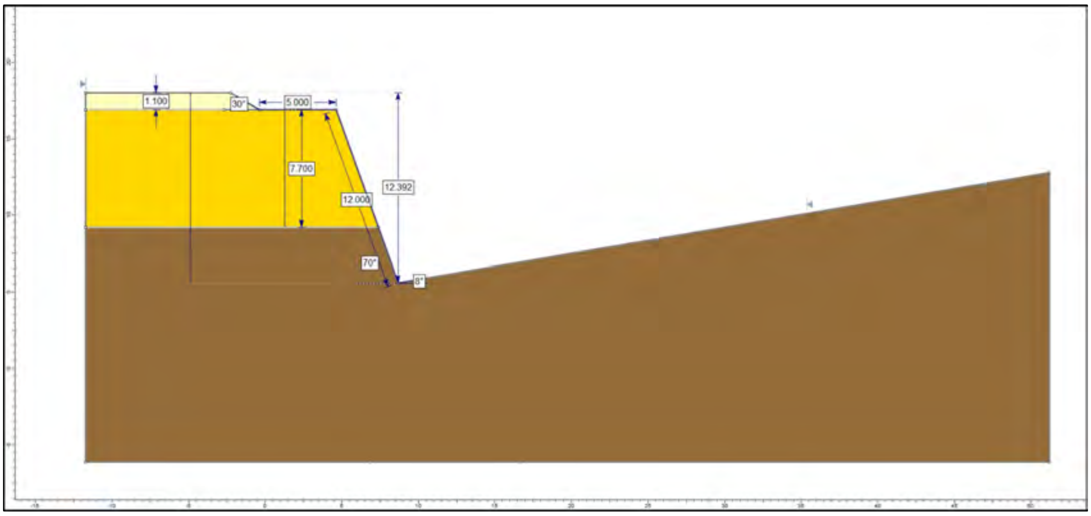


Figure 16-9: : Long Section Through Boxcut.

The support design and support requirements for all major excavations and access developments were calculated based on empirical methods derived by Barton (1974), Unal (1983), Grimstad (1993) and Jager and Ryder (2002). The support requirements for the boxcut are presented in Figure 16-10 and Figure 16-11 below.

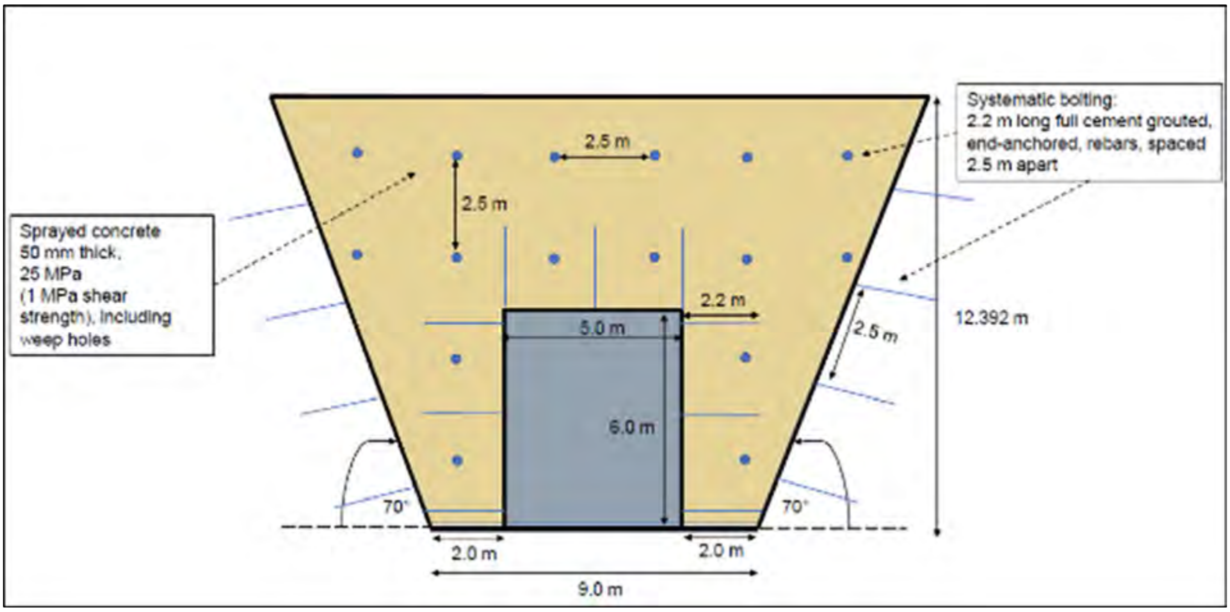


Figure 16-10: Support Requirements for Boxcut Highwall.

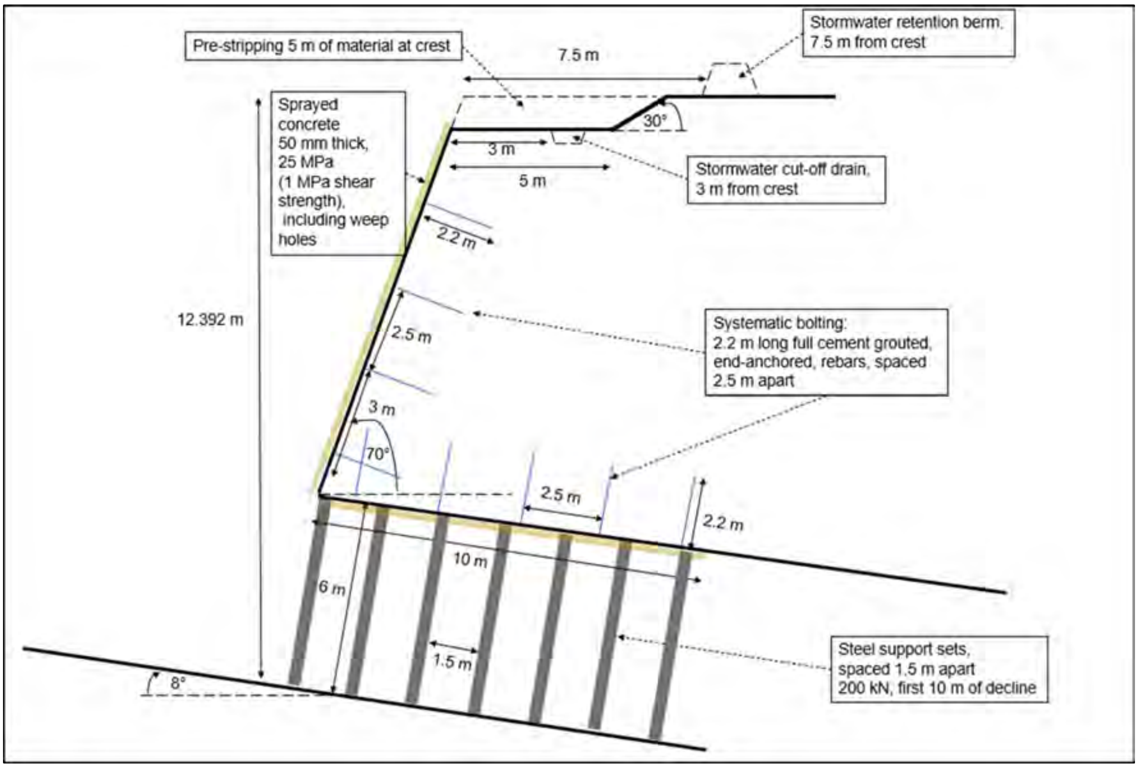


Figure 16-11: Section Through Boxcut Highwall Showing Highwall Support and Portal Area Support Requirements.

16.7. Underground Support Requirements

Support spacing, bolt length and secondary support requirements for all service and development excavations were determined and are detailed below.

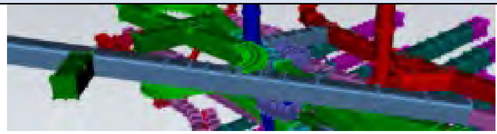
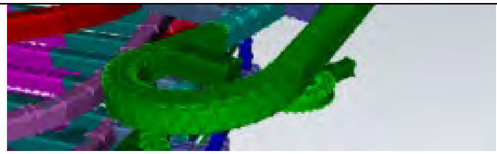
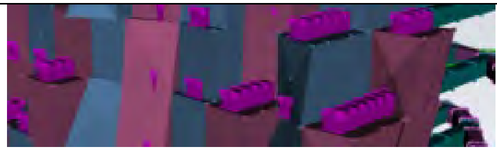
The support system for the large excavations was designed using a combination of empirical methods and design guidelines provided in Hutchinson et al (1996) such as:

- Tunnel support guidelines based on Q-Index case studies developed by Grimstad et al (1993), as an initial assessment of applicable support strategies.
- Theoretical guidelines based on the Q-Index proposed by Barton et al (1974) for primary support requirements.
- Guidelines based on work done by Unal (1983) with respect to RMR and span for secondary support as a result of the size of the excavations.

In addition, empirical guidelines or 'rules of thumb' for the length and spacing of tendons provided by Ryder and Jager (2002) as well as the support of similar excavations on neighbouring mines were utilised as a comparison. Historically, neighbouring mines used support in the form of cable anchors, mesh and shotcrete or thin sprayed liners.

The inputs per excavation type are summarised in Table 16-7and the span/ESR parameter versus Q-Index was plotted on the support guideline chart in Figure 16-12.

Table 16-7: Input Parameters for the Q Index Support Guideline Chart.

Excavation	Width (m)	Height (m)	Area (m^2)	SG	Tonnes/m	Schematic	Q	ESR	Span / ESR
Decline	5.00	6.00	24.80	2.36	58.60		6.72	1.6	3.75
Ramp	4.60	5.40	24.80	2.36	58.60		12.41	1.6	3.38
Level access	4.60	5.10	23.50	2.36	55.40		12.41	1.6	3.19
Footwall drive	4.60	5.10	23.50	2.36	55.40		12.41	1.6	3.19
Stope crosscut - waste	4.50	4.50	20.30	2.36	47.80		8.45	3	1.50
Stope crosscut - ore	4.50	4.50	20.30	2.36	47.80		8.45	3	1.50

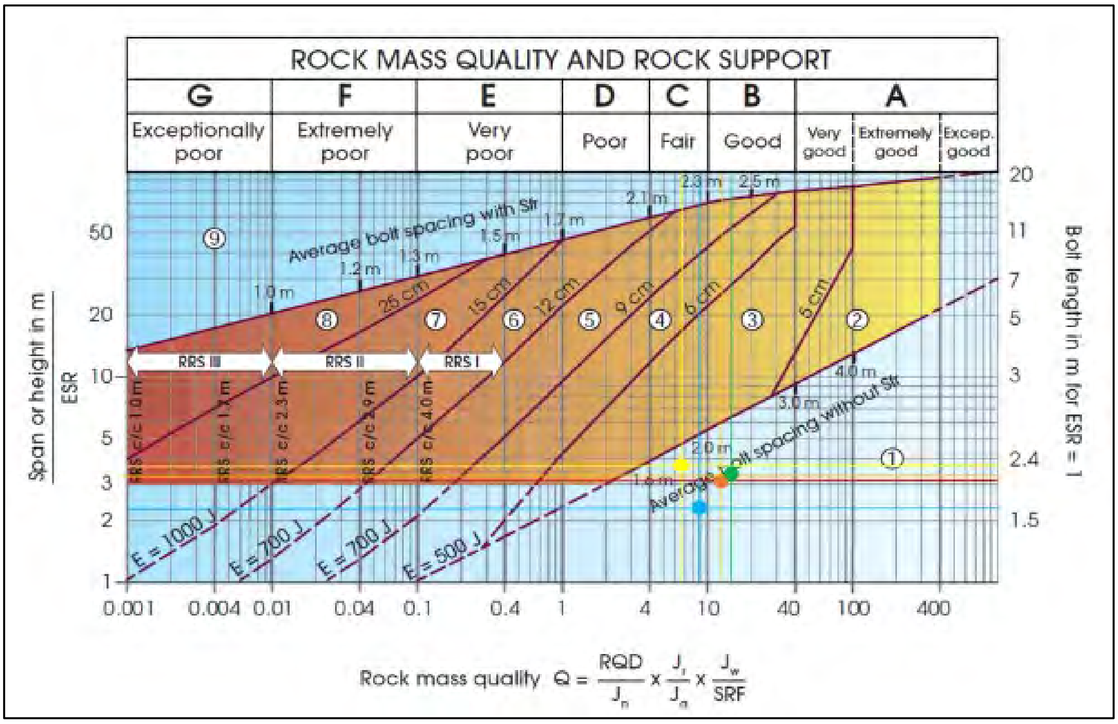


Figure 16-12: Support Guideline Chart.

Support spacing, bolt length and secondary support requirements for all service and development excavations were determined and are detailed in Table 16-8 below.

Table 16-8: Underground Support Requirements by Excavation.

Excavation	Primary support - tendons				Secondary support - anchors/shotcrete				
	Type	Specifications	Spacing	Maximum distance from face	Type	Specifications	Spacing	Application	Maximum distance from face
Decline	Rebar	2.20 m length 20 mm diameter 160 kN tensile strength Full column resin (FASLOC, 300 mm cartridge, 15 sec spin, 45 sec hold, 30 mm diameter, top cartridge point-anchored)	1.4 m x 1.4 m (1.5 m from floor on sidewall)	1 m from face (before blast). After blast 5 m.	Cable anchor	4.5 m length 38 tonne tensile strength 15 - 16 mm diameter Full column grout	3.0 m x 3.0 m	As and when required, and at all intersections	7 m
					Shotcrete	Unreinforced Minimum 25 mm thick	n/a	Estimated at 25% of length of excavation	14 m
Ramp	Rebar	2.20 m length 20 mm diameter 160 kN tensile strength Full column resin (FASLOC, 300 mm cartridge, 15 sec spin, 45 sec hold, 30 mm diameter, top cartridge point-anchored)	1.6 m x 1.6 m (1.5 m from floor on sidewall)		Cable anchor	4.5 m length 38 tonne tensile strength 15 - 16 mm diameter Full column grout	3.0 m x 3.0 m	As and when required, and at all intersections	7 m
					Shotcrete	Unreinforced Minimum 25 mm thick	n/a	Estimated at 25% of length of excavation	14 m
Level access	Rebar	2.20 m length 20 mm diameter 160 kN tensile strength Full column resin (FASLOC, 300 mm cartridge, 15 sec spin, 45 sec hold, 30 mm diameter, top cartridge point-anchored)	1.6 m x 1.6 m (1.5 m from floor on sidewall)		Cable anchor	4.5 m length 38 tonne tensile strength 15 - 16 mm diameter Full column grout	3.0 m x 3.0 m	As and when required, and at all intersections	7 m
					Shotcrete	Unreinforced Minimum 25 mm thick	n/a	Estimated at 25% of length of excavation	14 m
Footwall drive	Rebar	2.20 m length 20 mm diameter 160 kN tensile strength Full column resin (FASLOC, 300 mm cartridge, 15 sec spin, 45 sec hold, 30 mm diameter, top cartridge point-anchored)	1.6 m x 1.6 m (1.5 m from floor on sidewall)		Cable anchor	4.5 m length 38 tonne tensile strength 15 - 16 mm diameter Full column grout	3.0 m x 3.0 m	As and when required, and at all intersections	7 m
					Shotcrete	Unreinforced Minimum 25 mm thick	n/a	Estimated at 25% of length of excavation	14 m
Stope crosscut - waste	Split set	1.80 m length 46 mm diameter 39 - 45 mm hole diameter Ungrouted	1.0 m x 1.0 m (1.5 m from floor on sidewall)		Cable anchor	3.5 m length 38 tonne tensile strength 15 - 16 mm diameter Full column grout	3.0 m x 3.0 m	As and when required, at discretion of the on-site rock engineering department	14 m
Stope crosscut - ore	Split set	1.80 m length 46 mm diameter 39 - 45 mm hole diameter Ungrouted	1.0 m x 1.0 m (1.5 m from floor on sidewall)						

16.8. Raise-bore Shaft Design

In addition to the decline access to the mine, a number of raise bored shafts will be excavated for the purposes of ventilation and as a second means of egress from the mine in case of emergency.

The maximum unsupported span for large-diameter raise-bored shafts using a modification of the normal Q-index has been determined for five ventilation shafts as given in Table 16-9 below.

Table 16-9: : Maximum Unsupported Span for Raise Bored Holes.

Raise-bore Number	Length (m)	Diameter (m)	Inclination (^{0})	Function	Year required	Borehole Number	Borehole Depth (m)	Distance (m)
1	79.5	5.0	90	Fresh air raise	1	GTDH004	70	170
2	81.3	5.0	90	Return air raise	1	GTDH004	70	170
3	120.2	4.5	70.8	Fresh air raise	5	ASDH569	730	190
4	102.4	4.5	62.7	Return air raise	5	ASDH569	730	190
5	288.7	5.0	89.1	Fresh air raise	7	ASDH385	387	110

The maximum safe unsupported span with depth is illustrated in Figure 16-13 to Figure 16-15 where intervals of instability should be anticipated. For surface raises RB No. 1 and No. 2, design diameters of 5 m and 3. 5 m were analysed, and it was determined that approximately 20 m of support would be required for the 5 m diameter shaft.

For raises RB No. 3 and No. 4, a design diameter of 4. 5 m was analysed, and it was determined no support would be required for the shaft.

For raises RB No. 5, a design diameter of 5 m was analysed, and it was determined that approximately 45 m of support would be required for the shaft.

It is recommended that at least one borehole be drilled at the locations and along the orientations of each raise-bore to undertake a full ventilation shaft analysis.

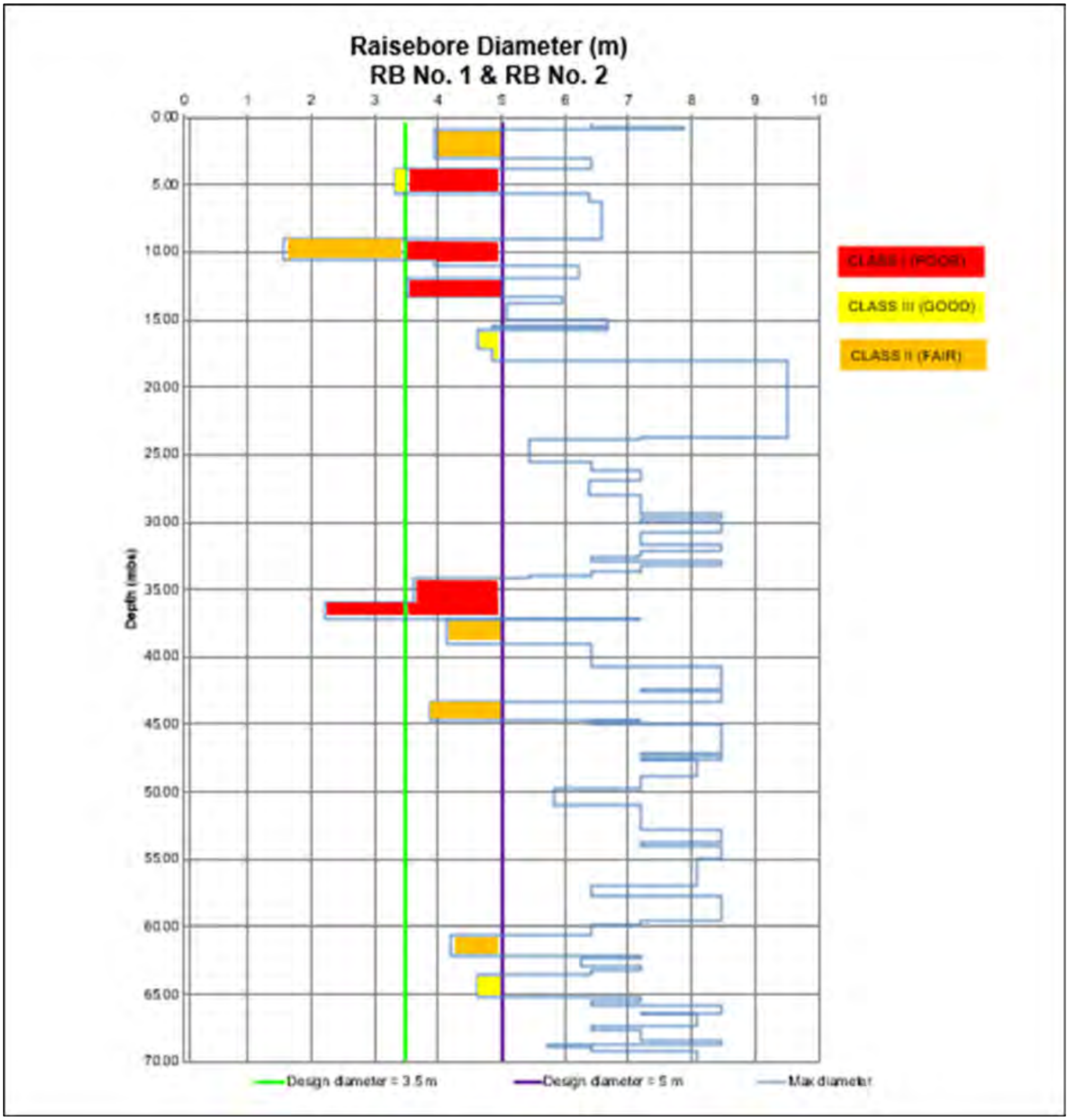


Figure 16-13: Variation of Maximum Diameter with Depth for RB1 and RB2.

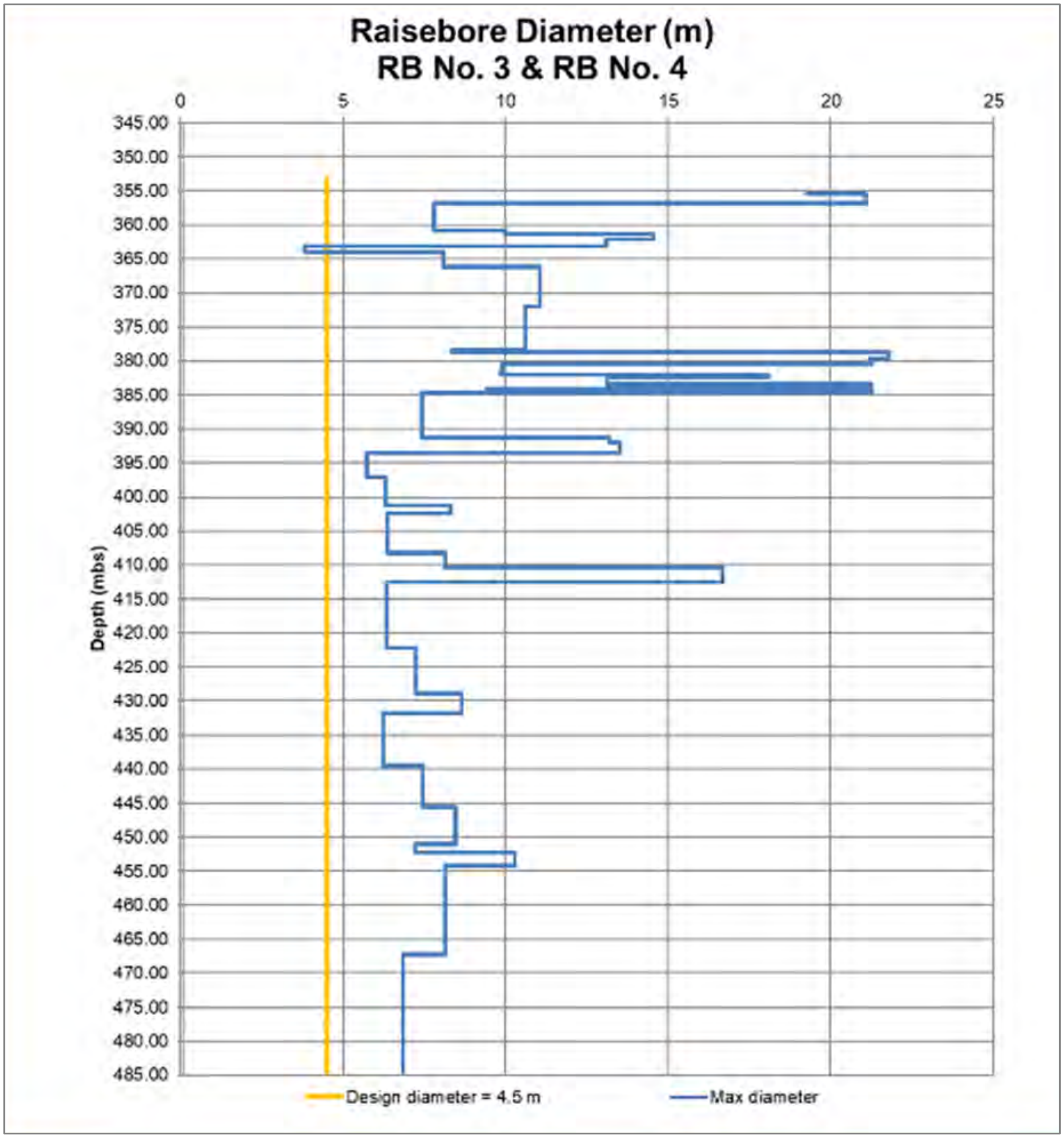


Figure 16-14: Variation of Maximum Diameter with Depth for RB3 and RB4.

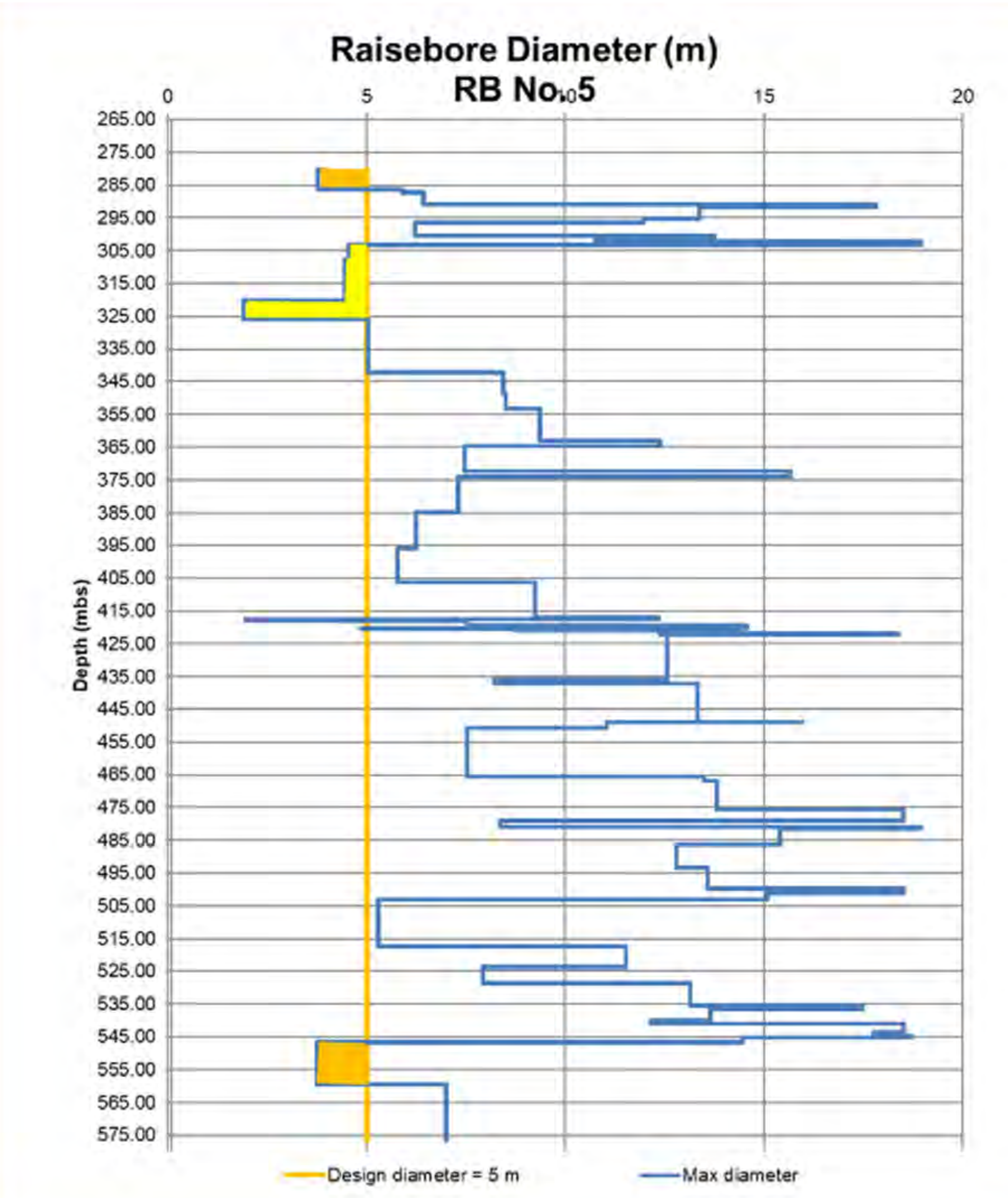


Figure 16-15: Variation of Maximum Diameter with Depth for RB5.

16.9. Mine Stability Analysis

On completion of the mine layout, an analysis of the stability of the entire mine system was undertaken.

The mining layout was digitised into the MAP3D Software modelling programme to analyse stress distributions and displacements because of mining. The layout was sequenced according to the excavation year spanning from year 2 up to year 14. The outcomes from the modelling results included the Major Principal Stress (σ1) Horizontal Displacement (δx) Vertical Displacement (δz) Horizontal Strain (εx) and Rockwall Condition Factor (RCF). Figure 16-16 below illustrates the layout and naming of the individual stoping zones considered during the modelling process.

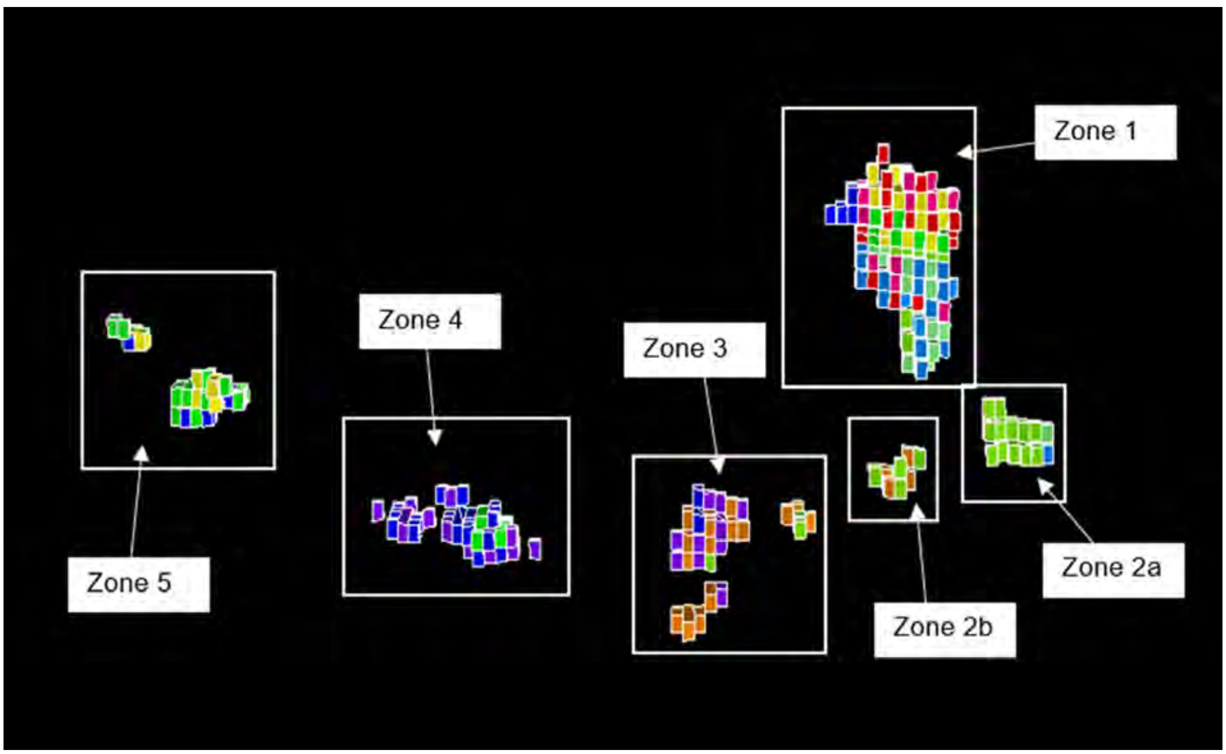


Figure 16-16: Mine Layout and Stoping Zones Considered in Modelling Process.

The outcome of the mine stability analysis was the following:

- Major Principal Stress (σ1): The rock wall conditions are likely to be fractured due to stress at Zone 3 for mining steps 10 to 15.
- Horizontal Displacement (δx): The threshold limit value that has been established for surface tilt will not be a concern for the layout provided.
- Vertical Displacement (δz): Vertical Displacement on surface is expected to be minimal, and the permanent surface infrastructure will not be influenced by the Vertical Displacement that will occur vertically above Zone 1, since it is off set at least 530 m away. The Vertical Displacement in the area around the ventilation shaft

collars was measured at 8 mm. This means that the collar construction of the ventilation shafts must be designed to accommodate approximately 8 mm of Vertical32.

- Displacement. None of the values determined for the crown pillars exceeded the threshold limit value, indicating that stope crowns will remain stable as designed.
- Horizontal Strain (εx): The maximum Horizontal Strain was determined to be -100 × 10-6, which is significantly less than the threshold for damage on surface.
- Rockwall Condition Factor (RCF): The results revealed that for most of the mine development infrastructure the RCF is below 0.7 throughout all the mining steps, which indicates good ground conditions. There are areas at Zones 3 and 4 that have an RCF of between 0.7 and 1.05 from mining step 10 up to mining step 15. The zone of increased RCF will affect the ramp, level access, footwall drive and stope crosscuts, meaning that these access tunnels will require more specialised support than the rest of the mine. The RCF surrounding Zone 2a and 2b was between 0.7 and 1.05 from mining step 9 up to mining step 15. The zone of increased RCF will affect only the stope crosscuts. For Zone 5, an RCF of between 0.7 and 1.05 was determined from mining step 13 up to mining step 15. The zone of increased RCF will affect only the stope crosscuts. Most of the areas identified with an increased RCF only occurred after step 9, which includes mining out of the sill pillar in Zone 1. This again indicates the effect of mining out the sill pillar, and further justifies more detailed assessment on the mining sequence and extraction percentage of the sill pillar in Zone 1.

16.10. Geotechnical Risk Analysis

A risk assessment for the geotechnical analysis of the orebody and the geometrical design of Dasa had been performed. A standard hazard identification and risk assessment process where hazards were identified, and risk assessed using a matrix highlighting the impact of the hazards and required mitigation measures to reduce the risk where required. The detailed risk assessment forms part of the main report, the major risks identified are listed below:

General

Insufficient geotechnical data available - Limited data included in analyses for Zones 2 – 5. Zone 1 has been validated trough additional rock testing and borehole data and optimisation potential exists.

Data Collection

Borehole core logging not undertaken to acceptable standards - Conservative design decisions were undertaken to allow for inaccurate data collection.

Data collection points centred around one stoping area – Collection of additional data sets around deeper stoping and development areas.

Geotechnical Characterisation

Defect orientations have not been acquired – No joint orientation data was available for design. Alpha angles analysed as indication of risk, however oriented logging or ATV surveys are required to properly quantify kinematic risk to design.

Geotechnical Design Aspects

Development in extensive quantities of weathered material - deep seated weathering along the faults (including decline, Boxcut) - Recommended that additional data be included in analyses and design/additional boreholes be drilled for rest of stoping zones.

Lack of information used in ventilation shaft design – Recommended to drill at least one borehole at east proposed shaft location and along its proposed orientation.

Boxcut slope design - Additional drilling required to confirm structural complexity of boxcut area so that adequate kinematic analysis can be undertaken.

16.11. Mining Method Selection

The formal evaluation of suitable candidate mining methods and the motivation for the selection of the optimal method to be employed in the FS were therefore included in the Bara scope of work. A trade-off study was conducted by Bara which considered a number of alternative mining methods.

Apart from the LHOS with backfill, the document specifically mentioned Sub-level cave (SLC) and cut and fill (C&F) mining as potential alternatives, these methods are included in the following discussion.

The geometry of the orebody lends itself to mechanised bulk mining methods. Therefore, any method considered will be based on utilising these efficient mining methods in order to take advantage of the economy of scale factor.

Bulk mining methods can be split into two general types being short hole mining and long hole mining. Short hole mining employs drifting or development type drill rigs and will result in smaller more frequent blasts while long hole mining employs a long hole drilling rig and will result in fewer and larger blasts. Bulk mining methods in these categories include:

Short Hole Methods

- Cut and fill mining.
- Drift and fill mining.

Long Hole Methods

- LHOS with backfill (scoping study method).
- Sublevel caving.
- LHOS with pillars.

It is considered that the long hole mining methods are generally more efficient and productive than the short hole methods and where possible these methods would be employed ahead of short hole mining methods. At Dasa it is considered that long hole methods are possible (based on scoping study and geotechnical considerations described in Section 16.3) and as such only long hole methods were considered in the trade off study.

There are various types of backfill that are possible in LHOS with backfill, the scoping study has assumed a cemented tailings backfill. It is also possible to use uncemented tailings backfill and waste rock as a backfill medium in certain cases.

The main difference between the LHOS with cemented fill and LHOS with uncemented fill is that the use of cemented fill allows adjacent stopes to be mined without a rib pillar between them. If uncemented fill is used a rib pillar must be left between adjacent stopes in order to contain the fill. Mining can only be conducted in a bottom-up sequence when using uncemented fill and a sill pillar must be left at the top of each stope lift (panel).

The use of cemented fill allows for mining in both bottom-up or top-down sequence and for adjacent stopes to be mined without a pillar separating them. If a sufficiently strong cemented fill is used mining can be conducted in a top-down sequence providing the advantage of mining flexibility.

The use of cemented and uncemented tailings backfill will be possible at Dasa and should be considered.

Waste rockfill was not considered, based on analysis of the volume of waste rock produced versus the volume of ore produced (as reported in the PEA). It is unlikely that this ratio will change significantly with the revised FS mine design and it is clear that insufficient waste rock will be generated to adequately fill the mined-out stope voids. Notwithstanding this it may be desirable to tip waste rock into the stopes for disposal prior to being backfilled with a tailing based backfill medium.

Work on backfill has indicated that paste fill cannot be produced using the Dasa tailings and the geotechnical recommendation is not to consider SLC, these two methods were discounted resulting in the following options being considered:

- Transverse LHOS with uncemented hydraulic fill (HF).
- Longitudinal LHOS with uncemented hydraulic fill (HF).
- Transverse LHOS with cemented hydraulic fill (CHF).
- Longitudinal LHOS with cemented hydraulic fill (CHF).
- Transverse LHOS with pillars (No backfill).
- Longitudinal LHOS with pillars (No backfill).

In comparing the mining method options, the following criteria were considered:

Stope productivity - Productivity calculations were completed on six candidate mining methods to determine if there was a difference in the productivity between methods. The productivity estimates were based on some common inputs.

Sequencing and scheduling – The effect of the mining and backfilling method on the mine sequencing and scheduling was evaluated.

Production rate – The ability of the mining method to support the required production rate was evaluated.

Dilution and RoM grade - In order to determine the differences in grade resulting from each of the mining methods, dilution calculations were conducted on each method, based on some assumptions for the amount of dilution that can be expected.

Operating Cost and Capital Cost

Cashflow – a cashflow model was produced for each mining method taking into consideration the capital cost, operating cost, and scheduling assumptions for each method.

In addition, a qualitative analysis was undertaken to consider non-quantifiable differences between the methods.

This trade-off study showed that, considering NPV as the selection criteria, the options of using cemented hydraulic fill are the most attractive. This is mainly due to the additional mining inventory (higher extraction ratio). Despite the increased operating cost of these two methods, they produce the highest NPV by a significant margin.

The difference in NPV between transverse or longitudinal orientation of the LHOS layout with CHF is less than 5%. At this level of study these NPVs can be considered equal and the decision between the two methods may need to be made on more qualitative issues.

The qualitative strengths and weaknesses of the candidate methods were listed. The strengths of transverse LHOS with CF compared to the longitudinal option are:

- Mining flexibility and additional stopes per level.
- Less congested working arrangement on the level.

The weaknesses are that the operating cost is marginally higher in transverse than in the longitudinal option mainly due to the requirement for additional waste development.

Bara is of the opinion that either transverse or longitudinal LHOS with cemented hydraulic fill are appropriate for Dasa but that the advantages inherent in the transverse layout option of mining flexibility and less congested working environment are important and outweigh the marginal cost saving which may be achieved opting for a longitudinal layout. Bara therefore recommended that transverse LHOS with cemented hydraulic fill be selected as the option for further study in the FS.

16.12. Mining Method Description

The mining method selected for the FS is transverse long hole open stoping with cemented backfill. Access to the orebody will be via a trackless ramp from surface.

The mining method selected, based on the findings of the mining method trade-off study described above, is fully mechanised transverse long hole stoping method with cemented backfilling of the mined-out voids, mining will be from the bottom of a stope block upwards. (LHOS with backfill).

Access to the stoping areas is from the strike development via stope crosscuts. The stope crosscuts are spaced approximately 15 m along strike, these stope crosscuts will traverse the entire width of the mineralised zone (approximately 70 m). This is repeated every 22.5 m vertically, which is the sub-level spacing selected.

A slot raise is developed between two sub-levels to create a free face for blasting and the stope is then drilled with a long hole drilling rig between the sub-levels. Drilling can take place from either, the upper of lower ore drive. The holes are then charged and blasted with the broken ore being loaded out of the lower ore drive. The stope is mined in retreat from the limit of the stope crosscut back to the stope limit adjacent to the strike drive. This LHOS layout is referred to, as a transverse long hole stoping approach.

Once the stope is mined out it will be filled with cemented backfill. Stopes will be mined in a primary secondary sequence to allow time for backfill to cure. The mining sequence will be bottom up in each mining block identified. The stoping sequence is illustrated in Figure 16-17 below.

5	8	**5**	8	**5**
4	7	**4**	7	**4**
3	5	**3**	5	**3**
2	4	**2**	4	**2**
1	3	**1**	3	**1**
Sill pillar				

Figure 16-17: Stoping Sequence.

The placement of sill pillars is prescribed by the geotechnical study. A single sill pillar has been designed which is 10 m thick and is located between the elevations of 277.5 and 267.5 m maximum above mean sea level (mamsl) or 5277.5 and 5267.5 (Mine grid elevation).

This sill pillar allows stoping to commence in a bottom-up sequence from the 5277.5 m elevation, which facilitates an earlier start of stoping than if the stoping commenced at the bottom of the orebody. It is anticipated that more than 50% of this sill pillar will be recovered.

16.13. Mine design

Geotechnical Design Criteria

The criteria for stope design, based on the geotechnical study are summarised in the Table 16-10 below.

Table 16-10: Geotechnical Design Criteria.

Item	Unit	Value
Maximum stope span	metres	16.5
Sublevel spacing	metres	22.5
Maximum length of stope	metres	90
Minimum backfill strength (vertical exposure)	kPa	400
Minimum stand-off distance between footwall drive and orebody	metres	20

Mining Design Criteria

A set of general mine design criteria was developed, based on the geotechnical guidelines detailed in Table 16-10 above. The design criteria are summarised in Table 16-11 below.

Table 16-11: Mine Design Criteria.

Main Access and Transport	Trackless decline, trucking, rubber tyre vehicles
Level Access and Transport	Footwall drives with truck and LHD access
Stoping Method	Transverse long hole open stoping with cemented backfill.
Modifying Factors.	
Stope width.	5 m to 16 m
In-stope dilution.	10% estimated.
Explosives.	Bulk emulsion, ANFO
Development Max Advance Rate (m/month)	
Decline.	75
Level waste development.	60
Drop raises.	24
Ore development.	50
Advance rate per crew.	3.7 m per shift, 223 m per month.

Stope Productivity and Design	
Shift System.	7-day week, 2 shifts per day
Ring Burden.	200 cm
Hole spacing.	150 cm
Stope Drilling.	Electrohydraulic Production rig
Mining Drive Dimensions	4.5 m (w) x 4.5 m (h).
Face cleaning	LHD onto dump truck
Support	Cemented hydraulic tailings backfill.
Pillar size (Sill)	10 m thick sill pillar below main levels.
Pillar size (Rib)	None, primary and secondary stopes, backfilled
Engineering Infrastructure	
Water Consumption.	0.5 tonne water per tonne rock
Fissure water.	21 Ml per day (max)
In situ ore density kg/m³.	2.36
Broken ore density kg/m³.	1.48
In situ waste density kg/m³.	2.36
Broken waste density kg/m³.	1.48
Development Dimensions	
Access Ramp.	
Main Decline (broken).	4.6 m (w) x 5.4 m (h)
Roadway depth.	0.3 m
Muck bay frequency.	75 m
Decline angle.	-8° (1 in 7.2)
Level Development	
Access crosscut.	4.6 m (w) × 5.1 m (h)
Strike footwall drive.	4.6 m (w) × 5.1 m (h)
Stope crosscut.	4.5 m (w) × 4.5 m (h)
Equipment Specification	
Trackless.	
LHD (Development).	Atlas Copco ST14 (14T)
LHD (Stoping).	Atlas Copco ST14 (14T)
Drilling (development).	Atlas Copco Boomer S2 (twin boom development jumbo)
Drill rig stope.	Atlas Copco Simba S7, or equivalent
Dump truck.	Atlas Copco MT42 or equivalent
Personnel carriers.	None

Service vehicles/explosives vehicle.	Aardrunner UV83, or equivalent
Ventilation Total air quantity required. In-take airways. Exhaust shaft/s.	 400 m^3/s (Ventilation study). Decline (24m^2) Downcast ventilation shaft (5.0 m diameter) Upcast ventilation shaft (5.0 m diameter)

Mining Modifying Factors

Table 16-12 below shows the mining modifying factors proposed for use in the feasibility study mining plan.

Table 16-12: Modifying Factors.

Factor	Unit	Value	Comment
Minimum mining width	m	4.5	Minimum width for LHD
Pillar loss	%	2	60 % extraction applied to sill pillar recovery
Mining dilution	%	10	Zero Grade
Ore recovery (from mining)	%	95	
Cut-off grade	ppm	1500	Applied in MSO for stope optimisation

Primary Access

The primary access to the underground workings is via a single decline, developed from surface. The decline facilitates truck hauling of rock, transport of personnel and material in and out of the mine as well as supplement the intake ventilation capacity. The boxcut was constructed in 2022, with the first portal blast in November 2022, and has been followed by ramp development of the mine.

To establish a safe mining face in hard rock to start the decline development a boxcut was excavated. The boxcut will be 13 m deep and 31 m wide at the high wall position with an inclination of minus 8 degrees. The ramp roadway is 5 metres wide and approximately 105 metres long, the roadway surface is of a typical haul road design, with cast concrete drains running down either side of the boxcut excavation to interface with the cut-off drain that is positioned at the base of the highwall to direct water towards the collection sump. The top of the boxcut area has a water retaining berm, approximately 3 metres wide at the base and 1 metre high, to lead the flow of excess run-off water away from the boxcut. Figure 16-18 and Figure 16-19 show a plan and a long section view of the boxcut and portal excavations.

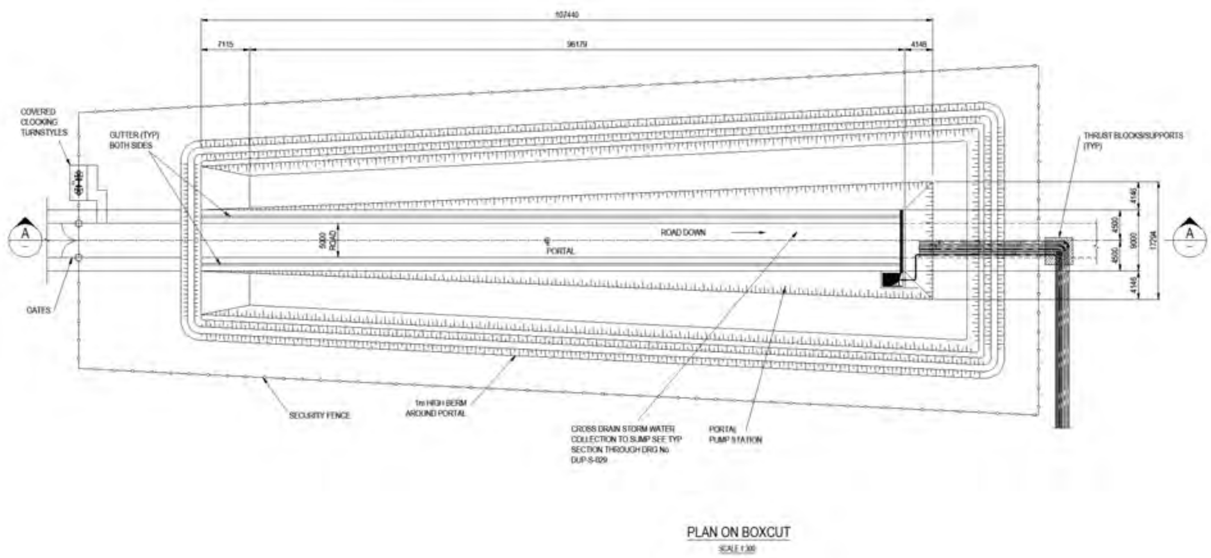


Figure 16-18: Plan View of the Boxcut.

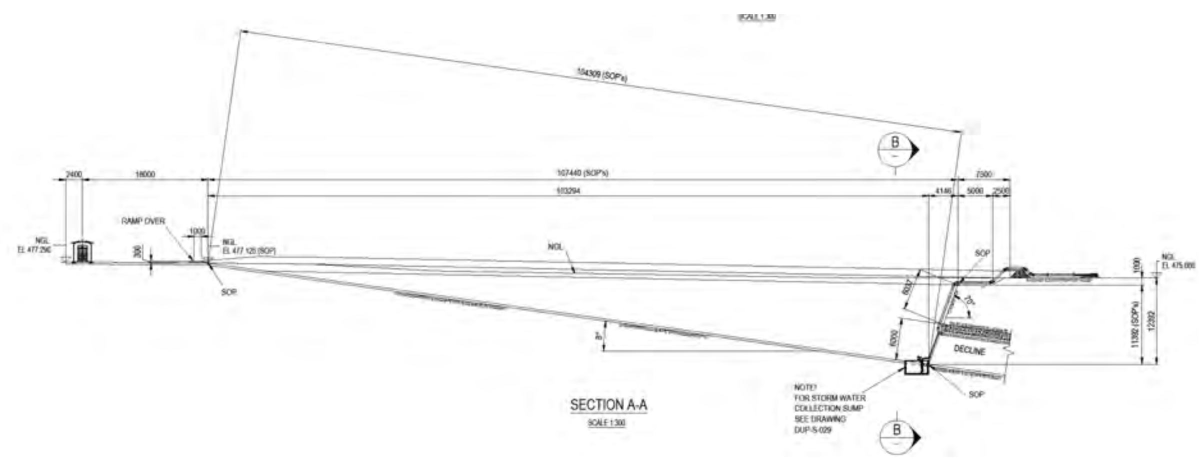


Figure 16-19: : Cross Section of the Boxcut and Portal Excavation.

The portal area and decline are sized to accommodate the trucks selected for ore and waste haulage, the selection of which is discussed in 0. The finished dimensions of the portal and decline are 4.6 m high × 5.1 m wide as illustrated in Figure 16-20. To allow for a road base 30 cm thick, the excavation was blasted at 4.6 m wide × 5.4 m high. The decline is developed at the same inclination as the boxcut roadway, an inclination of minus 8 degrees or 1 in 7.1 (14%).

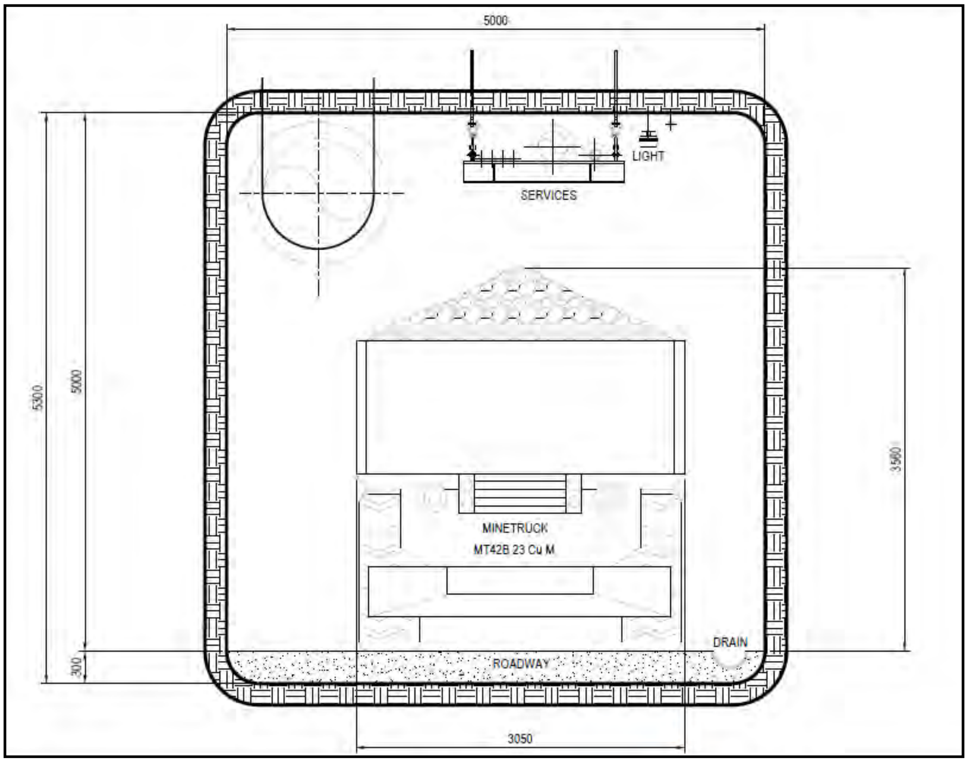


Figure 16-20: Cross Section Through Decline.

The portal entrance hanging wall is supported by fabricated steel sets, together with the recommended support requirement as specified by the geotechnical engineer. The undercut is supported by the first set, which will be installed approximately 0.5 metres from a suitable point just outside the portal entrance. The piping and electrical reticulation services are planned have been installed on a chain suspended steel bracket arrangement, pinned to the highwall. A steel reinforced concrete support arrangement is positioned at the top of the highwall and designed to carry the loads induced by the services. The piping and electrical reticulation services are supported from the underside of the steel sets and carried into the decline, on a typical hanger bracket system.

The highwall will be supported with a combination of anchors, mesh and shotcrete, and the immediate entrance will be supported with spile bars and shotcrete, as required.

The total length of the boxcut and decline system from surface to the lowest level of the mine is 4495 m. Figure 16-21below shows a plan view and vertical projection of the boxcut and decline system from the mine design software used to layout the mine.

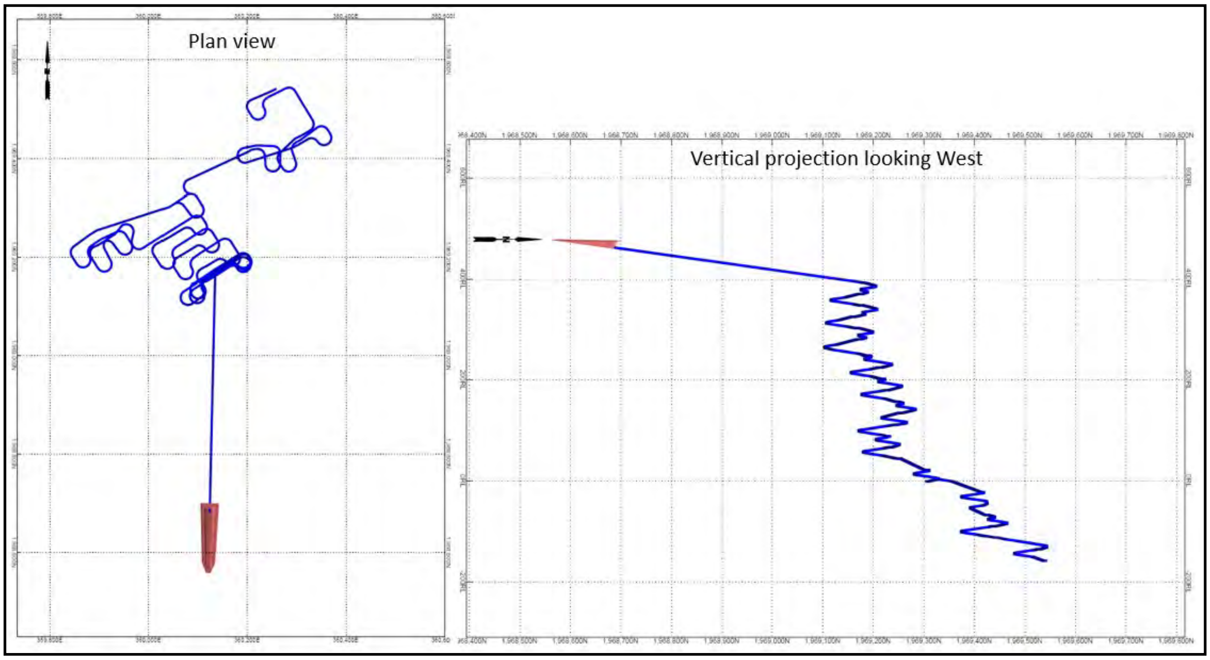


Figure 16-21: Plan View and Vertical Projection of Decline.

16.14. Development

The decline is located in the footwall of the orebody, at a minimum of 60 m from the footwall contact, to allow for the selected stand-off distance between the stopes and the footwall drive and adequate middling between the footwall drive, cubbies and the decline.

Access to the orebody will be established every 22.5 m vertically, by means of a level access crosscut. The level crosscuts will also be excavated at 4.6 m × 5.1 m. Between the decline and the footwall drive (strike drive) breakaway, a stub drive of 15 m length will be established. This will initially be used as muck-bays. These will later be used to house infrastructure such as pump stations, dams, and electrical switchgear.

Three sets of ventilation raise’s will be established linking the levels - an intake ventilation raise established to the centre of the decline system (further into the footwall) and two sets of return ventilation raises, one at either end of the orebody.

The intake ventilation raises will also serve as service raises and will contain all service reticulation including:

- Pump columns.
- Service water columns.
- Electrical cables.
- Compressed air columns.

The raises will also be equipped with a ladderway which will allow for installation and servicing of the services but will also allow the raises to be used as emergency escapeways from the underground workings, should the trucking ramp become compromised.

Stope crosscuts will be developed from the footwall drive at centre spacings of 16.5 m and will be developed perpendicular to the strike direction intersecting the footwall contact of the orebody first but then advancing through the orebody to the hanging wall contact.

Figure 16-22 illustrates a vertical projection of the development layout, while Figure 16-23 shows a plan view of a typical production level.

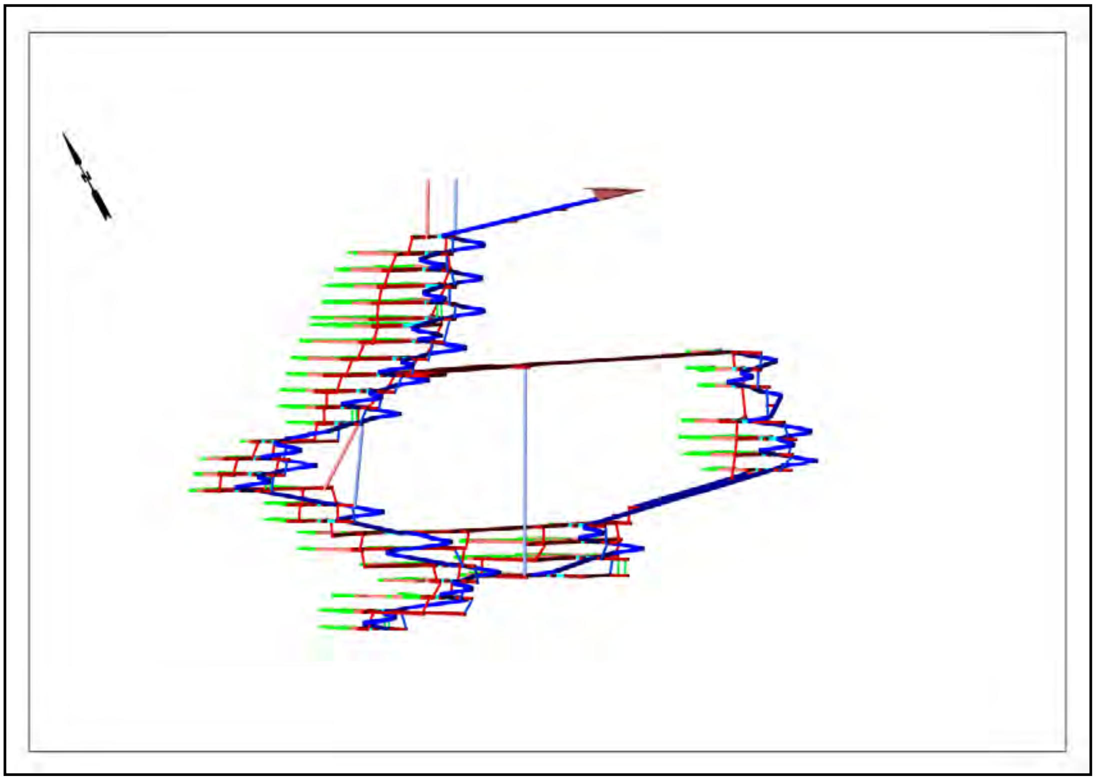


Figure 16-22: Isometric View of Development Layout Looking Northeast.

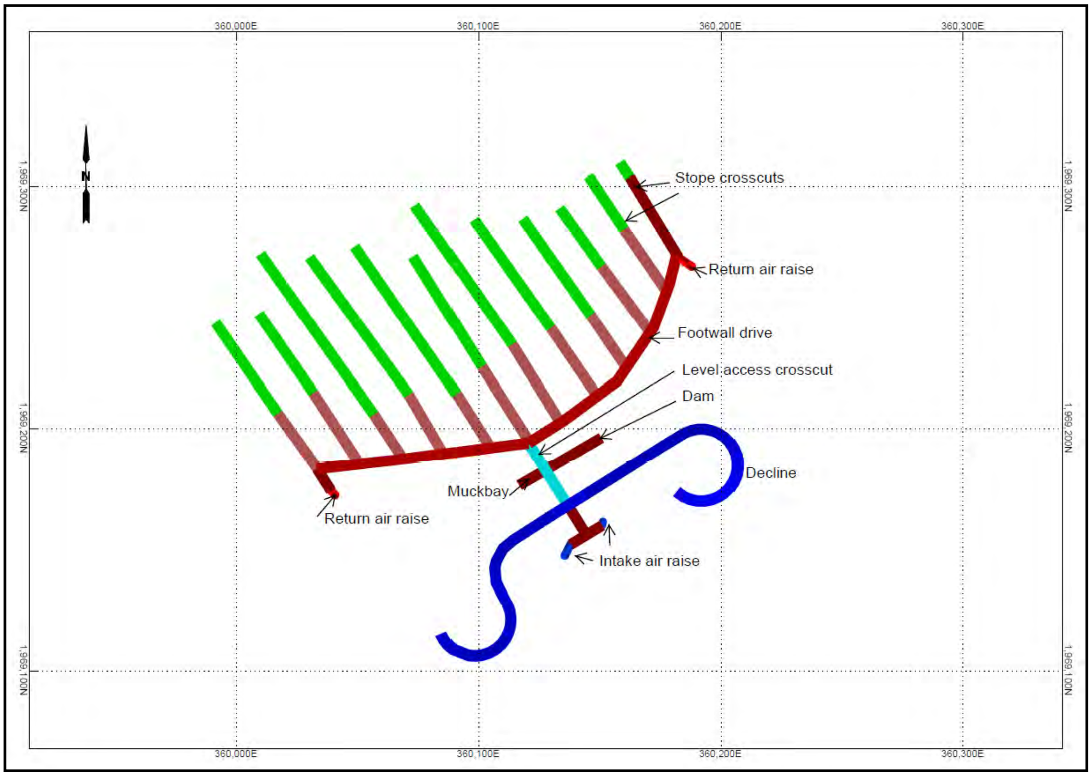


Figure 16-23: Plan View of Typical Production Level.

All development will be completed using mechanised drill, blast, support, load, and haul methods. Ore and waste will be hauled to surface by underground articulated dump truck (ADT), where it will be tipped on the RoM pad or waste rock dump. Once stoping commences and adequate voids are available underground for the tipping of waste into, waste will not be hauled to surface but will be placed in mined out stope voids as backfill.

The dimensions of the various excavations included in the development design are shown in Table 16-13 below.

Table 16-13: Development Excavation Dimensions.

Excavation	Width (m)	Height (m)	Area (m^2)	SG (t/m^3)	Tonnes/m
Ramp	4.6	5.4	24.8	2.36	58.6
Level access	4.6	5.1	23.5	2.36	55.4
Footwall drive	4.6	5.1	23.5	2.36	55.4
Stope crosscut - waste	4.5	4.5	20.3	2.36	47.8
Stope crosscut - ore	4.5	4.5	20.3	2.36	47.8
Fresh air raise	3.5	3.5	12.3	2.36	28.9
Fresh Air Raise (Raise bore)	4.4		15.2	2.36	35.9
Return air raise	3.5	3.5	12.3	2.36	28.9
Return air raise (Raise bore)	5.0		19.6	2.36	46.3
Service raise	3.5	3.5	12.3	2.36	28.9

The primary waste development which is made up of the ramps, level access crosscuts and footwall drives will all be 4.6 m × 5.1 m (finished dimensions), although the ramp is excavated 0.3 m higher to allow for a permanent road base to be established. Stope crosscuts are 4.5 m × 4.5 m.

Figure 16-24 shows a cross section of a stope crosscut (4.5 m × 4.5 m finished) with the largest equipment expected to operate in these ends, a 14-tonne capacity loader (LHD).

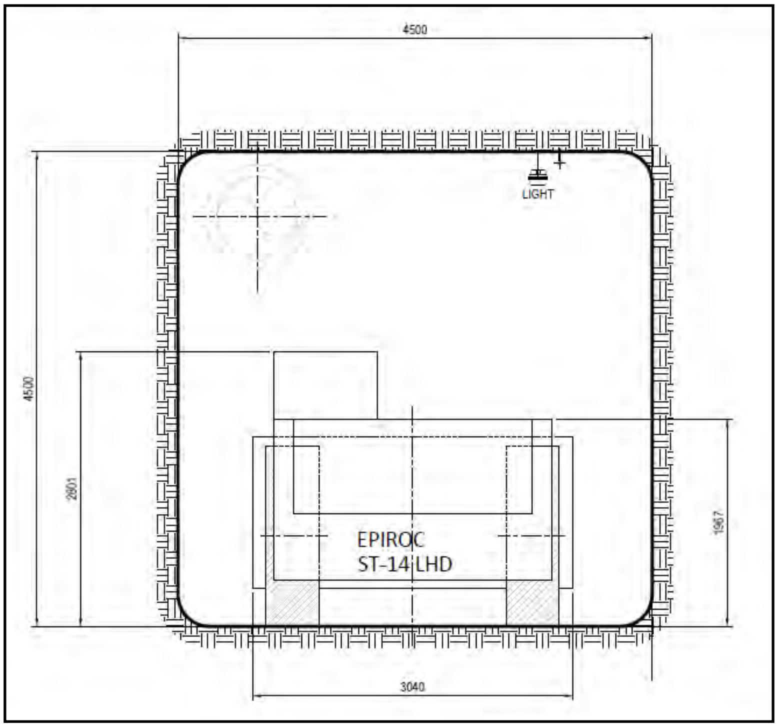


Figure 16-24: Ore Drive Showing 14-tonne Loader.

All development drilling will be done by a twin boom electrohydraulic drill rig (Epiroc Boomer). The blast-hole length will be 4.20 m, resulting in an expected advance per blast of 3.9 m. Holes will be charged with bulk emulsion. The initiation system will be shock-tube long period detonators (LPDs).

16.15. Stoping

Stope Optimisation

To determine the optimum stope shapes, considering the design input parameters as defined in the mine design criteria, DeswikSO® was used. This is a mineable shape optimizer which seeks to create stope shapes, based on pre-set input parameters, while optimising the extraction of ore from the resource.

A primary input to the DeswikSO® process is, the specification of the cut-off grade to apply. DeswikSO® will attempt to produce stopes that have a grade higher than the specified cut-off grade. Table 16-14 below shows the cut-off grade estimate used in the stope optimisation process.

Table 16-14: Cut-off Grade Calculation.

Item	Value
U_3O_8 Price ($/lb)	70
lb/kg	2.204
U_3O_8 Price ($ /kg)	154.32
Operating cost	
Mining	83
Processing	77
Overheads (G&A)	40
Operating cost ($/ t milled)	**200**
Breakeven recovered grade (kg/t)	1.30
Metallurgical recovery	92.45%
Breakeven RoM grade (ppm) before royalty	1402
Royalties (%)	7
Breakeven after royalty	**1500**
Note: Operating costs used in stope optimisation may not be the same as those reported in the final operating costs as this process is based on preliminary data available at the start of the project.	

Other inputs into the stope optimisation include the stope dimensions and pillar locations. Table 16-15 shows the input parameters used in the Dasa stope optimisation.

Table 16-15: Stope Optimisation Input Criteria.

Item	Unit	LHOS	Comment
Stope height (sublevel spacing)	m	22.5	Geotech guideline
Stope strike length	m	Min = 5, Max=100	
Minimum mining width	m	5	
Maximum stope width	m	16.5	Geotech guideline
Minimum stope dip	Degree	60	
Rib pillar width	m	5	One rib pillar in Block C at Y=360290
Sill pillar thickness	m	8.5	One sill pillar in Block A at approximately Z=300 m
Stope direction	azi º	325	Transverse stopes
Cut-off Grade	ppm	2074	RoM grade from Stopes

Inferred resources were not targeted in the stope optimisation process. Only Inferred material which was included in a stope which contains mostly indicated material was included and treated as zero grade waste. For a stope to be included in the mine design it needed to contain a minimum of 1000 tonnes of indicated resources. Figure 16-25 shows the stope shapes produced by DeswikSO®.

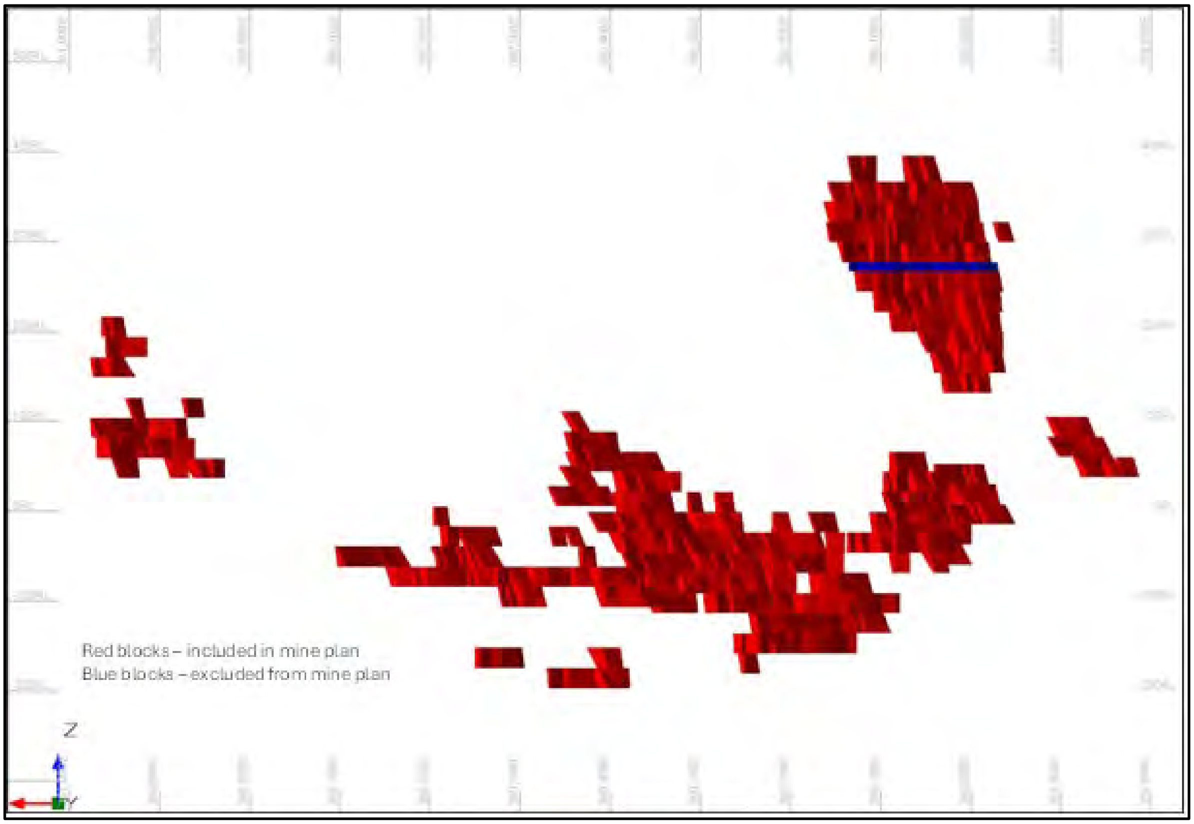


Figure 16-25: Stope Shapes Produced by DeswikSO®.

The stopes coloured red in Figure 16-25 were include in the mine design. Those coloured blue were excluded due to being small, isolated blocks of ground, i.e., requiring a significant amount of development to access them which would render them unviable. Table 16-16 shows the total stoping inventory after removal of the excluded stopes.

Table 16-16: Mining Inventory from Stopes – Cut-off 1500 PPM U3O8 ($70/lb).

Category	Unit	Tonnes	U_3O_8 PPM	U_3O_8 (t)
In-situ		7,164,467	4,278	30,648
Measured	Tonne	0	0	0
Indicated	Tonne	7,161,088	3,957	30,634
Inferred	Tonne	3,379	3,399	14

Stope Design

Once the lateral development on a level is completed and the upcast ventilation raises linking to the level above have been developed, the level is ready for the stope crosscuts to be developed, after which stoping can commence. Stope preparation starts with the development of a slot raise at the end of the stope.

The slot raise will be mined at 2.0 m × 2.0 m and will hole between sublevels. Slot raises will be mined using drop raising techniques. The drilling of the holes will be completed with the long-hole production rig. Once the slot raise is completed, the raise will be opened, up to the full width of the stope, to form a complete slot, using a pattern of fanned blast holes. Once this slot is complete the stope is ready for production stoping to commence.

The preferred method of operation of the stope is for several rings to be pre-drilled and then blasted, one ring at a time. However, if ground conditions do not allow for pre-drilling of holes (i.e., holes closure is excessive between the time of drilling and charging of the holes), then rings will be drilled one at a time, immediately prior to blasting.

The sequence of mining is to retreat from the ore block extent, back towards the stope access. After the stope has advanced to the orebody limit or a maximum distance of 90 m from the initial slot raise, backfilling must take place and the backfill must cure, before re-slotting and recommencing the stoping operation. The backfill will be deposited into the stope from the stope access crosscut on the level above.

The sequence of mining is bottom-up from a main level. Conventionally, a permanent sill pillar would be left below the main level in this type of mining. In this case the sill pillar will be mined in a similar manner to a normal level once all mining in the mining panel is completed. A reduced mining extraction factor is assumed in the sill pillar mining as it is expected that ground conditions will be poor when mining the sill pillar.

16.16. Mine Scheduling

A mine layout for the underground mine was developed using the Deswik® suite of mine design and scheduling software. A 3-D layout was drafted in DeswikCad®. The layout includes the development excavations and stopes for the LoM. The mine layout was then exported to DeswikSched®, the mine scheduling module of the Deswik® suite. The activities making up the mine schedule were sequenced logically.

Mining productivities were estimated for each excavation type and used in the production of the mining schedule. The productivities by excavation type are shown Table 16-17 below.

Table 16-17: Mine Scheduling Parameters.

Item	Unit	Value
Stope Productivity		
Long hole drilling.	m/month	5700
Stope Loading (LHD).	t/day	900
Backfilling.	m^3/day	400 (minimum)
Backfill curing time.	days	14
Development Advance		
Ramp.		75
Level Access.		60
Footwall drive.	m/month	60
Stope crosscut – waste.		60
Stope crosscut – ore.		50
Drop raising.		24

The activities were then scheduled, considering, the productivities detailed in Table 16-17, as well as the availability of resources (mining equipment) allocated to each activity.

The key aspects in the mining schedule are:

A full mining schedule for the underground project is detailed in Table 16-18, with a graphic profile presented in Figure 16-26.

- SOMIDA had completed the boxcut excavation and 526 m ramp and access development have been completed by the end of December 2023. Scheduling of development started from survey positions of as-built development as at end December 2023.
- The Upcast ventilation system is established by the completion of ventilation raise in August 2024.
- Level development commences on the 5367 Level in September 2024.
- First ore is produced from development in September 2024.
- Stoping commences in October 2025 on the 5323 Level.
- Steady state production of 32,000 tonnes per month of ore is achieved in January 2027.
- The Phase 1 mine has a life of just over 26-years.

A full mining schedule for the underground project is detailed in Table 16-18, with a graphic profile presented in Figure 16-26.

Table 16-18: Dasa Underground Mining Schedule by Year.

	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049	Total
Ramp Development Metres	921	992	739	1,116	727	758	589	1,010	338	-	536	395	351	542	371	306	-	-	-	-	-	-	-	-	-	-	**9,690**
Cubby Metres	762	791	778	986	1,378	624	985	795	376	157	709	313	529	494	163	393	-	-	-	-	-	-	-	-	-	-	**10,231**
FW Drive Metres	734	686	547	1,021	1,872	1,737	844	435	628	230	502	491	248	522	219	78	-	-	-	-	-	-	-	-	-	-	**10,792**
Access Xcut Metres	114	173	92	150	123	84	299	109	68	16	113	56	49	59	79	60	-	-	-	-	-	-	-	-	-	-	**1,643**
FAR Metres (Drop Raise)	86	91	65	67	75	67	88	25	25	23	32	33	23	59	42	-	-	-	-	-	-	-	-	-	-	-	**800**
RAR Metres (Drope Raise)	146	256	107	20	213	86	147	36	61	24	115	32	147	101	41	39	-	-	-	-	-	-	-	-	-	-	**1,571**
Dam Metres (Drop Raise)	-	35	35	-	35	-	35	-	-	-	36	-	35	34	-	-	-	-	-	-	-	-	-	-	-	-	**245**
FAR Metres (Raisebore)	80	-	-	-	-	-	-	171	-	-	-	53	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**303**
RAR Metres (Raisebore)	66	-	-	85	445	-	-	-	10	-	-	-	166	-	-	-	-	-	-	-	-	-	-	-	-	-	**772**
Stope Xcut (W)	413	1,127	678	838	926	863	542	284	733	762	679	469	849	681	992	749	254	-	-	-	-	-	-	-	-	-	**11,837**
Stope Xcut (O)	299	1,895	1,753	2,057	1,076	685	342	716	1,457	2,436	1,062	1,876	1,574	1,019	1,595	1,698	828	-	-	-	-	-	-	-	-	-	**22,368**
Slot Raise Metres	-	90	288	450	504	414	522	630	432	558	342	486	468	414	409	450	482	441	396	522	432	400	608	515	331	162	**10,746**
Stope tonnes LHOS	-	18,959	158,242	308,957	326,201	286,270	307,178	352,265	308,208	245,041	262,816	308,504	297,269	341,523	299,801	318,151	352,656	307,109	366,834	296,767	338,641	343,650	278,620	324,606	312,454	103,745	**7,164,467**
Stope grade	-	4,344	5,750	6,866	5,840	7,562	5,062	5,070	5,490	4,286	5,514	5,717	4,290	4,723	3,494	4,514	4,593	2,569	2,458	2,738	2,590	2,751	2,259	2,723	3,246	2,982	**4,276**
Stope U308 (t)	-	82	910	2,121	1,905	2,165	1,555	1,786	1,692	1,050	1,449	1,764	1,275	1,613	1,048	1,436	1,620	789	902	813	877	946	629	884	1,014	309	**30,634**
Ore Development	12,298	78,981	69,757	66,004	46,020	28,729	14,961	31,733	66,261	101,642	34,482	73,915	53,078	42,476	66,827	65,845	29,135	-	-	-	-	-	-	-	-	-	**882,143**
Development ore grade	2,442	6,796	3,477	3,926	1,158	1,480	2,292	3,085	1,853	1,666	2,967	2,205	4,884	1,678	1,334	1,885	2,224	-	-	-	-	-	-	-	-	-	**2,791**
Development ore U308 (t)	30	537	243	259	53	43	34	98	123	169	102	163	259	71	89	124	65	-	-	-	-	-	-	-	-	-	**2,462**
Total Ore Tonnes	**12,298**	**97,939**	**228,000**	**374,962**	**372,221**	**314,999**	**322,139**	**383,997**	**374,469**	**346,683**	**297,298**	**382,419**	**350,348**	**383,999**	**366,627**	**383,996**	**381,791**	**307,109**	**366,834**	**296,767**	**338,641**	**343,650**	**278,620**	**324,606**	**312,454**	**103,745**	**8,046,610**
RoM U308 ppm	**2,442**	**6,322**	**5,055**	**6,349**	**5,261**	**7,008**	**4,933**	**4,906**	**4,846**	**3,518**	**5,218**	**5,038**	**4,380**	**4,387**	**3,101**	**4,063**	**4,412**	**2,569**	**2,458**	**2,738**	**2,590**	**2,751**	**2,259**	**2,723**	**3,246**	**2,982**	**4,113**
RoM U308 (t)	**30**	**619**	**1,152**	**2,381**	**1,958**	**2,207**	**1,589**	**1,884**	**1,815**	**1,220**	**1,551**	**1,927**	**1,535**	**1,684**	**1,137**	**1,560**	**1,685**	**789**	**902**	**813**	**877**	**946**	**629**	**884**	**1,014**	**309**	**33,097**
Waste Tonnes	178,871	220,723	162,231	230,042	305,220	225,093	187,619	168,543	117,023	60,224	143,806	96,535	123,131	131,561	97,894	84,485	12,146	-	-	-	-	-	-	-	-	-	**2,545,147**
Hauled Tonnes	191,169	318,662	390,230	605,003	677,440	540,092	509,758	552,541	491,492	406,908	441,104	478,954	473,478	515,560	464,521	468,481	393,937	307,109	366,834	296,767	338,641	343,650	278,620	324,606	312,454	103,745	**10,591,757**
Tonne.km	194,935	379,695	562,624	948,222	1,078,695	839,990	769,887	1,024,803	1,147,423	1,175,819	1,210,787	1,698,422	1,634,334	1,792,218	1,706,723	1,706,185	1,601,763	1,304,939	1,580,867	1,269,866	1,430,757	1,446,170	1,171,639	1,240,724	1,111,699	350,465	**30,379,651**

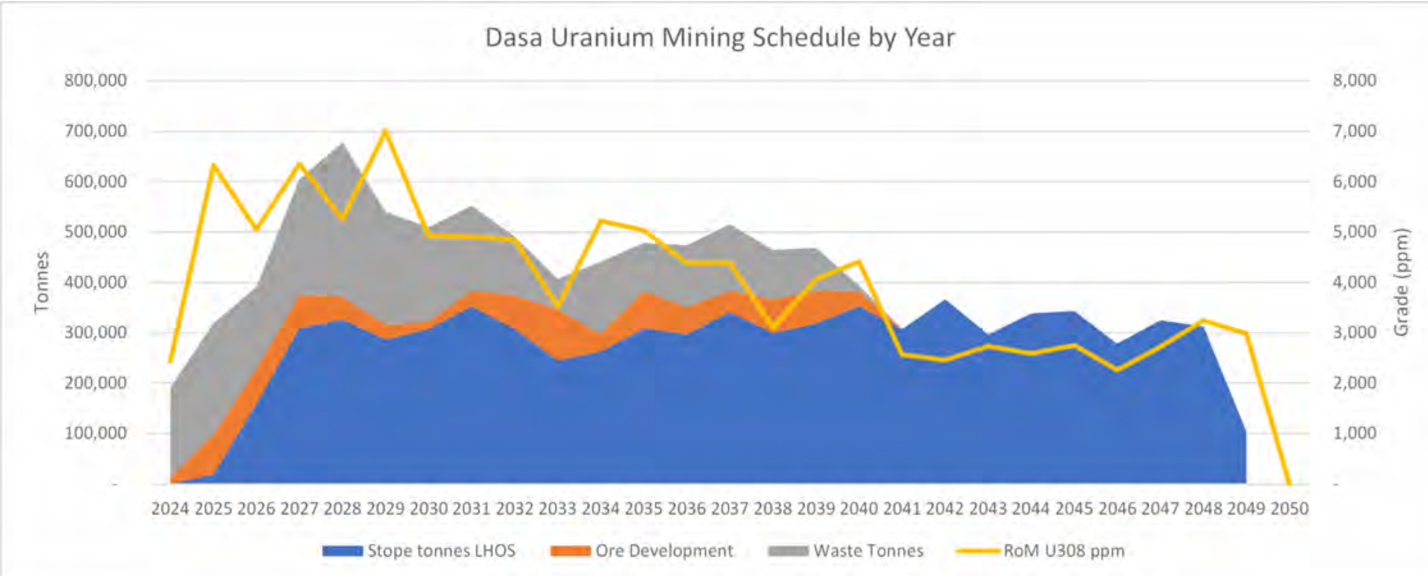


Figure 16-26: Dasa Phase 1 Underground Production Profile Update.

16.17. Mining inventory

The total mining inventory resulting from the mine design and layout is shown by mining zone and by resource category in Table 16-19 and Table 16-20 below.

Table 16-19: Mining Inventory by Mine Zone Update.

Zone	In-situ Tonnes	U_3O_8	RoM Tonnes	RoM U_3O_8	RoM U_3O_8
Unit	**Tonnes**	**PPM**	**Tonnes**	**PPM**	**Tonnes**
1	2,479,319	6,057	2,372,172	6,014	14,266
2	397,589	2,745	365,778	2,834	1,037
3	3,838,229	3,716	3,643,477	3,719	13,551
4	1,273,076	2,336	1,203,921	2,346	2,825
5	515,316	2,959		3,141	1,449
Total	**8,503,528**	**4,101**	**8,046,610**	**4,117**	**33,127**

Table 16-20: Mining Inventory by Resource Class.

Category	RoM tonnes	U_3O_8 ppm	U_3O_8 (t)	U_3O_8 (Million lbs)
Measured	-	-	-	
Indicated	8,035,902	4,119	33,097	72.964
Inferred	10,708	2,819	30	0.066
Total Mining inventory	**8,046,610**	**4,117**	**33,127**	**73.031**

The limited quantity of inferred resources is a result of inferred resources occurring within the stope shapes resulting from DeswikSO®. The inferred material amounts to 0.13% of the RoM tonnes and was treated as zero grade waste.

16.18. Mining Equipment

The mining equipment fleet at Dasa has been selected with the production rate, mine design, orebody dimensions and geometry in mind and is aimed at a balance between minimising dilution and losses while maximising productivity. Dasa will provide the underground mining equipment which will be operated by a mining contractor. Dasa has already acquired several equipment units.

In order to select the size class of mining equipment for Dasa a trade-off study was undertaken. The selection of the equipment has a direct bearing on the size of the access excavations required. With waste development being the largest item of mining capital expenditure, Bara undertook a trade-off study in order to support the selection of the appropriate mining equipment and development excavation size. The trade-off study considered the following cost items:

- Capital cost of equipment purchase.
- Operating cost of equipment over the life of mine.
- Development cost (which is dependent on size of excavations).

In total three size classes of equipment were considered. Since the focus of this work is to select the size of equipment, not specifically the model to be purchased, all models were selected from a single supplier, Epiroc. In addition to the three size classes of equipment, Epiroc also offer one size class (40 tonne ADT and 14 t LHD) as a battery electric vehicle (BEV) option. This was included as an additional option in the study.

The equipment considered in the trade-off is listed below in Table 16-21.

Table 16-21: Equipment Considered.

Option	ADT	LHD
Option 1	MT54 (54 tonne)	ST 18 (18 t)
Option 2A	MT42 (42 t)	ST14 (14 t)
Option 2B	MT42 BEV	ST14 BEV
Option 3	MT436B (33 t)	ST1030 (10 t)
Option 4	MT2200 (22 t)	ST7 (7 t)

To accommodate the equipment considered in each option the access excavations need to be adjusted. The smaller equipment will allow for smaller ends while the larger units require larger excavations to operate in. In determining the excavation cross sections, the following minimum tolerances were allowed:

- Side clearance - minimum 0.75 m each side of the truck.
- Top clearance - minimum of 1.5 m (to accommodate ventilation column and services.

The development cost was determined using the average development cost from two mining contractor submissions obtained by GAC and provided to Bara. Atlas Copco (Epiroc) South Africa was approached to supply the estimated purchase cost and operating cost of the loaders and trucks.

The operating cost for the BEVs, provided by Epiroc, excluded the cost of power and the replacement cost of the batteries over the life of the equipment. In verbal communication with the Epiroc Product Manager it was stated that the power plus battery cost is likely to be close to the fuel cost of the diesel unit. This assumption was used in the cost estimate for the BEVs.

The study considered the timing of the expenditure, and a cashflow model was created for each option and a discounted cost determined, using a discount factor of 10% per year. The results of the cost comparison, total overall cost and the net discounted cost are summarised below in Figure 16-27 and Figure 16-28.

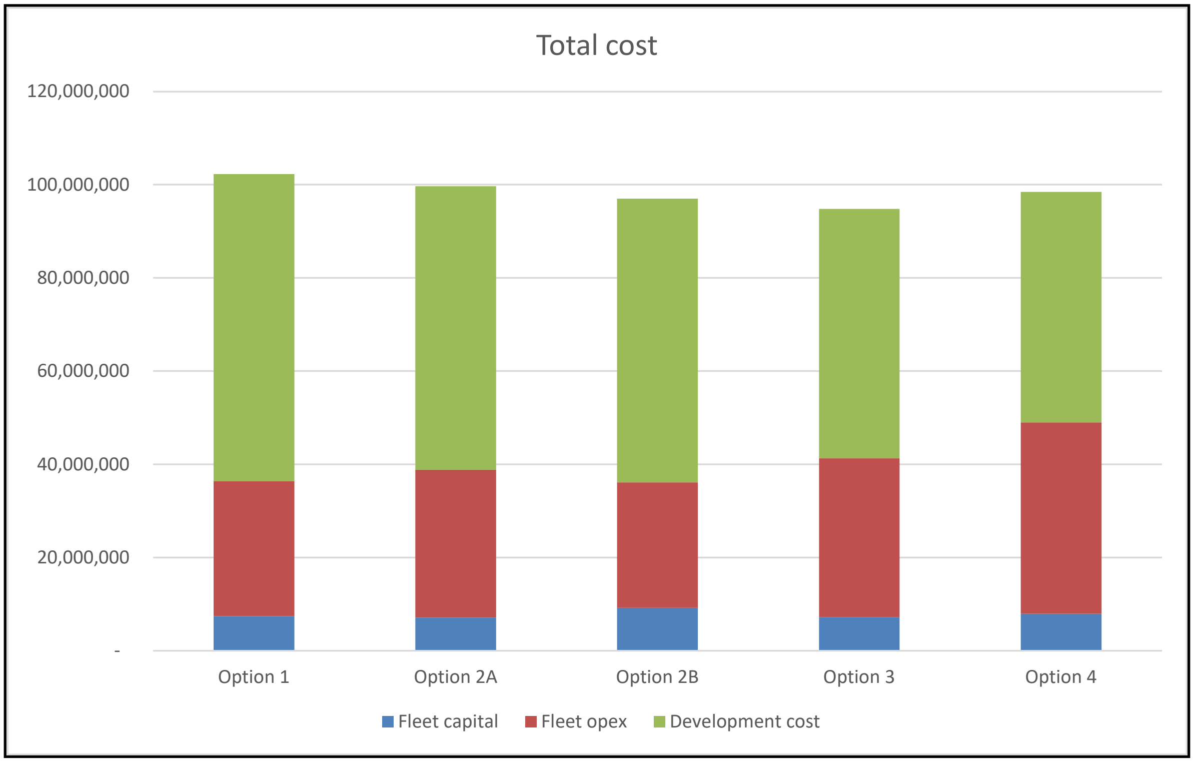


Figure 16-27: Comparison of Total Costs.

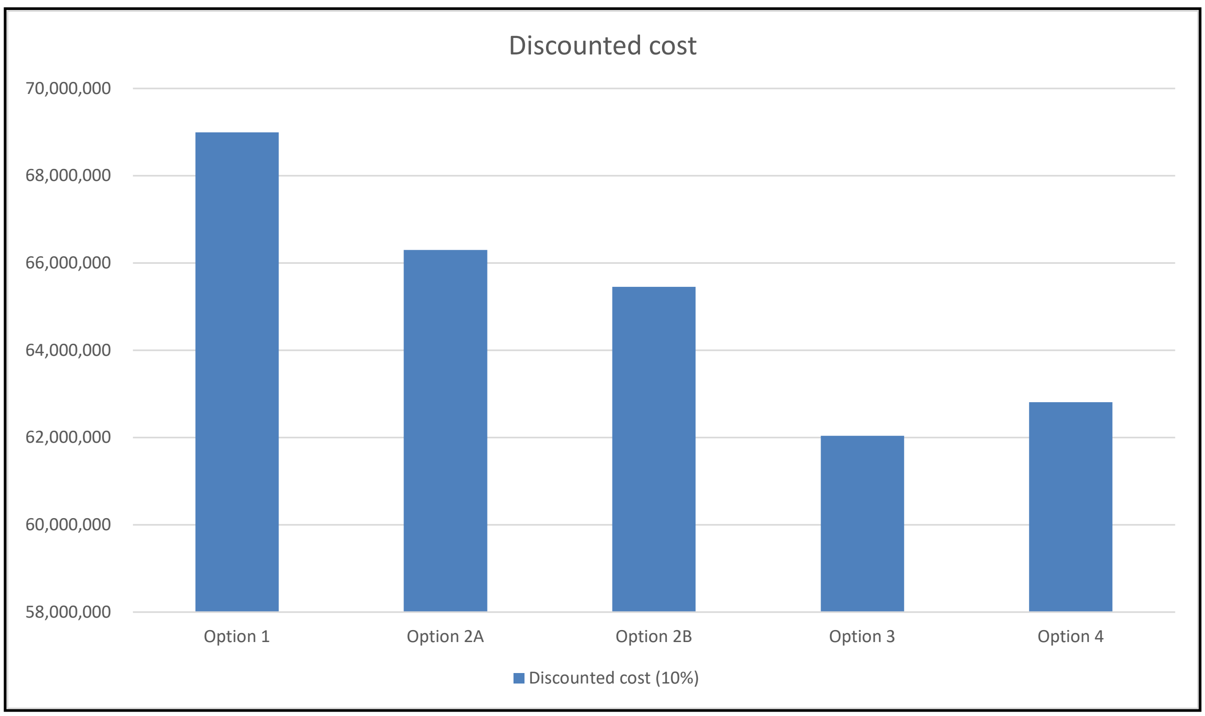


Figure 16-28: Comparison of net Discounted Cost.

For the DFS production rate of 1000 tpd, based on costs, the most favourable option is Option 3, the 32 tonne trucks and 10 tonne loaders. It must be noted that the overall costs are all within $7 million (7%) of each other. This is within the accuracy of the study work completed, so the numbers could be considered as equal. However, the relative value does give an indication of which option is likely to be cheaper.

The study has shown that Option 2B (BEV) is more cost effective than the diesel option of the same size class. After discussion of the results with the client, GAC initially selected the BEV option. After further investigation into the delivery times of the BEV equipment, GAC opted for diesel units of the equivalent size (14 t LHD and 40 t trucks) to start the mine, leaving the option open for conversion to BEV at a later stage in the mine life.

Table 16-22 shows a summary of the mining fleet required to support the mining schedule.

Table 16-22: Summary of Mining Equipment Fleet.

Description	Qty
Development drill rigs Twin boom	3
Bolter	2
Long Hole Drill Rig	2
LHDs (14 t)	3
ADTs (42 t)	5
Grader	1
Utility vehicles	3
Transmixer	1
Shotcreter	1
Telehandler	2
Pick-ups	6
Troop carrier	2
Grade control drill rig	1
Service holes drill rig	1

16.19. Ventilation

Ventilation System Overview

The mine ventilation system is a key aspect of the mine design due to the presence of radioactive elements in the air. The ventilation system designed is a once through system (no recirculation of air) and will replace the volume of air in the mine on average every 15 minutes. Primary intake air will flow through the decline system as well as a dedicated Fresh Air raise (FAR) and from here into the sub-levels and stoping areas. Used air will be collected from the stoping areas by exhaust ventilation columns and directed into the return airway system and eventually out of the mine through a dedicated Return Airway (RAW)

Ventilation of excavations within the orebody where radiation risk is higher is by use of an exhaust system which removes contaminated air from the workings directly into the return airway system, ensuring that risk relating to exposure to radiation is always minimised.

Ventilation Design Criteria

The ventilation design criteria used conforms to established international best practices to provide a safe and healthy working environment. Table 16-23 to Table 16-27 below list the principal criteria used in the design of the ventilation systems for Dasa.

Table 16-23: Environmental Criteria.

Description	Unit
Design intake air temperature [wet bulb/dry bulb]	22.0/32.0 ºC
Design relative humidity	40%
Design reject air temperature [wet bulb/dry bulb]	32.0/37.0 ºC
“Withdraw from working place” wet bulb temperature	32.5 ºC
Air to engine rated diesel power ratio at point of use	0.06 $m^3/s/kW$
Overall air leakage factor for the mine	20%

Table 16-24: Shaft and Airway Criteria.

Excavation	Criteria
Declines and intake air tunnels – air velocity	Max 8.0 m/s
Return airways – air velocity where people need to enter	Max 10.0 m/s
Maximum Air Residence Time for Production Crosscuts	4 mins
Maximum Air Residence Time for Other Excavations	15 mins
Unequipped air raises and raise bored holes – air velocity	Max 22.0 m/s
Return air raises with emergency ladders, pipes, and cables – air velocity	Max 15.0 m/s
Declines, haulages, and crosscuts [average blast] – Friction Factor	0.015 Ns^2/m^4
Unequipped Raises [rough blast] – Friction Factor	0.02 Ns^2/m^4
Ladder way, pipes & cables equipped raises – Friction Factor	0.03 Ns^2/m^4
Raise Bored Hole [upcast]	0.004 Ns^2/m^4

Pressure drops, economics and practical issues were considered when determining final velocity for dedicated return airways and other airways, Table 16-25 gives the minimum excavation dimensions required by the ventilation design.

Table 16-25: Excavation Dimensions.

Excavation	Width [M]	Height [M]
Ramp	4.6	5.4
Level Access	4.6	5.1
Footwall Drives	4.6	5.1
Stope Crosscut	4.5	4.5
Fresh Air and Return Airways	3.5	3.5
Vent Shaft [upcast & intake]	5.0 m Diameter Raise Bored Hole	

The lack of measured geothermal rock properties for the orebody and host rock at the Dasa Uranium Project site has led to some assumptions being made, namely that most of the access excavations and airway are situated in Sandstone with only the production crosscuts in the orebody. The generic thermal properties of Sandstone were used in the computer simulation models and was applied to all excavations, they are shown in Table 16-26 below.

Table 16-26: Rock Properties.

Rock Type	Sandstone
Geothermal Gradient	2.5 ^{o}C/100 m
Rock Density	2,700 kg/m^3
Rock Specific Heat	790.0 J/kgC
Rock Thermal Conductivity	2.0 W/mC
Rock Thermal Diffusivity	0.938 m^2/s 10^{-6}
Surface Datum Rock Temperature	22.0 ^{o}C

The mining fleet selected was also evaluated in the ventilation design, Table 16-27 below shows the selected units, the number of units, the kilowatt rating and the percentage utilisation used in the ventilation design process.

Table 16-27: Vehicle Fleet.

Equipment	Number	Rating kW	Utilization
Development Rigs	3	72	30%
Mine truck MT42	5	319	80%
ST 14 Loader	4	250	80%
Roof Bolter	2	72	80%
Stope Rigs	1	72	30%
Telehandler	2	120	50%
UV Trans mixer	1	120	30%
UV Shotcrete	1	120	30%
UV Maintenance	3	120	50%
Grader	1	120	30%
LDV	2	150	50%

Primary Ventilation System

Traditionally, the primary airflow quantity for mechanised mines is based on amount of air required to dilute the exhaust gases released by all the diesel equipment to maximum accepted levels in the atmosphere. Uranium Mines on the other hand will require enough ventilating air to ensure that all employees in the mine have minimum exposure to radiation levels.

To minimise radiation exposure of the workforce, the decline system, all airways, and the footwall drives will be situated in the host rock outside the orebody and only the production crosscuts and open stopes will be situated in the orebody. The excavations in the host rock can be ventilated with conventional force ventilation methods and the excavations in the orebody will be ventilated by the exhaust overlap ventilation method.

In uranium mines, excavations in the orebody should not be ventilated in series, and in order to protect the workforce from radiation exposure from the orebody, the air needs to be replaced every three to four minutes. It is recommended that a rigid exhaust column with an exhaust quantity of 10-15 m^3/s be used to ventilate a Production Crosscut situated in the orebody and will replace the total volume of a completed crosscut in 3 to 4 minutes.

Planning indicates that three operating open stopes will be required to produce the design tonnage of approximately 30,000 tonnes per month. Each stope has a top crosscut and a bottom crosscut so there will be six crosscuts to be ventilated, and the same will apply to the four stopes required for backfill, to maintain the production target. Therefore, a minimum of fourteen crosscuts needs to be ventilated, there will also be additional crosscuts that will be required for stope preparation, and these also need to be ventilated. The total number of production crosscuts that will need to be ventilated is twenty crosscuts spread over four levels which translates to a minimum of 200 m^3/s and a maximum of 308 m^3/s of ventilation air. In addition, a single development level will require two development ends and the decline development which will add an additional 90 m^3/s to the total air requirement.

In addition to the requirement to replace air every 3 to 4 minutes in production areas, it is also planned that the retention time of the air flowing through the mine will not exceed 15 minutes. To achieve this, will require an additional intake airway in parallel to the decline as the decline cannot handle the required airflow and allow for sufficient air to be removed from the decline at the point where the air has been travelling for 15 minutes and then replace the air with fresh air from the fresh air raise (FAR). This point will be at the bottom of the Zone 2 mining area. The fresh air travelling in the FAR travels at high velocity and does not exceed the 15-minute limit. The minimum airflow in the decline to achieve the required retention time will be 150 m^3/s.

To achieve the above airflows will require a total airflow of 400 m^3/s flowing via the decline and the FAR next to the decline to the operating levels, where the ventilating air will enter through the access crosscuts to the footwall drives. The air will then be drawn from the footwall drive along the length of the production crosscut to the end of the exhaust column situated in the crosscut. Air exhausting along the exhaust column in each production crosscut flows back to the footwall drive into a collector exhaust column which connects through a fan to the Return Airways Raises (RAW) on each end of the footwall drive. From these RAWs the air is exhausted through the main surface fans to the atmosphere on surface.

Figure 16-29 illustrates the primary ventilation flow described above.

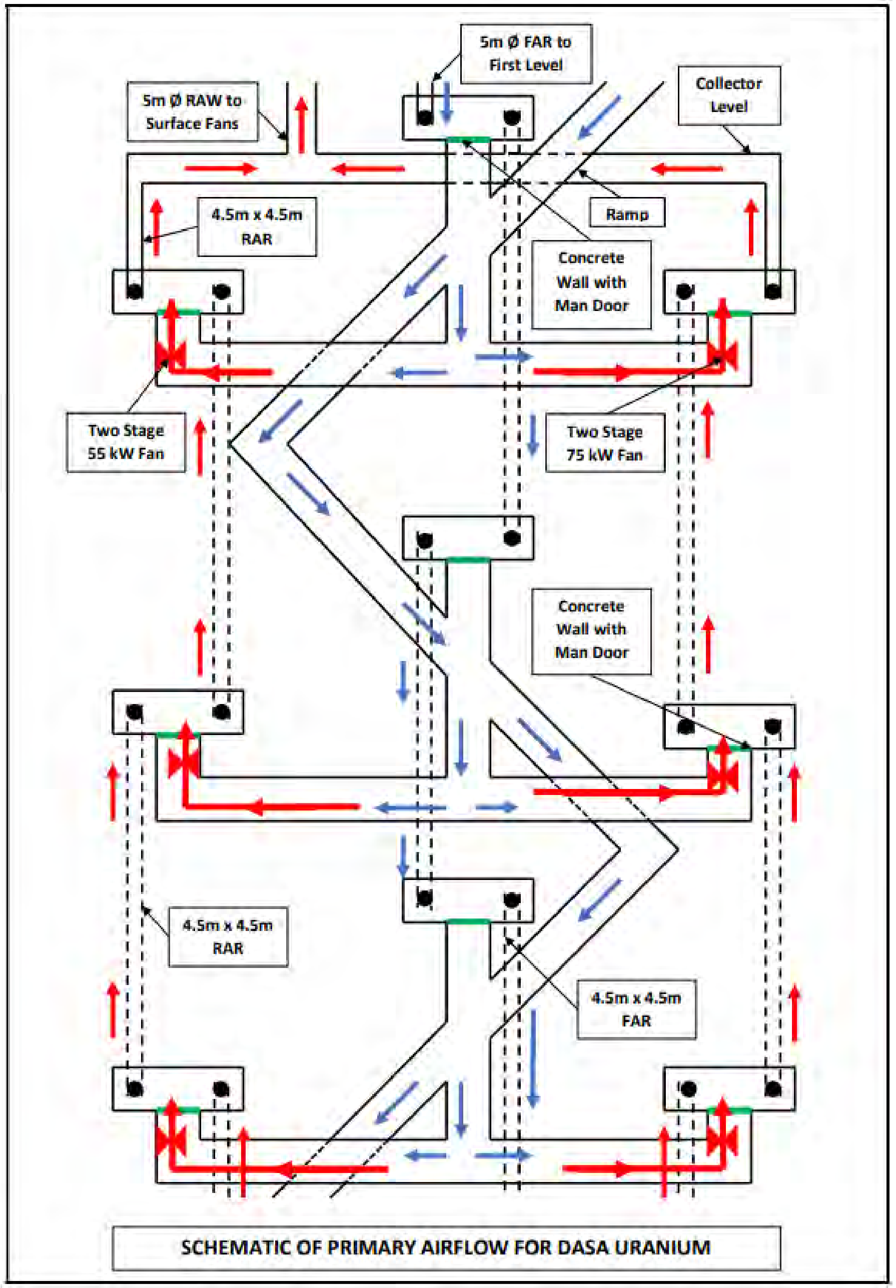


Figure 16-29: Schematic of Primary Airflow at Dasa.

Production Level Ventilation

As previously stated, the production levels will be ventilated by means of an exhaust overlap system, consisting of 1015 mm diameter rigid steel ducting carried continuously 40 metres from the face with a 760 mm diameter lay flat ducting powered by a 22-kW axial flow fan to force ventilate to the face in an overlap format. The production crosscut exhaust column connects to a 1200 mm diameter exhaust duct connected to the RAW through an axial flow fan at either end of the footwall drive.

Five crosscuts will be ventilated on each level, so the exhaust system will be split into two distinct systems consisting of one 1200 mm exhaust column connected to one of the RAWs on the level. Each of the main exhaust columns will stretch from the halfway point of the footwall drive to a RAW through a two stage 75 kW fan exhausting three production crosscuts and the other will connect through a two stage 55 kW fan exhausting two production crosscuts. The 1200 mm exhaust column is equipped with a lateral piece 1200 mm – 1015 mm and a damper at each production crosscut position where the 1015 mm diameter production exhaust column connects to the main 1200 mm diameter exhaust column. This arrangement allows for various combinations of five ventilation crosscuts and provides for flexibility in the ventilation arrangements on a level.

Based on the mine scheduling, the number of production crosscuts that need to be ventilated, the number of exhaust fans required, and the number of force fans required for each stage are listed in Table 16-28 below

Table 16-28: Secondary Fan Requirements.

Item	Zone 1	Zone 2	Zone 3	Zone 4	Zone 5
No. of Crosscuts to be ventilated	22	19	16	17	16
No. of two stage 75 kW Fans	5	5	4	3	3
No. of two stage 55 kW Fans	5	5	4	3	3
No. of single stage 22 kW Fans	22	19	16	17	16

The open stopes will be backfilled but during the period before backfilling the production crosscut must be ventilated or sealed with ventilation curtaining if the exhaust ventilation system infrastructure is required in another crosscut. Open stopes must not be used as an airway but must be isolated from the ventilation system when work is not being undertaken there.

When a crosscut or level in the orebody has been mined out, it is necessary to seal off these areas with concrete walls to limit the leakage of air through the orebody even although the system is designed to only leak into the RAWs.

Figure 16-30 illustrates a schematic of the ventilation arrangements on a production level.

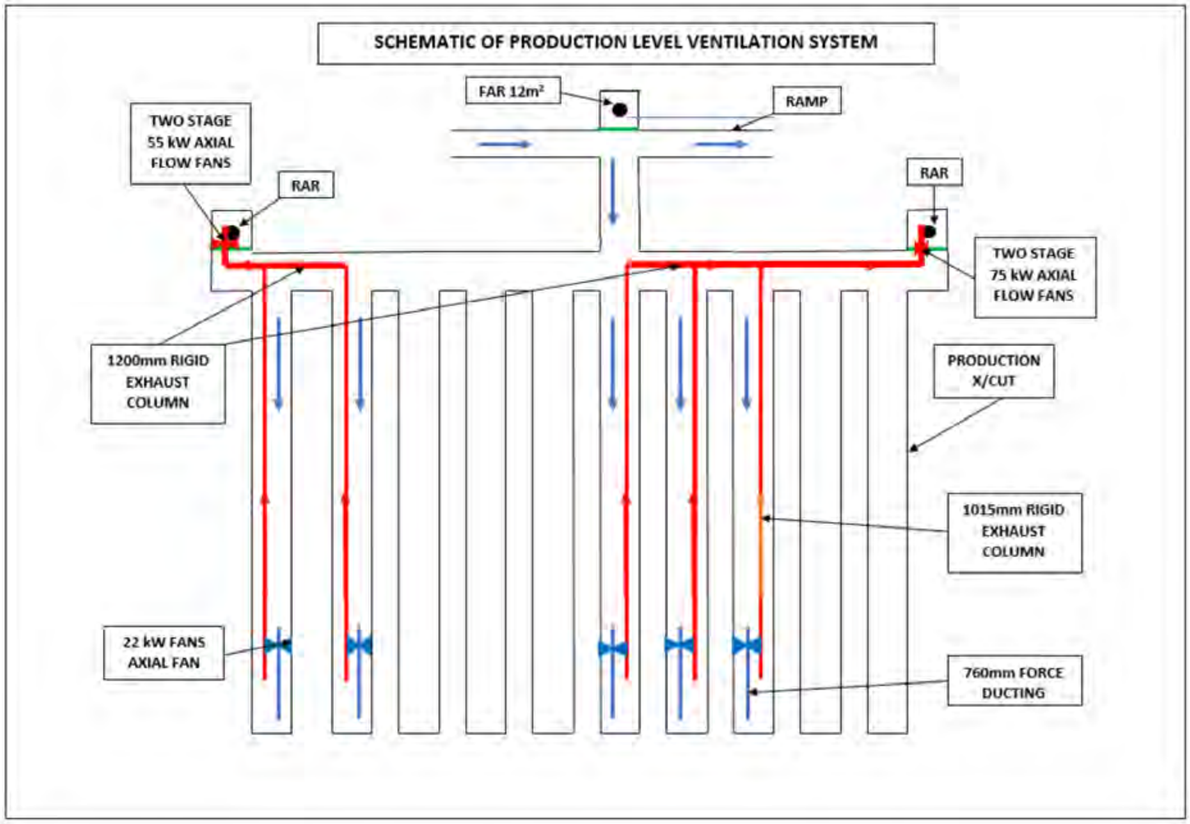


Figure 16-30: Schematic of Production Level Ventilation System.

Development Ventilation Host Rock

All access development in the host rock can be conventionally ventilated by means of axial flow fans and lay flat ducting with the intake of the ducting situated upstream of the last point of through ventilation.

The diesel exhaust gases that need to be diluted and the heat of the diesel engines of rock loading and transport equipment must be taken into consideration. In the areas beyond the point of through ventilation it will be necessary to ventilate for both the LHD and a Truck and thus it will be necessary to use twin 1200 mm diameter lay flat ducting each powered by two stage 75 kW axial flow fans. As the twin ducts proceed down the decline to the next level where one duct will ventilate the level and the other will ventilate the decline. Development on the level to establish the footwall drives and the RAWs is the priority so that through ventilation can be established and the fans moved forward. Figure 16-31 below shows a schematic of the ventilation of the development ends.

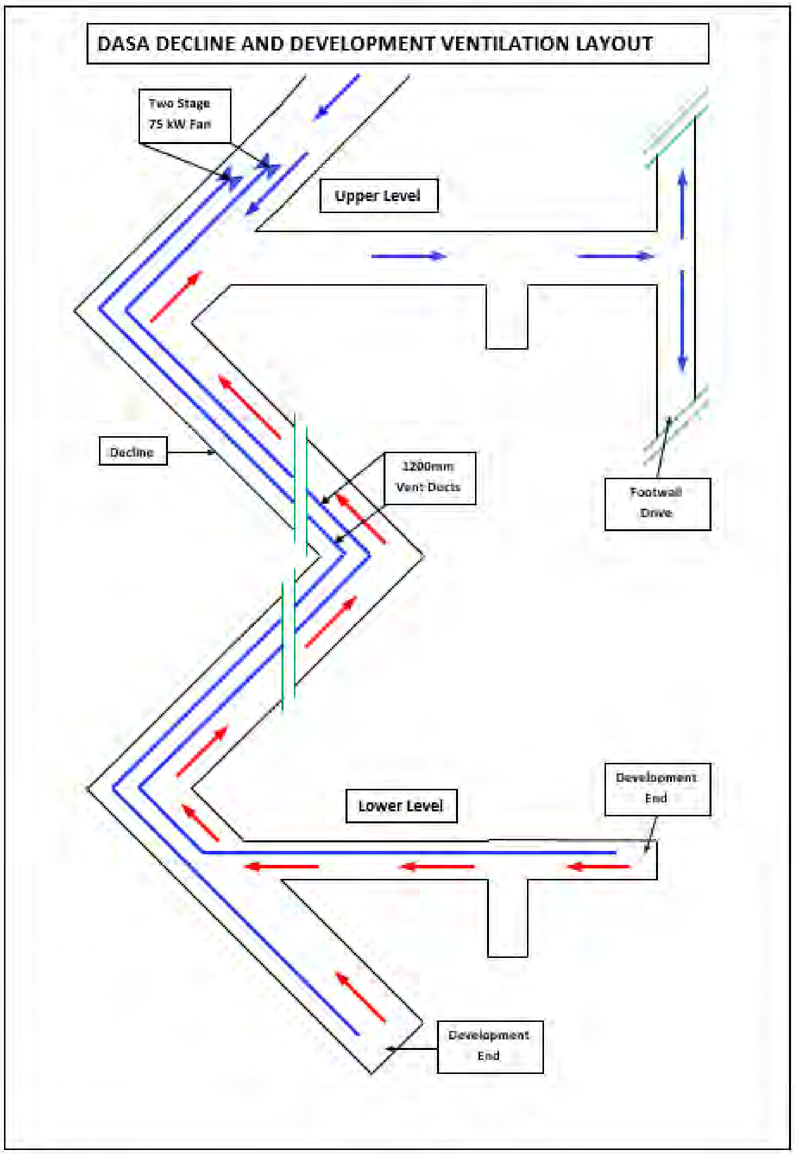


Figure 16-31: Decline and Development Ventilation Layout.

Main Ventilation Fans

Ventsim®, the computer simulation programme, was used to model the ventilation design and make the necessary changes to ensure that the ventilation design is achievable. In addition, the computer simulation also calculates the resistance in the mine and thus the operating parameters of the main and secondary fans.

From the modelling, the operating parameters of the main fans supplied to the fan manufacturers are as detailed in Table 16-29.

Table 16-29: Main Fan Operating Parameters.

Item	Zone 1	Zone 2	Zone 3	Zone 4	Zone 5
Primary Airflow	400 m^3/s	400 m^3/s	400 m^3/s	400 m^3/s	400 m^3/s
Pressure (Shaft Collar)	2696 Pa	1743 Pa	2564 Pa	3721 Pa	2643 Pa

Health and Safety

There are several ventilation related health and safety issues that will be addressed during the mines operational phase as follows:

Fires

Fires involving rubber tyre vehicles present a considerable risk to any underground mine. If the fire is not quickly suppressed and spreads to the tyres, then it can only be extinguished with water. Dry powder is unable to extinguish a rubber tyre fire as it does not remove sufficient heat to prevent re-ignition of the gases emanating from the hot rubber. If the fire is not quickly extinguished with water, then it is almost inevitable that the vehicle will be destroyed.

Dense black smoke with high levels of Carbon Monoxide from burning rubber tyres will quickly circulate through the mine and for this reason body worn Self Contained Self Rescuers (SCSR) and availability of Refuge Bays should be mandatory.

All rubber tyre vehicles should be equipped with on-board fire suppression and a fire extinguisher to control any fire quickly. Vehicle maintenance plays a part in ensuring no issues which can cause fires are present.

Flammable Gas

A robust ventilation system coupled with flammable gas testing to a defined standard and a clear procedure for dealing with any gas intersection is the solution to preventing flammable gas explosions. Appropriate instruments are available to give warning and/or take measurements should be obtained.

Heat

The possibility exists of heat stroke conditions (wet bulb temperature more than 27.5 °C) occurring if the ventilation is not up to standard or some other abnormal circumstance occurs. Instruments to measure wet and dry bulb temperature, air velocity and air humidity are available. It is also recommended that truck and LHD cabs are equipped with air conditioning. As part of the Mine Heat Management Strategy, any workplace where the temperature exceeds wet bulb of 27.5 °C or dry bulb of 32.5 ^{o}C persons should be withdrawn.

Gases

There are two principal situations in an underground mine where immediate danger to persons can be caused.

- Deficiency of oxygen in the general atmosphere. (Normal level 21 per cent and minimum level of 19 per cent). Usual causes – Displacement by other gases or due to a fire.
- Presence of high levels of Carbon Monoxide (CO) - Usual causes – Fire or inadequate air to dilute the diesel exhaust gases.

Other harmful gases can occur in mines, if there is insufficient air to dilute and remove them, examples are:

- Oxides of nitrogen from diesel engines.
- Blasting fumes.
- Welding fumes (Welding/cutting of cadmium plated metal is particularly dangerous).
- Fumes from chemicals used on the mine (Check product data sheets).

Occupational Hygiene

The first defence against occupational exposure is a good ventilation system and, where appropriate, measures to suppress or allay dust (watering down). There are several products available to allay dust on the underground roadways.

Escape and Rescue

Persons proceeding underground should be equipped with a Self-Contained Self Rescuer (SCSR). In addition, refuge bays with a source of breathable air should be made available and positioned so that any person underground can reach one within the duration of their SCSR. Portable refuge chambers are recommended and should be positioned to ensure that persons remote from the escape way in the FAR has a safe refuge in an emergency, for example, one 8-man chamber situated on every operating level and positioned in the footwall drive. Because the FAR is being used as an escape way, it is recommended that the concrete wall sealing off the FAR be equipped with a man door airlock for easy access and then the area behind the concrete wall can be used as a fresh air base (Figure 16-32) by installing a concrete floor, having water available and installing a communication device.

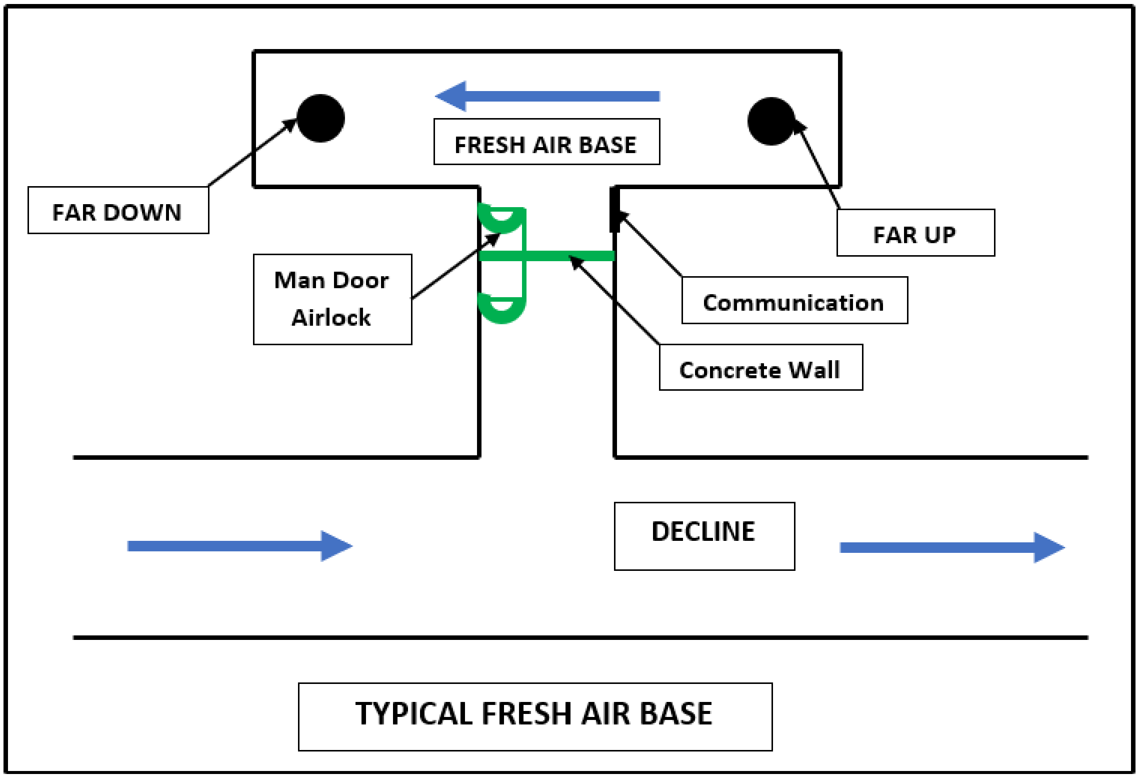


Figure 16-32: Typical Fresh Air Base.

During an emergency where persons need to evacuate the mine, it will be necessary to stop one of the surface fans to reduce the velocity in the Intake Airway (FAR) to allow people to travel safely in this excavation.

Radiation

In the ventilation design for this project, radiation was treated as another contaminant and the recommendations in the reference documents were applied. The ventilation has been designed to minimise the workforce exposure to the radiation, but the level of contamination is highly variable, and is managed pursuant to a Radiation Management Plan to ensure that dose rates are kept as low as possible. The mine has consulted Radiation Protection Experts to provide competent advice on the compilation of the Radiation Management Plan, the purchase of Radiation Measurement Instruments and Personal Radiation Dosimetry System and advise on Qualifications and Expertise necessary for the employment of both a full time Radiation Protection Officer and Occupational Medical Practitioner. According to the schedule development in the Uranium Orebody will only take place at the end of 2024, however, a Radiation Management Plan is currently in place and is administered by management personnel with significant uranium mining and radiation protection experience. Personal Radiation Dosimetry and site Radiation Measurement systems are currently in place with additional specialist instruments available. Radiation measurement measurements are taken daily and sent for outside analysis to specialist laboratories for expert analysis.

Radiation Management Plan

Management is responsible for ensuring that exposures to radiation are limited, that protection is optimised, and that appropriate radiological protection programmes are developed and implemented. The Radiation Management Plan (RMP) relates to the full life cycle of operations, i.e., from inception to close-out. The objectives of the RMP are to reflect management responsibility for radiation protection through the development of management policies, structures, organisational arrangements, and procedures to ensure the management of the risks.

The basic elements of the RMP are:

- An organisational structure for the allocation of the various levels of accountability, responsibilities, and roles.
- The provision of suitable and adequate resources for protection.
- Arrangements for the measurement of radiation levels at the site and potential exposures of workers and the public.
- The designation of areas where radiation control is required.
- Safe operating procedures and rules, including supervision.
- Maintenance of a data recording and reporting system related to the control of radiation, exposure of workers and decisions on measurements for occupational radiation protection.
- A training programme on radiation hazards and requirements for protection.
- An emergency response plan (mostly in terms of environmental pollution)
- A health surveillance programme.
- Quality assurance (often in line with the ISO or OHSA standards)

16.20. Human Resources

Shift Cycles

It is proposed that two distinct shift cycles will be applied to personnel, one for production related personnel and a second for management, administration, and technical support personnel.

The production personnel include the following:

- Mining.
- Development labour.
- Stoping labour.
- Backfill operators.
- TMM operators.
- Engineering.
- Selected surface, underground and general engineering personnel.

There is no significant town or village within 90 km of the project site, making it impossible for employees to travel to and from the mine. In addition, the local road conditions are poor, making travel hazardous and time consuming. Practically, it will not be possible to commute to the mine on a daily basis and therefore it is proposed that a camp system will be operated. Employees would travel to site to work for a planned period, while being accommodated at the camp before leaving on an extended period of rest.

The proposed shift cycle in this regard for the production personnel will be 10 days on and 5 days off cycle. To allow for a continuous operation, there will be three rotating shifts (Day, Night, and Off shift). The below figures indicates the proposed shift cycle; these shifts will be 12 hours in duration.

	Day 5	Day 10	Day 15	Day 20	Day 25	Day 30
Shift A	D/S			N/S		
			Rest			Rest
Shift B		N/S			D/S	
	Rest			Rest		
Shift C	N/S		D/S			N/S
		Rest			Rest	

Figure 16-33: 10/5 Shift Cycle.

The management and services personnel include all the Management, Technical/Engineering Services and Administrative personnel. The shift cycle for the management and services personnel is based on an 11-day / Fortnight shift cycle (normal working week of 5 days on, Monday to Friday, and 2 Saturdays per month). These employees would travel into site, to arrive by the Monday morning and leave on Friday afternoon, while being accommodated through the week in the site camp facility. Figure 16-34 below indicates the proposed shift cycle; these shifts will be 12 hours in duration.

		Day 5		Day 10		Day 15
Shift A	On		OFF	On	OFF	On
	Day 16	Day 20		Day 25		Day 30
		OFF	On		OFF	On

Figure 16-34: 11/14 Shift Cycle.

Organogram and Manpower Complement

A high-level organogram which is shown in Figure 16-35, illustrates the overall structure for all the Dasa operations and areas. The mining labour structure is indicated by the orange block and is shown in more detail in Figure 16-36. The posts highlighted in green indicate the mining and engineering positions, which will be on the 10/5 shift rotation.

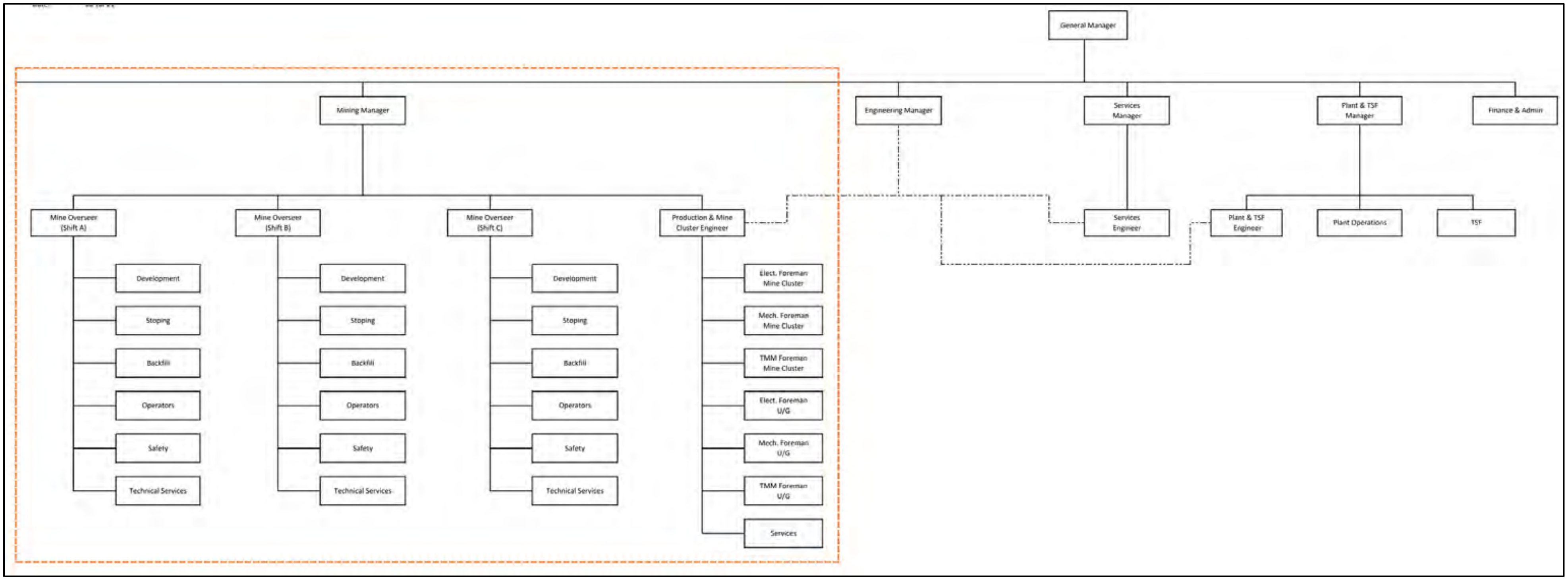


Figure 16-35: High Level Mine Wide Organogram.

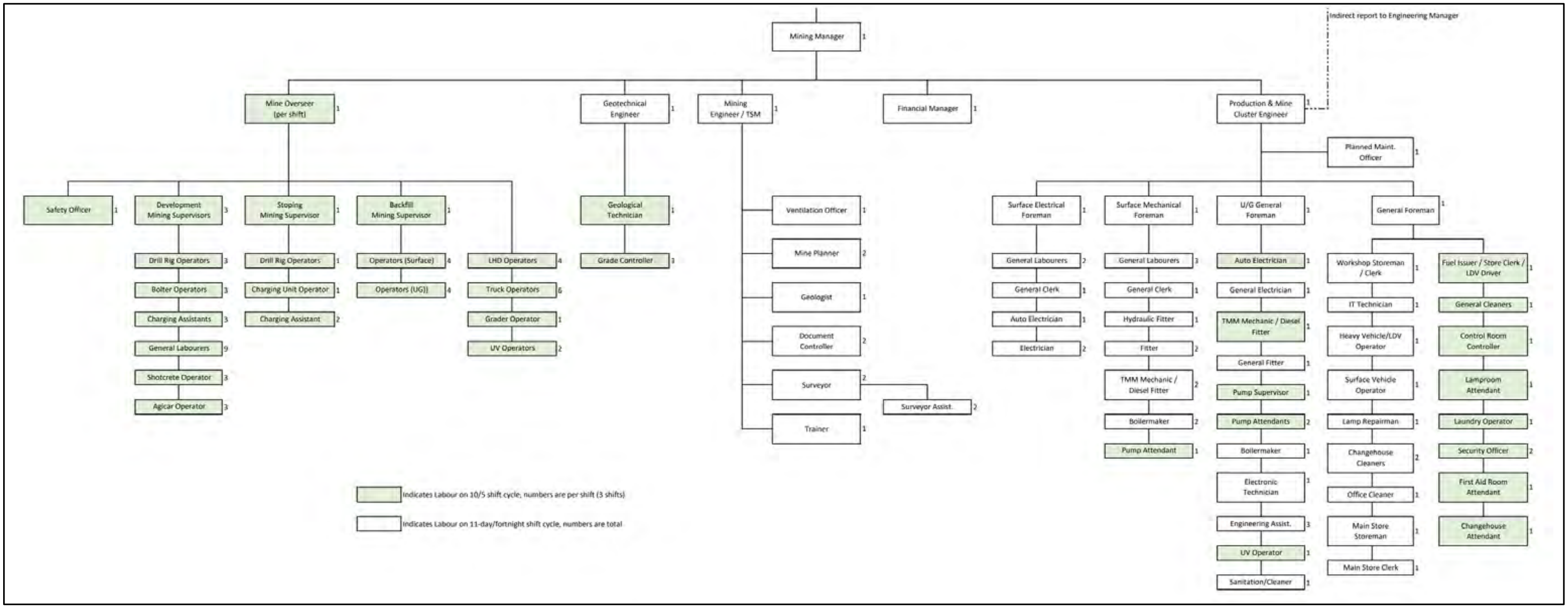


Figure 16-36: Mining Labour Focused Organogram.

Based on the shift cycles and the proposed organogram, a total steady state labour complement was generated, this is summarised in Table 16-30 below.

Table 16-30: Mining Manpower Complement.

Manpower Area	Complement
Total Mining Manpower Complement	**284**
Mining Manpower	**193**
Supervision	**8**
Site Manager	1
Financial Manager	1
Mine Overseer	3
Safety Officer	3
Development	**81**
Mining supervisor	9
Drill rig operator	9
Bolter operator	9
Charging assistant	9
General labourer (services)	27
Shotcrete operator	9
Agicar operator	9
Stoping	**15**
Mining supervisor	3
Drill rig operator	3
Charging Unit Operator	3
Charging assistant	6
Backfill	**27**
Mining supervisor	3
Backfill operators (surface)	12
Backfill operators (UG)	12

Manpower Area	Complement
Load and Haul	**39**
LHD operators	12
Truck operators	18
Grader operator	3
UV operator	6
Technical services	23
Geologist	1
Geological technicians	3
Grade controllers	9
Surveyor	2
Survey assistants	2
Geotechnical engineer	1
Mining Engineer/TSM	1
Mine planner	2
Trainer	1
Ventilation officer	1
Engineering Surface	**24**
Resident Engineer	1
Planned Maintenance Officer	1
Engineering Foreman - Electrical	1
Engineering Foreman - Mechanical	1
Auto Electrician / Technician	1
Electrician	2
Hydraulic Fitter	1
Boilermaker	2
TMM Mechanic / Diesel Fitter	2
Fitter	2

Manpower Area	Complement
General Labour - Surface and Workshops	5
General Clerk	2
Pump attendants	3
Engineering Underground	27
General Underground Foremen	1
Auto Electrician / Technician	3
Electricians	1
TMM Mechanic / Diesel Fitter	3
Fitter Production	1
Pump Supervisors	3
Pump Attendants	6
Boilermakers	1
Electronic Technicians	1
Engineering Assistants	3
UV Operator	3
Sanitation / Cleaner	1
Engineering General	40
General Foremen	1
Workshop Storemen/clerks	1
Computer / IT Technician	1
Heavy Vehicle Driver / Operator	1
Surface vehicle drivers	1
Lamp Room Repairman	1
Change House Cleaners	2
Office Cleaner	1
Main Store Storeman	1
Main Store Clerk	1

Manpower Area	Complement
Fuel issuers / Stores Clerk / LDV driver	3
General Cleaners	3
Laundry Operator	3
Controller - Control Room	3
Lamp Room Attendant	3
Change House Attendant	3
Security Officer	6
First Aid Room Attendant	3
Document Controllers	2

The steady state labour complement broken down per shift, is summarised in Table 16-31 below.

Table 16-31: Labour Complement Per Shift.

Shift	Management & Services	Production	Total
Day Shift	56	76	132
Night Shift	0	76	76
Rest Shift	0	76	76

The labour complement builds up over a period of time, based on the mining production schedule. Figure 16-37 shows the labour build-up per month over the life of mine.

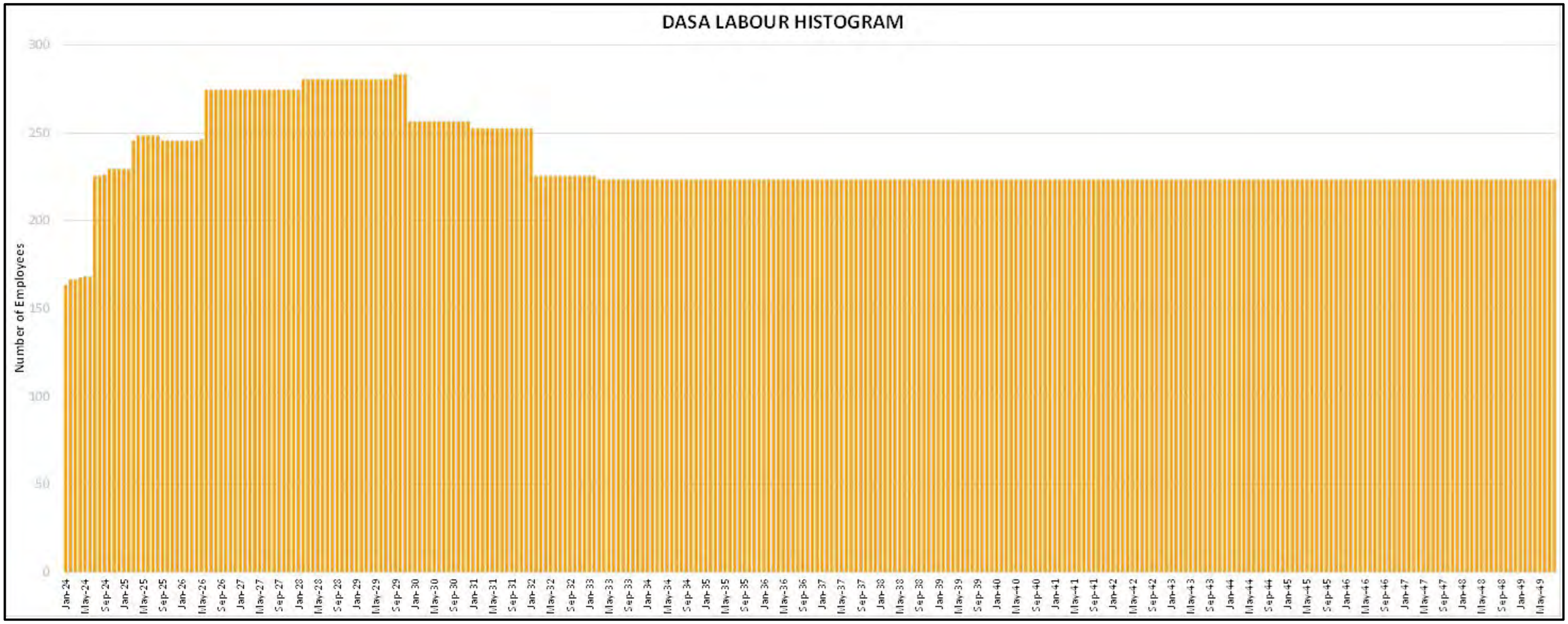


Figure 16-37: Dasa Labour Histogram.

17. RECOVERY METHODS

17.1. Introduction

This chapter will detail the development of the design for the process plant and includes the optimisation that was achieved through the various process trade-off studies in the finalisation of a process plant that can economically deliver high quality uranium products with good recovery for the Dasa Project.

Laboratory and pilot plant test work discussed in section 13 of this report confirmed the process design parameters for the Dasa Plant. The process route selected is similar to Orano operations at Arlit in Niger processing a similar ore type.

The pugging and curing process has been successfully applied at the Cominak and Somair operations. It utilises the addition of high strength lixiviant and oxidative chemicals (sulphuric acid and nitric acid) to create aggressive reaction conditions in a low moisture feed, which promotes the uranium leaching characteristics with limited dissolution of undesirable gangue elements like silica which would otherwise have negative downstream process implications.

The design development followed a logical design sequence in developing the extraction plant namely:

- Process design basis.
- Block flow diagram.
- Mass balance.
- Mechanical equipment list.
- Process flow diagrams.
- Piping and instrumentation diagrams.
- Process control philosophy.
- Water balance.
- Process Description.

To optimise the plant design, trade off studies were completed covering dry milling, final product precipitation reagents and leach tailings disposal options.

The plant design is currently in execution and the design has been optimised with detailed vendor information. The execution design is not complete, and costing is still in progress.

As the updated design is not fundamentally changing the previous plant design remains for the feasibility study and is within the design accuracy required. The capital cost of these items has been escalated using detailed escalation factors and reported under capital.

The only changes to the design relate to the final tailing's solids do not partially reporting to the backfill plant (plant now uses crushed waste backfill which is detailed under infrastructure) and as such all tailings reports to the tailing's storage facility.

In addition, some infrastructure items have changed and are now detailed specifically under infrastructure.

The previous process section remains unaltered for the feasibility study update. Note trade-off financial evaluations have not had their costs updated but keep the 2021 costing unaltered.

17.2. Trade-off Studies

Three trade-off studies were completed as part of process options selection. These included:

- Dry milling processing.
- Final product precipitation.
- Tailing disposal.

The trade-off reports are summarised below.

Dry Milling Options

The aim of this study was to evaluate the most suitable grinding option for the Dasa Project, based primarily on technical feasibility, capital and operating costs and flowsheet complexity.

Mineralogical characterisation of the ore indicated that quartz is the primary gangue mineral and has a coarse grain size. The uranium containing minerals are fine-grained and are hosted within the matrix holding the quartz particles together. The grinding requirement is to liberate the quartz from the matrix without excessive breakage of quartz beyond its natural grain size and to impart a polishing effect on the quartz particle surfaces to liberate the uranium.

The dry grinding options investigated were:

- High Pressure Grinding Rolls (HPGR).
- Vertical Shaft Impactor (VSI).
- Semi Autogenous Grinding (SAG).

Technical feasibility was assessed by identifying the dominant mechanisms used by each technology in breaking down particles.

The capital and operating costs for each option were obtained from technology suppliers. The total net present cost (NPC) for each option was calculated by combining the capital and operating costs over 12-years (the anticipated project duration of Phase 1) discounted at a rate of 8% per annum.

High level layout drawings of the flowsheet for each technology were used to evaluate flowsheet complexity.

A weighting method was used to rank the technology options based on technical risk, total cost (present value CAPEX and OPEX), flowsheet complexity, speed and ease of construction, technology maturity and maintenance requirements.

The weighted scoring of the technology options has been summarised in Table 17-1,below.

A scoring method using a points system between 0 and 5 was used for each criterion as follows:

a. 5 = best score in the criteria.

b. 3 = medium score in the criteria.

1 = worst score in the criteria.

Table 17-1: Weight Scoring and Ranking Methodology.

Feature	**Weight**	**Semi-Autogenous Mill (SAG)**		**High Pressure Grinding Rolls (HPGR)**		**Vertical Shaft Impactor (VSI)**	
		Score	**Total**	Score	**Total**	Score	**Total**
	(0 - 10)	(0 - 5)	W × S	(0 - 5)	W × S	(0 - 5)	W × S
Technical Risk	10	5	**50**	1	**10**	3	**30**
Combined CAPEX and OPEX	5	3	**15**	1	**5**	5	**25**
Consequential Flowsheet Complexity	8	5	**40**	3	**24**	3	**24**
Ease of Construction	3	3	**9**	3	**9**	5	**15**
Technology Maturity	3	5	**15**	1	**3**	3	**9**
Maintenance Requirements	3	3	**9**	3	**9**	5	**15**
Weighted Total Score			**138**		**60**		**118**

The evaluation indicated that:

- The ability to minimize quartz breakdown was a concern with HPGR and VSI technologies which predominantly use impact techniques for size reduction. The SAG offers greater control flexibility to minimize overgrinding of quartz as well as promote polishing effects on coarse quartz particles.
- The total net present cost (NPC) was lost lowest for the HPGR and highest for the VSI.
- The SAG offers the simplest flowsheet in terms of layout and number of unit operations.
- The use of primary crushing followed by SAG milling technology is a proven technology in the Republic of Niger.

The semi-autogenous (SAG) mill achieved the highest score and was the recommended comminution technology option for the Dasa flowsheet.

Final Product Precipitation Options

The aim of this study was to assess the most cost-effective precipitation reagent option for the Dasa Project's final product.

The precipitation reagents considered were as follows:

1. Magnesia.
2. Ammonia.
3. Hydrogen peroxide.

From the test work completed, hydrogen peroxide precipitation produced a product with the highest uranium grade as summarised in Table 17-2, below.

Table 17-2: Precipitation Products Quality from Test Work with Various Reagents.

Description	Magnesia (10% m/m)	Ammonia (26 %m/m)	Hydrogen Peroxide (50 %m/m)
Precipitation Method	Bench Scale	Bench Scale	Bench Scale
U, %m/m	52	60	76
Filtration Rate (time)	slow	very slow	extremely fast
Particle Size	fine	fine	coarse

Precipitation Reagent Cost and Consumption

The reagent costs and consumptions are summarised in Table 17-3, below. The reagent costs are landed costs at Dasa site in Niger. The consumptions were calculated based on the required stoichiometric amounts with an excess amount added to ensure sufficient reagent to complete the reactions. Hydrogen peroxide is used in combination with sodium hydroxide to achieve precipitation conditions while the other reagents are used on an individual basis. The total operating cost of the peroxide precipitation option is $0.30/lb which is higher than the other two reagent

Table 17-3: Costs and Consumption of Reagents.

Reagent	Reagent Price, $/t	Reagent Consumption, kg/kg U_3O_8	Operating Cost, $/lb U_3O_8
Magnesia	2 932	0.165	0.21
Ammonia	2 920	0.209	0.28
Hydrogen Peroxide	1 912	0.201	0.17
Sodium Hydroxide	994	0.285	0.13

Final Product Packaging and Shipping Costs

- Packaging and shipping costs were estimated based on a dried yellow cake product.
- Final product to be transported from Niger to converter plants in France, Canada, or the United States. Packaging to be done in 210 L steel drums.
- 20 ft containers to be used for shipping.
- Each container to accommodate 40 drums.

Table 17-4: Final Product Packaging and Transport Costs.

Description	Magnesia	Ammonia	Peroxide
Uranium Content, U_3O_8 % m/m	65.1	69.7	89.8
Product Tonnage, t/a U_3O_8	2 754	2 572	1 993
Bulk Density, t/m^3	1.7	1.7	2.4
Drum Volume, L/drum	210	210	210
Product Mass, kg/drum	357	357	504
Number of Drums/a	7 713	7 204	3 961
No of Drums/Container	40	40	40
Number of Containers/annum	193	180	99

Because of the higher grade of U_3O_8 in peroxide product, the requirements for packaging and shipment are lower than for magnesia and ammonia precipitated products.

Production of high-grade yellow cake concentrate is one way of reducing shipment costs which can be achieved through precipitation reagent selection.

Precipitation Reagents Overall Cost Comparison

Table 17-5: Combined Annual Costs Comparison.

Cost Item	Magnesia ($)	Ammonia ($)	Peroxide ($)
Reagent	867 784	1 095 717	1 195 030
Packaging and Transport	2 410 406	2 458 088	1 351 424
Total	**3 278 190**	**3 347 043**	**2 432 778**
Cost Ratio	135%	138%	100%

The overall cost of the peroxide system is at least 30% lower in comparison to either magnesia or ammonia and is recommended for the Dasa Project.

Tailings Disposal Options

The purpose of this trade-off study was to evaluate the most cost-effective tailings disposal method between dry stacking and wet tailings disposal.

The evaluation was based on estimating the net present cost (NPC) by calculating the total capital and operating costs over a life of mine of 12-years discounted at a rate of 8% per annum.

A high-level study for the equipment, installation and operating costs for the dry and wet tailings disposal systems was completed and considered as follows:

- Dry stacking by
  - Conveyor.
  - Trucking.
- Wet disposal by piping

Capital costs were based on mechanical equipment for each option. Budget cost estimates were obtained from suppliers. Steel, piping, and installation costs were estimated by calculation. Operating costs were estimated based on power consumption and maintenance requirements.

Conveyor System (Dry Stacking)

The following assumptions were made for costing a conveyor system for transferring dry tailings from the plant to the TSF:

- Distance from plant to TSF ~ 1000 m.
- Tailing tonnage ~ 500 tonnes/day.
- Tailing solids content ~ 80% m/m solids.
- Maintenance cost per annum ~ 2.0% of capital cost.
- Life of mine ~ 12-years.
- Discount rate ~ 8.0%.

The total installed cost for the conveyor system and the annual operating costs have been summarised in Table 17-6 and Table 17-7 respectively below.

Table 17-6: Conveyor System Capital and Installation Costing.

Item	Description	Cost ($)
1	Steelwork and Painting	190 944
2	Head + Take-up Structure	10 471
3	Stacker structure (Grasshopper)	10 471
4	35° Carry Idlers (1.2 m spacing)	52 595
5	20° Carry Idlers	126
6	Training Idlers	5 075
7	Return Idlers (3 m spacing)	25 988
8	Belting - Class 315 3 ply "N"	62 894
9	Drive - 18,5 kW / 37 kW	9 677
10	Pulleys - head, tail, bend, and take-up	9 078
	Total Supply Cost	**377 319**
11	Installation – including Preliminary & General Costs	205 148
	Total Installed Cost	**582 467**

Table 17-7: Conveyor System Operating Cost.

Item	Description	Cost Per Annum ($)
1	Power	3 385
2	Maintenance	7 546
	Total Operating Cost	**10 931**

The total NPC (CAPEX and OPEX) for this option has been estimated at $664 843 for the life of mine.

Piping System

In this option tailings will be transferred from the plant to the TSF by pumping. A key requirement for this option is that the tailings discharged from the horizontal belt filter (at 80 % m/m) be repulped to 50% m/m solids by the addition of water.

The following assumptions were made for costing:

- Distance from plant to TSF ~ 1000 m.
- Tailing tonnage ~ 500 tonnes/day.
- Required tailings solids content~ 50 % m/m solids.
- Distance from plant to water makeup borehole ~ 2000 m.
- Maintenance cost per annum ~ 1.5 % of capital cost.

The total installed cost and the annual operating costs for the piping system have been summarised in Table 17-8 and Table 17-9 respectively, below

Table 17-8: : Piping System Capital and Installation Cost Estimate.

Item	Description	Cost ($)
1	20 m³ SS Cylindrical Tank	25 462
2	SS Agitator	3 885
3	Weir 3/2 AH-WRT Pumps - 30.0 kW Each	37 116
4	SAM CPO 3 × 1.5 × 11 Pumps - 5.5 kW Each	15 743
5	KSB UPAC 150-030/19EE - 5.5 kW Each	23 732
6	Tailings Line - 110 mm OD HDPE	7 111
7	Return Water - 125 mm OD HDPE	9 190
8	Borehole Water - 140 mm OD HDPE	30 141
9	Borehole Sinking Cost	100 000
	Total Capital Cost	**252 380**
10	Installation and P&G's Costs	137 219
	Total Installed Cost	**389 599**

Table 17-9: Piping System Operating Cost Estimate.

Item	Description	Cost Per Annum ($)
1	Power	15 868
2	Maintenance	3 786
	Total Operating Cost	**19 654**

The total NPC (CAPEX and OPEX) for the piping system option has been estimated at $537 713 for the life of mine.

Mobile Equipment (Dry Stacking)

In this option tailings will be transferred from the plant to the TSF by trucking. Tailings discharged from the Horizontal Belt Filter (HBF) will be stored on a stockpile from which a loader and truck will load and haul the tailings to a TSF.

The following assumptions were made for costing:

- Distance from plant to TSF ~ 1000 m.
- Tailing tonnage ~ 500 tonnes/day.
- Required tailings solids content~ 80 % m/m solids.
- Maintenance cost per annum ~ 1.5 % of capital cost.

The total installed cost and the annual operating costs for the piping system have been summarised in Table 17-10 and Table 17-11 respectively.

Table 17-10: Trucking Equipment Capital Cost Estimate.

Item	Description	Cost (US$)
1	Front End Loader (DL 300)	183 542
2	Rigid Body Truck	300 612
	Total Capital Cost	**484 154**

Table 17-11: Trucking System Operating Costs Estimate.

Item	Description	Cost Per Annum (US$)
1	Fuel and services	58 619
2	Spares parts	11 244
	Total Operating Cost	**69 863**

The total NPC (CAPEX and OPEX) for tailings disposal by the trucking option has been estimated at

$1 010 645 for the life of mine.

Piping equipment offers the least NPC over the life of mine period considered.

To complete the evaluation, the establishment of the tailing storage facilities was also added to the overall cost.

Tailings Storage Facility Establishment Costs

The capital and operating cost estimation for the establishment and operations of the dry versus wet tailings facilities was completed by Epoch Resources. Complete details of the trade-off study are contained in the report "Dasa Uranium Project – Tailings Storage Facility Site Selection and Trade-off Study" by Epoch Resources.

Capital and operating costs for the dry stacking and wet disposal TSF options were extracted from the Epoch report and are summarised in Table 17-12 below.

Table 17-12: Total Net Present Cost of Each Option for the TSF (from Epoch email summary).

Net Present Cost Elements	Wet Tailings ($)	Dry Stack (conveyor) ($)	Dry Stack (Truck) ($)
Capital Cost (Life of Mine)	12 050 000	5 480 000	5 480 000
Operating Cost	2 190 000	4 380 000	7 670 000
Total	**14 240 000**	**9 860 000**	**13 140 000**

The large difference in TSF capital cost between dry and wet tailings is due to the large difference in wall volumes required.

- Wet tailings TSF require full containment walls within which the slurry is pumped.
- Dry-stack TSF require much smaller perimeter bund walls as the dry tailings can extend well above the outer containment walls.

Overall Tailings Disposal Cost Summary

Table 17-13, below, presents the total net present cost estimates for tailings transfer equipment systems and storage facilities for the options over life of mine.

Table 17-13: Total Net Present Cost for the TSF and Equipment Systems (CAPEX and OPEX) Over Life of Mine.

NPC Element	Wet Tailings, Piping ($)	Dry Stack, (Conveyor) ($)	Dry Stack (Truck) ($)
Tailings Storage Facility	14 240 000	9 860 000	13 140 000
Equipment Systems	537 713	664 843	1 010 645
Total	**14 777 713**	**10 524 843**	**14 150 645**

Over the life of mine operation dry stacking by conveyor has the least total net present cost.

The dry tailing stacking method requires a lower start-up capital compared to wet tailings disposal. The initial capital requirement for the dry stacking by conveyor versus stacking by truck was similar.

Based on the results of this evaluation, dry stacking of tailings is the most favourable and is recommended for the Dasa Uranium Project.

17.3. Process Design

The Dasa Plant has a design for an ore treatment of 365,000 dry t/a based on a plant availability of 86%. The process design criteria are summarised in Table 17-14, below.

Table 17-14: Dasa Plant Process Design Criteria Summary.

Description	Units	Value
Ore Throughput	T/A, Dry	365,000
Operating Schedule		
Crusher Availability	%	60
Plant Availability	%	86
Plant Throughput – Daily Average	T/D, Dry	1,000
Plant Capacity – Design	T/H, Dry	50
Uranium Grade – Design	%	0.45
Overall Uranium Recovery	%	94.15
Crushing (Single Stage)		
Primary Crusher	Type	Jaw Crusher
Run of Mine (RoM) Feed Size, F_{100}	Mm	500
Crushing Product Size, P_{100}	Mm	250
Crushing Product Size, P_{80}	Mm	80
Crushed Ore Feed Moisture	% M/M Moisture M/M	7
Ore Feed Drying		
Inlet Temperature	°C	450
Outlet Temperature	°C	120
Drying Air Required	Nm^3/H	35,000
Grinding		
Circuit Type		Dry SAG Milling with Screens
Semi-Autogenous (SAG) Mill Feed Size (F_{100})	Mm	250
SAG Mill Feed Size, F_{80}	Mm	80
SAG Mill Product, P_{95}	Mm	0.6
Pug Leaching		

Description	Units	Value
Pug Leach	% M/M Solids	85
Pug Drum Residence Time	Mins	10
Curing Time	Hr	3
Belt Filtration		
Filter Flux	$T/H/M^2$	0.504
Flocculant Consumption	G/T	150
Coagulant Consumption	G/T	25
Solvent Extraction		
Circuit Configuration	Number of Stages	4 X Extraction – 3 X Scrub – 3 X Strip
Alamine 336 – Extractant	% V/V	8.7
Exxsol D80 – Diluent	% V/V	84.4
Exxal 13 – Modifier	% V/V	7.0
Raffinate Bleed	% Bleed	52% V/V
Sodium Di-Uranate Precipitation		
Number of Tanks	No	5
Temperature	°C	60
Residence Time	H	5
Thickener Flux	$T/M^2/D$	0.089
Belt Filter Flux	$T/M^2/D$	0.086
Uranyl Peroxide Precipitation		
Number of Tanks	No.	5
Residence Time	H	6.5
Thickener Flux	$T/H/M^2$	0.069

Utilisation and Buffer Capacities

The Dasa Plant has a capacity to treat 365,000 t/a of run-of-mine (RoM) at a head grade of 0.45% U_3O_8. The RoM is supplied from an underground mine operation with a life of mine of close to 24-years.

Equipment Utilisation

Utilisation is a factor which affects the operating time of the plant unit processes. Most milling operations can achieve a utilisation greater than 90%. Dry milling along with dry screening and hot air blowers result in a need for more frequent equipment checks and maintenance. In the case of the Dasa Plant, the main power supply is from Sonichar, a 40-year-old coal fired power station. A visit to the power station as part of the feasibility study, raised some concerns with operational reliability. With the plant reliability factors along with the concerns of power stability means a design utilisation of 86% has been chosen. Options to improve the electric power reliability will be investigated to increase the overall plant utilisation.

Good operations have an opportunity to achieve a higher operating time creating potential to mill additional tonnage and create additional revenue if these are realised.

Process Buffer Capacities

Buffer capacities are located within the process flowsheet as follows:

RoM Pad Stockpiles

RoM is hauled from the mine and tipped on the RoM pad. The RoM pad is designed to accommodate sufficient RoM to run the plant for at least 7 days. This storage capacity also provides time for excess ore moisture to be evaporated by the sun, reducing the moisture content of feed to the plant.

SAG Mill Stockpile

The crushed material is conveyed to a stockpile that provides approximately 8 hours of live storage capacity ahead of the SAG mill. The residence time of the stockpile permits opportunity to perform maintenance on the crushing circuit without affecting the rest of the process plant.

Pug Drum Feed Bin

A bin with a 3-hour storage capacity is provided for in the design between the SAG mill and the pug drum to allow for controlled feeding of the leach pug drum. This storage bin also assists in accommodating routine maintenance inspections which are necessary on the screens without affecting feed to the pug drum.

Pregnant Leach Solution Storage Tanks

Two 500 m^3 tanks with a 24-hour pregnant leach solution (PLS) storage capacity were provided for in the design. The PLS storage residence time allows for maintenance to be performed on the belt filters without affecting the solvent extraction (SX) and downstream process operations. These also provide stabilised solution flows to the SX allowing more accurate control and recovery in the SX area.

Block Flow Diagram

A block flow diagram of the process flowsheet for the Dasa Plant is represented in Figure 17-1, below.

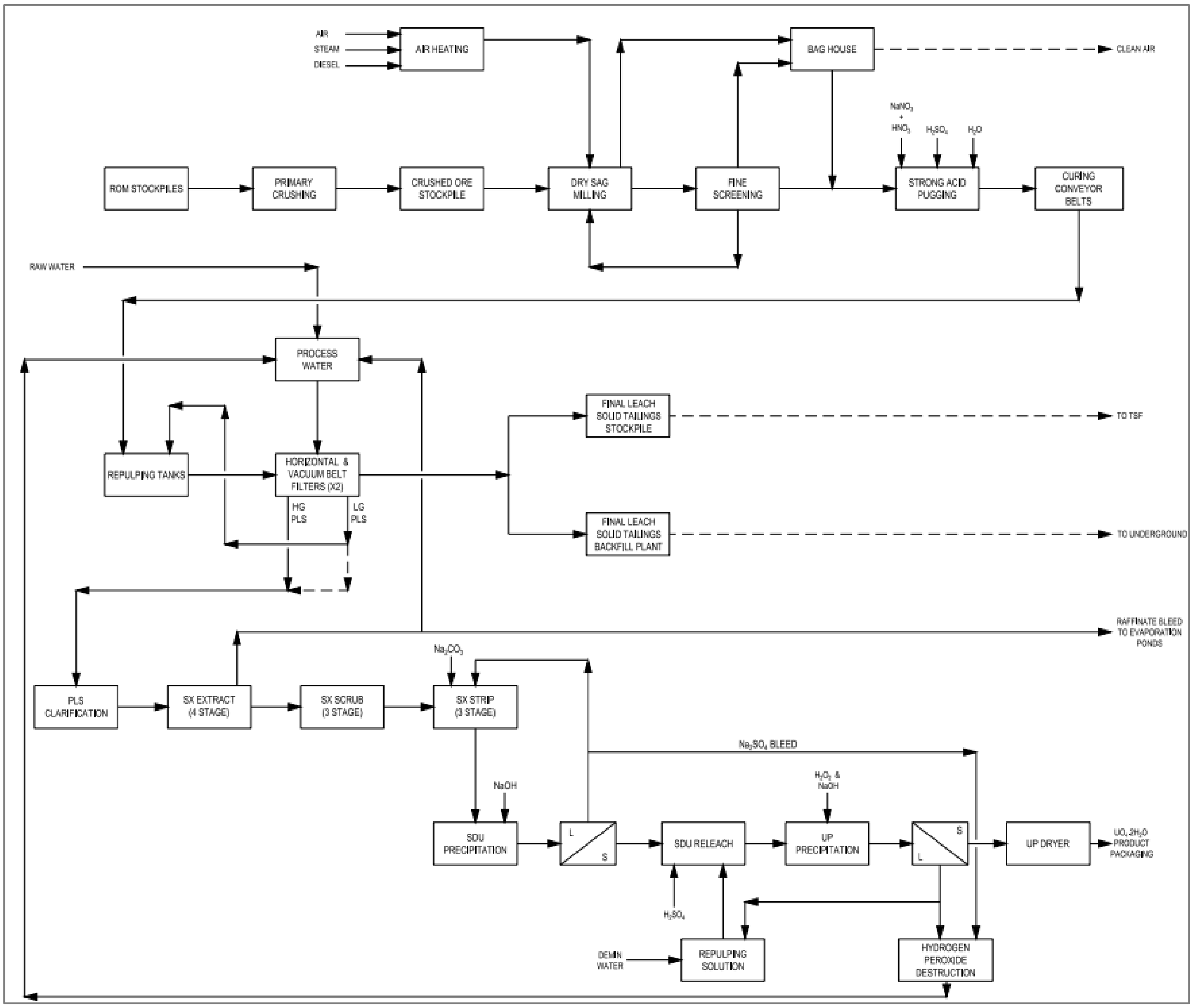


Figure 17-1: Process Block Flow Diagram.

Process Flow Diagrams

The block flow diagram was developed into a set of process flow diagrams (PFD's) which was used to develop the mass balance and mechanical equipment list (MEL). The PFD's were used to develop a preliminary equipment list and P&ID's.

Mechanical Equipment List

A mechanical equipment list was developed from the process flow diagrams and mass balance. A summary of the major equipment is presented in Tale 17-15, below.

Table 17-15: Summary of Process Plant Major Equipment.

Item	Equipment Name	Sizing/Description	Number Required
1	Feed Bin	Volume 40 m^3	1
2	Jaw Crusher	69 t/h design throughput, RoM size F100 ~ 500 mm, P100 ~ 250 mm, P_{80} ~ 80 mm	1
3	Crushed Ore Stockpile	8-h live capacity	1
4	SAG Mill	50 t/h design throughput, 4.88 m (D) × 1.83 m (L), F_{80}~ 80 mm	1
5	Screens	Dry high frequency sizing screens, Final Product P_{95} ~ 600 µm	2
6	Pug Drum	60 t/h throughput, F_{95} ~ 600 µm, 10 mins residence time	1
7	Curing Conveyors	2 × 135 m long conveyors, 2.4 m wide, 0.025 m/s velocity	2
8	Nitric Acid Regeneration Plant	1.25 t/h, 40 % m/m concentration	1
9	Repulp Tanks	20 m^3 live capacity tanks, 50% m/m solids, 21 minutes residence time, mechanically agitated, + 1 × 30 m^3 live capacity filter feed tank	4 + 1
10	Belt Filter	75 m^2 belt filters, 3 wash stages, 50% m/m solids feed slurry	2
11	Pin Bed Clarifier	3.5 m diameter	1
12	SX Mixer-Settler	15.7 m^3 effective volume mixer tank, 79.1 m^2 effective settler area, 10 × mixer settler tanks	1
13	SDU Precipitation	10 m^3 live capacity tanks, mechanically agitated	5
14	UP Precipitation	6 m^3 live capacity tanks, mechanically agitated	5
15	Product Drying and Packaging	250 kg/h uranium yellow cake drying and packaging plant	1

Piping and Instrumentation Diagrams

The process flow diagrams (PFDs) were developed into a set of piping and instrumentation diagrams (P&ID's). P&ID's are one of the primary design documents used to determine piping valves and instrumentation designs as well as finalising the mechanical equipment details.

Water Balance

The process plant is the main user of water on site. The process plant raw water consumption is approximately 41.0 m^3/h, equivalent to 0.82 m^3/t of ore processed. Some of this water would be in the form of demineralised water for use in SX and concentrate purification.

The balance of the water is consumed by various services which include dust suppression, offices, acid plant and others. Water consumed by mining operations and the camp facility is excluded from this calculation.

Table 17-16: Raw Water Balance for Process Plant.

Item	Parameter	Units	Value
1	Pug Leaching including water in Reagents	m^3/h	4.2
2	Belt Filtration Tails Washing & Flocculants	m^3/h	32.0
3	Solvent Extraction & Precipitation Processes	m^3/h	4.8
	TOTAL (for process plant)	m^3/h	41.0
4	Services (dust suppression, acid plant etc.)	m^3/h	23.1
	TOTAL (plant and Services)	**m^3/h**	**64.1**

The following issues have an impact on water consumption:

- The re-use of raffinate solution from the solvent extraction circuit is limited to approximately 48% due to the build-up of metals in solution which increases the solution density impacting the belt filter performance. The raffinate solution bled from the circuit is replaced with raw water.
- The bleed solution from the SDU circuit and thickener overflow solution from the UP circuit are combined and utilised as top-up process water for leach tails filters washing requirements.

17.4. Process Plant Description

RoM Pad

Haul trucks will deliver Run-of-Mine (RoM) ore to the RoM pad where it will either be direct tipped to the RoM bin or dumped onto blending stockpiles. A front-end loader (FEL) will be used to reclaim and tram ore from the various stockpiles to the RoM bin. Ore will be blended to maintain a relatively constant feed grade, acid consumption, and filtration characteristics to the process plant.

Crushing Circuit

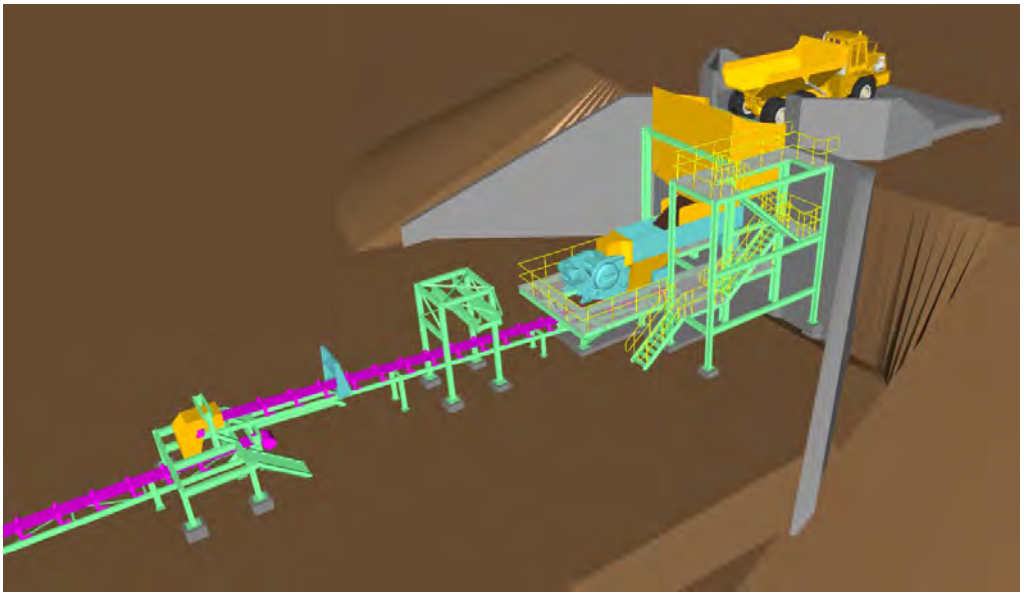


Figure 17-2: Typical Crushing Circuit.

The primary crushing circuit consists of a single tip with a dedicated RoM bin and a single jaw crusher in open circuit. Primary crusher product reports to the Crushed Ore Stockpile (COS). The RoM ore (F_{100} 500 mm) is loaded into a 70 t RoM bin by means of a front-end loader (FEL), or by direct tipping by haul trucks.

A static grizzly will be fitted to the RoM bin to protect the downstream equipment from oversize material and will be inclined for easy removal of oversize to minimise stoppages.

The RoM ore is drawn from the RoM bin at a controlled rate by a single, variable speed vibrating grizzly feeder, and fed directly to the jaw crusher. The speed of the vibrating grizzly feeder is controlled to maintain crusher throughput. Undersize material from the vibrating grizzly feeder reports to the primary crushing conveyor, where it is combined with the primary crusher product (P_{100} 250 mm, P_{80} 80 mm). The primary crushing conveyor is fitted with a belt magnet to remove any tramp iron material. The primary crushing conveyor discharges the crushed material onto the COS.

Dry Grinding and Classification

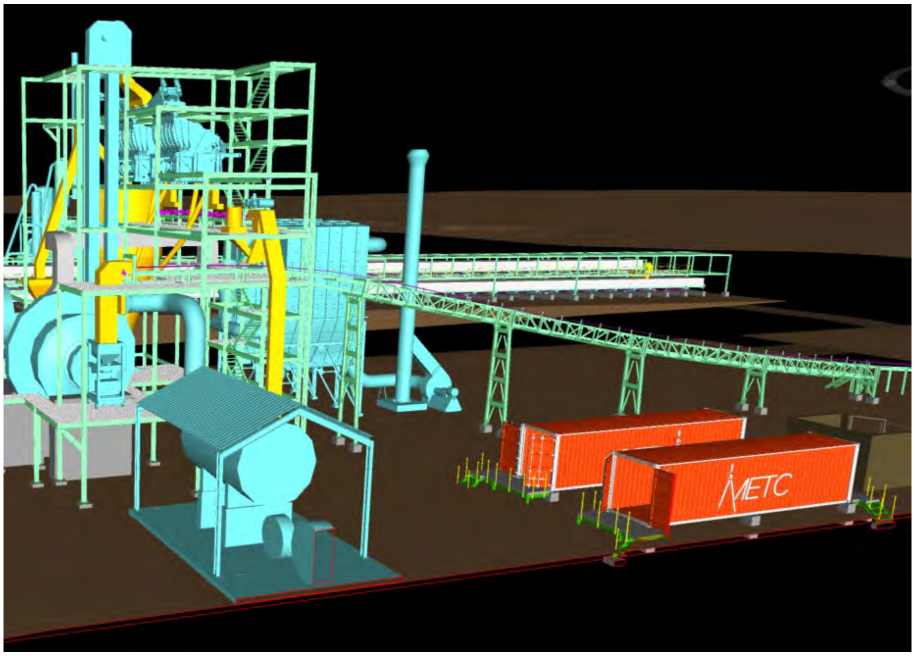


Figure 17-3: Typical Grinding and Classification Circuit.

The milling circuit is configured as a dry SAG mill in closed circuit with dry screens. Ore is withdrawn from the COS by vibrating feeders feeding onto the SAG mill feed conveyor. A weightometer indicates the instantaneous and totalised crushed ore mill feed tonnage and is used to control the SAG mill feed rate via the vibrating feeders. The SAG mill feed conveyor discharges directly into the SAG mill feed hopper. The SAG mill discharge is conveyed to the sizing screens via a bucket elevator conveyor system.

The crushed ore will be ground dry in an air heated semi-autogenous (SAG) mill to a product P_{95} 600 µm. The SAG mill is in closed circuit with dry sizing screens. The screened ore oversize (>600 µm) is recycled to the mill. The <600 µm material from the screens, together with dust recovered from a bag filter system (dust generated during the screening process and the SAG mill), is sent to the leaching process.

The SAG mill is equipped with a variable speed drive to assist with managing variations in feed hardness. When treating soft ore, the SAG mill will be operated at a lower speed and reduced ball charge to minimise overgrinding of the low competency ore. When treating hard ore, the ball mill speed and ball charge will be increased to apply more power to mill the competent ore.

The air heating and drying system feeding the SAG mill will use steam from the acid plant to preheat the air with diesel fed burners providing the final heating requirements.

A ball loading system is used for loading of grinding media into the SAG mill (via the mill feed conveyor).

Pug Leach and Curing

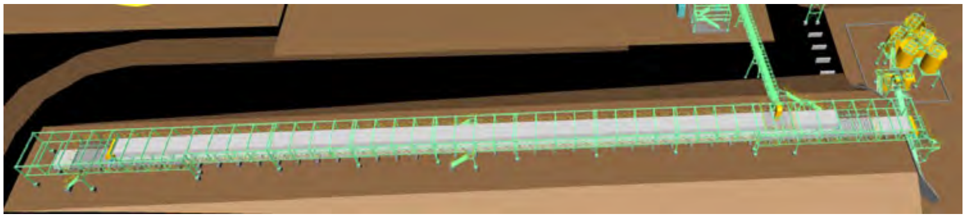


Figure 17-4: Typical Curing and Repulping Circuit.

Pugging is carried out in a revolving pugging drum where sulphuric acid, oxidant (sodium nitrate and nitric acid) and a small amount of water are added to get a total liquid/solids ratio of about 0.15. The retention time in the drum is maintained at 10 minutes as per the test work performance.

During pugging, NO_x gases are released and collected in a column to recover and recycle nitric acid using absorption technology. The pug drum is gas sealed to ensure that the NO_x gases produced during pug leach are collected with minimum dilution and used to regenerate nitric acid in an absorption column.

Curing is carried on two off 2.4 m wide conveyor belts with a total length of 270 m and a belt speed of 0.025 m/s to give a residence time of 3 hours.

After curing and pugging, the dissolution process is carried out in agitated slurry tanks at a solid-to-liquid ratio of 1:1 with low grade pregnant leach solution (belt filter wash solution). The re-pulp tank train consists of five flat bottom mechanically agitated tanks in a series cascade configuration. The tanks are equipped with slurry transfer launder systems that allow any tank to be bypassed to facilitate maintenance.

The final tank is sized to have sufficient capacity to accommodate hydraulic head drainage from the preceding tanks in case of an uncontrolled stoppage.

Solid-liquid Separation

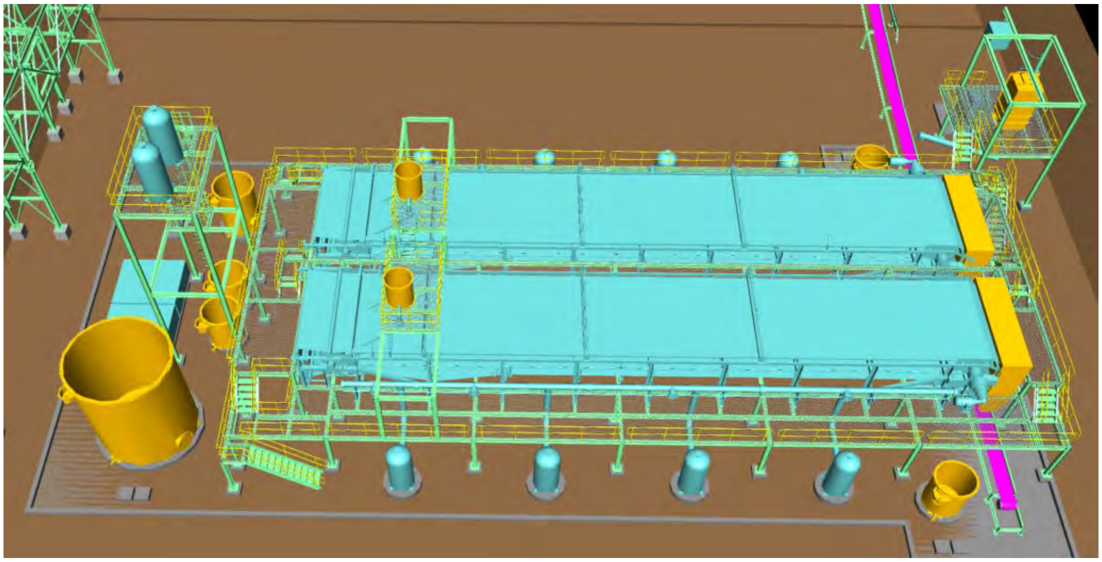


Figure 17-5: Typical Solid Liquid Separation Circuit.

Solid-liquid separation is carried out using two 75 m^2 belt filters operating in parallel. The filtered pregnant leach solution (PLS) is clarified using a pin bed clarifier and is stored in two tanks with a twenty-four-hour storage capacity (500 m^3 capacity each).

Each of the horizontal vacuum filters is equipped with 3 wash stages to limit soluble losses. Raffinate solution topped up with raw water is used for washing tailing solids on the belt filters. The wash solution produced from the belt filters (low grade PLS) is used for re-pulping the solids discharged from the curing belts. Excess low grade PLS is combined with high grade PLS ahead of the clarification stage.

The solids are filtered to approximately 80% m/m solids before discharge onto a tailing conveyor. The tailing conveyor transfers the washed tailing solids to either the backfill plant or onto a tails intermediate stockpile with a 24-hr capacity from which the tails are later reclaimed for final disposal at the Tailing Storage Facility (TSF) by truck haulage. Lime is added to the tailings neutralise residual acid.

To assist with filtration efficiency, a flocculant and coagulant are added to the feed to belt filters.

Solvent Extraction

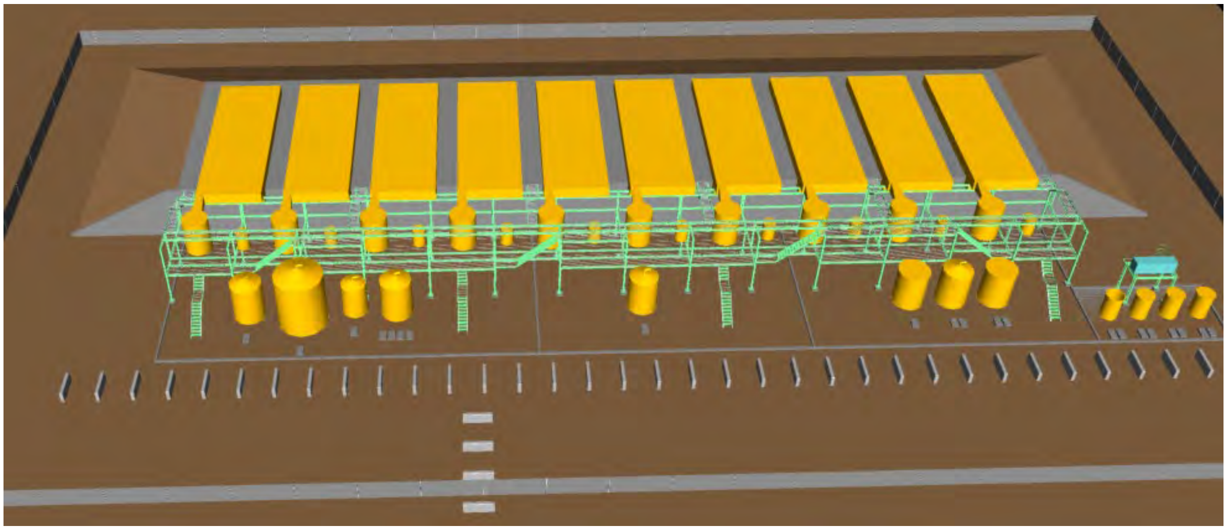


Figure 17-6: Typical Solvent Extraction Circuit.

A conventional solvent extraction (SX) treatment is applied with a tertiary amine dissolved in kerosene for the purification of the uranium solution. Four mixer-settlers are used at the extraction step while stripping of the organic is carried out with a carbonate solution in three mixer-settlers. Transfer of impurities to the stripping stage is minimised by using a scrubbing step with three mixer-settlers.

Laboratory test work indicated that the raffinate solution builds up with impurity metals, which leads to an increase in solution density, which has a negative impact on the operation of belt filters. Thus, at least 50% of the raffinate solution is discharged to evaporation ponds for disposal. The balance of the raffinate solution is topped up with raw water in order to control the wash solution density to the belt filters.

The spent OK liquor (after uranium precipitation) is recycled to stripping.

When there is a crud build up in the settlers, the handling and treatment is done as per need as a batch operation. Crud slurry is pumped from settlers using portable crud pumps to a crud collection tank. Crud is separated from aqueous using a Tricanter. The solids are disposed of while the solutions recovered may be returned to the process.

Sodium Di-uranate Precipitation and Dewatering

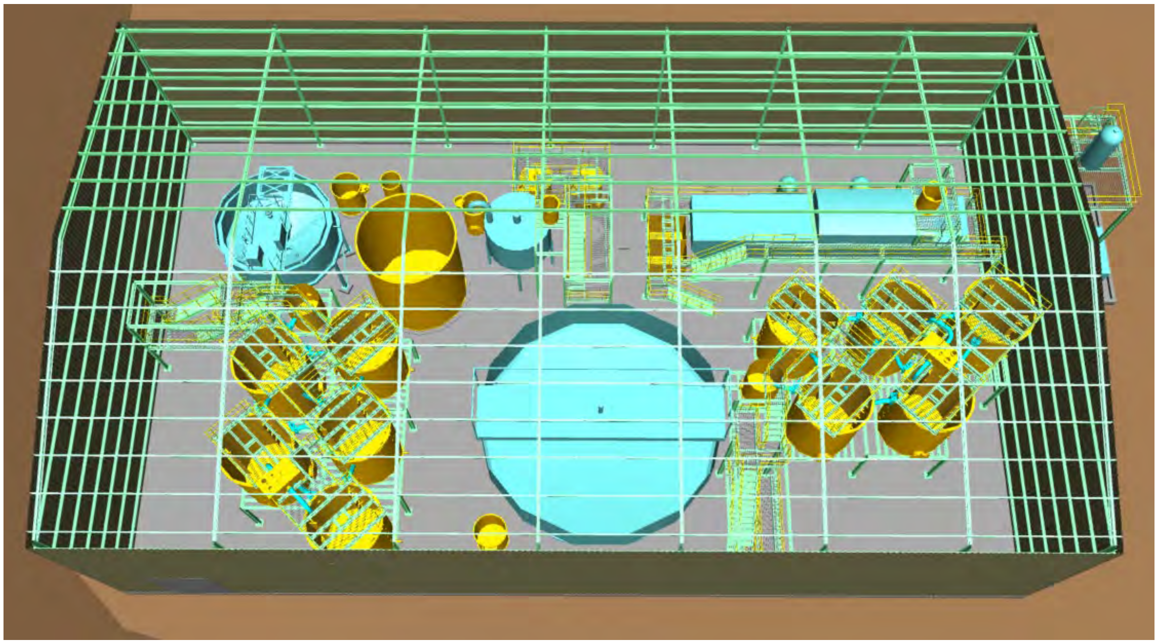


Figure 17-7: Typical SDU Precipitation, UP and Dewatering Circuits.

The uranium is precipitated from the OK solution as sodium di-uranate followed by solid-liquid separation. This route was selected after testing several options for the recovery of uranium from the carbonate strip solution. The precipitation is carried out in 5 tanks in a series arrangement which provide a total residence time of 5 hours. The carbonate solution is steam heated to 60 °C. Caustic soda is added in selected tanks as the precipitation agent.

The precipitation of uranium as sodium di-uranate (SDU) and recycling of the barren liquor to strip provide the following benefits:

- Reduces reagent consumption.
- Improves the water balance.

The barren strip liquor is recycled back to strip after adjustment of the carbonate concentration.

A portion of the spent OK solution is bled out from the circuit to control the build-up of sodium sulphate. As the bled-out solution contains some uranium, it is combined with raffinate solution and used as wash solution on the belt filters.

Uranyl Precipitation and Dewatering

The SDU solids are re-leached with sulphuric acid to give a solution of at least 60 g/L of uranium, which advances to peroxide precipitation.

Hydrogen peroxide was chosen as the precipitation reagent because it is more selective against impurities when compared to ammonia or magnesia. Sodium hydroxide addition is required to neutralize the acid formed during the precipitation reaction, typically carried out at a pH of 3.5.

Uranium precipitation is conducted in five mechanically agitated tanks with an overflow discharge. The tanks are designed with launder systems that allow for any tank to be bypassed if required. The precipitation process is designed with a seed recycle to provide nucleation sites for precipitation and allows the formation of large crystals. The seed recycle is taken from the product thickener underflow to ensure minimum volume of solution is pumped back to seeding, which would result in unnecessarily large precipitation tanks being required.

The resultant uranyl peroxide is dried at approximately between 120 °C and 180 °C in a kiln prior to packaging.

The uranium concentrate required from the Dasa plant is uranyl peroxide ($UO_4.2H_2O$).

The precipitation procedures and methods to be employed for this Project have been extensively used and are standard in modern conventional uranium process operations.

Final Product Drying and Packaging

A fully integrated and proven packaging and drying plant has been specified for the project.

The underflow from the uranyl peroxide thickener is pumped to the final drying and packaging plant. The thickened slurry is centrifuged to produce a dewatered product containing 60% solids. The wet solids are then dried at 120 °C – 180 °C to remove both free and hydrated water.

The uranium drying and packaging plants can be hazardous areas due to potential dust toxicity and dust exposure. The plants are contained in sealed rooms which are maintained under a slight negative pressure which is achieved by a dust extraction fan that is attached to a bag house filter.

Tailings Disposal

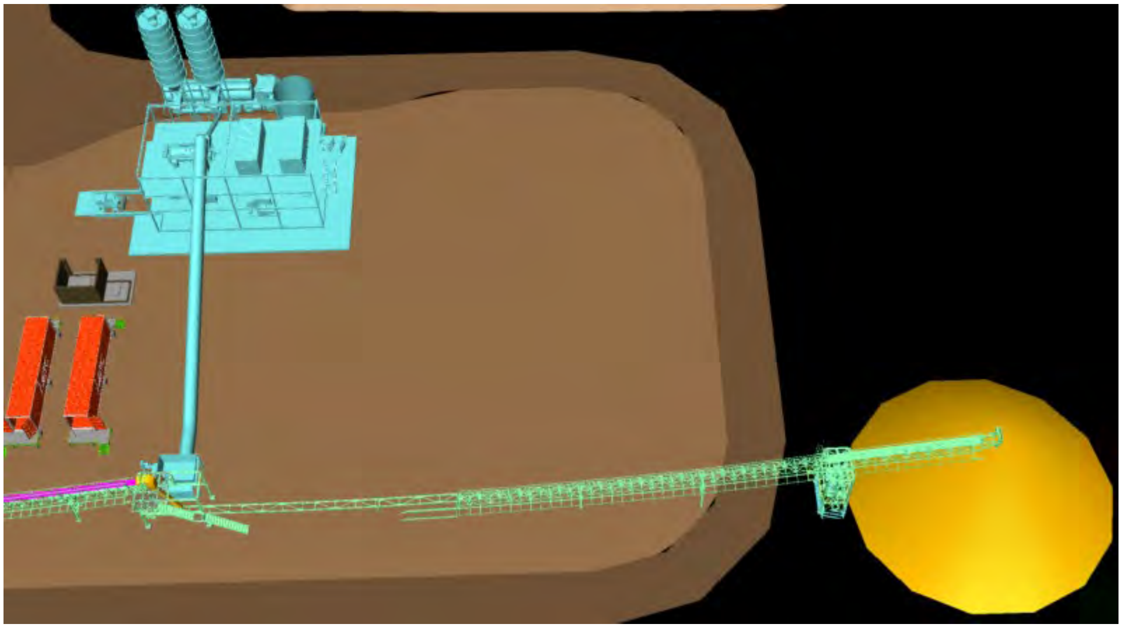


Figure 17-8: Typical Tailings Disposal and Backfill Paste Plant.

A backfill plant is provided to allow for the preparation of a backfill mixture for backfilling the mined-out areas underground. Approximately 50% of the tailings generated will be treated in the backfill plant for backfilling purposes while the balance will be disposed of in the dry solids-based tailings storage facility. The backfill plant has been designed to process the full tailings stream volume.

Services

Raw water for the project will be sourced from boreholes and pumped to a water storage pond on the plant. The raw water pond is a 1 500 m^3 capacity to minimise the impact of short-term supply interruptions. Water will be pumped from the water storage pond to the plant process water tank.

Fire Water

Fire water for the process plant will be drawn from a raw water tank.

The fire water pumping system will contain:

- An electric jockey pump to maintain fire ring main pressure.
- An electric fire water delivery pump to supply fire water at the required pressure and flowrate.
- A diesel driven fire water pump that will automatically start if power is not available for the electric fire water pump or that the electric pump fails to maintain pressure in the fire water system.

Fire hydrants and hose reels will be placed throughout the process plant, fuel storage, and plant offices at intervals that ensure complete coverage in areas where flammable materials are present.

Process Water

Plant process water will consist of raffinate solution from the SX, overflow solution from uranyl peroxide dewatering and bleed solution from sodium di-uranate circuit. Approximately 50% of the raffinate will be sent to the evaporation pond and replaced with raw water to control in plant wash water density.

Potable Water and Demineralised Water

The water treatment plant will consist of clarification through flocculant addition, sand filtration, carbon filtration and biocide dosing. Filtered water will report to the filtered water tank and will be distributed to potable water and demineralised water processing.

Plant and Instrument Air

Plant and instrument air will be supplied from air compressors. The air will be filtered and dried before distribution to separate area specific air receivers which will supply the plant.

17.5. Conclusions

The following conclusions are drawn from the study:

- In Mill Drying Energy Requirements - The energy consumption of ore drying were better clarified and understood in the feasibility study resulting in a higher demand than what had been estimated in the PEA. It is based on 50 t/h ore feed to the SAG mill at less than 7% m/m moisture being dried to less than 0.5% m/m for subsequent processes.
- SX Strip Design - In the PEA study, the SX strip design was based on a sodium chloride strip system, with concerns raised about the inadequate stripping efficiency of uranium from organic during pilot plant Campaign 1. In pilot plant Campaigns 2 and 3, the strip was changed to sodium carbonate, a much stronger stripping reagent with better stripping results and incorporated in the design.
- Production of SDU intermediate product - To improve the water balance and save on reagent consumption, the production of an intermediate sodium di-uranate (SDU) product was considered. This allowed a significant improvement in water consumption along with sodium carbonate, sulphuric acid, and hydrogen peroxide savings.
- Raffinate Bleed - Due to high densities of raffinate solution with poor resultant wash efficiency, approximately 52% of the raffinate solution is sent to an evaporation pond for metal removal.

17.6. QP Comments

The Dasa Plant flowsheet has been investigated extensively and is deemed to be a robust process flowsheet. The circuit relies on proven technology with a low risk of technological flaws. The relevant risks for the plant are as follows:

- RoM moisture content - The plant has been designed to tolerate an ore feed moisture into the SAG mill of up to 7% m/m. It is critical to ensure that the feed moisture is maintained below this. This RoM pad has been designed with at least 7 days holding capacity which provides opportunity for excess ore moisture evaporation.
- Belt Filtration - The belt filters have been designed to handle ore variability as well as minimise uranium losses in solution. This requires limited clays in feed ore, prevention of overgrinding of the ore, careful reagents addition and washing of solid tails on the belt filters with a clean raffinate solution. The belt filters are a critical part of the effective plant operation.
- SDU Process - The SDU process has been operating at Somair successfully since the 1970's. The Dasa Project test work has proven this as a viable process but had identified unfavourable settling and filtration characteristics. New test work has identified superior filtration characteristics when using fluid bed reactor technology which is incorporated in the final design.

- Previous concerns with poor filtration rates associated with sodium di-uranate have also been addressed with the fluid bed reactor technology by Insight prior to filtration which now shows significantly improved filtration rates which could use a smaller filter for this step. The equipment design size has not been reduced for the FS update, so an opportunity exists in execution.

18. PROJECT INFRASTRUCTURE

The Dasa deposit is a greenfield mining Project, with an exploration camp and access road that have been in existence since 2012. Multiple boreholes have been drilled to assess the water quantity and quality. The Dasa Project is located approximately 95 km north of the city of Agadez and 105 km to the south of the town of Arlit. There is no discernible human activity or habitation within the immediate area; however, nomadic herders have been reported.

The site is currently accessed from the main sealed road (N25) via an unsealed sand piste (track) in reasonable repair. There are no waterways or rail networks within close proximity to the Project area and all construction material, equipment and consumables will need to be transported via truck and trailer from ports located in the Gulf of Guinea.

A 132 kV overhead electricity distribution line runs past the Dasa Project site alongside the main sealed N25 road. The construction of a new 5.2 km overhead line will be required to connect the 132 kV line to the new incoming substation located at the proposed processing plant.

In 2021 CSA Global was commissioned to undertake a hydrogeological assessment. Preliminary surface and ground water estimates indicate that there will be significant water inflows into the underground mine workings, peaking at over 600 m^3/hr as the mine develops over the first 5-years. This water will be pumped to surface and evaporated using an evaporation pond and electro/mechanical evaporators. Alternative methods to handle the excess water, including re-injection into the aquifers will be pursued in the next phase of study.

The process plant and surface infrastructure have a requirement for approximately 80 m^3/hr of water which will be provided from a borehole field to be established. The borehole field will be positioned such that it reduces the inflow of water to the underground workings.

Figure 18-1 below presents a schematic of the proposed site infrastructure at the Dasa Project site.

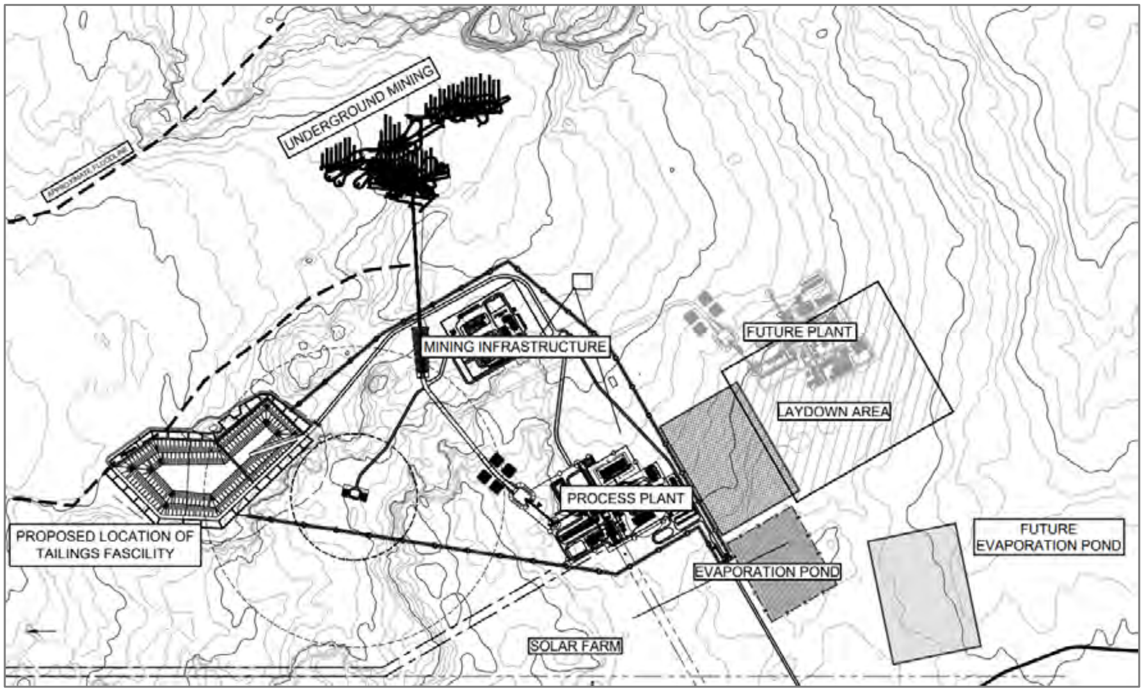


Figure 18-1: Proposed infrastructure and Spatial Placement of Key Infrastructure at Dasa Uranium Project

18.1. Process Plant and Services

Power Supply

The production and distribution of electricity are provided by Sonichar and the Nigerien Electric Company (NIGELEC), respectively. Sonichar operates the coal bed of Anu-Araren located 70 km from Agadez and about 50 km from the Dasa Project. This company produces electricity from a power plant installed on-site (2 ×18.8 MW generators) and ensures energy distribution to the town of Agadez and to the Arlit and Akokan mining sites via a 132 kV line. The Cominak mine, located in Akokan has ceased operations, so there is excess capacity available that Sonichar has indicated can be made available to power the Dasa Project

Transmission and Distribution

The existing 132 kV overhead power line runs from the Sonichar coal fired power plant, in a northerly direction, alongside the sealed N25 road to Arlit.

It is proposed that a new 132 kV T-off will be installed at the line, and a new 132 kV overhead power line will run in an easterly direction, for 5.2 km, towards the new process plant.

The process plant High Voltage (HV) switchyard will step the supply voltage down from 132 kV to 11 kV using two 15 MVA transformers, which will supply the total electrical load of 12.1 MW, including mining, process plant, infrastructure buildings and the accommodation camp. The 11 kV will be distributed using overhead lines or feeder cables and will primarily be used for distribution, main surface fans, mill motors and other large electrical loads. The 11 kV will be transformed to 400 V three-phase for use with process plant motor and equipment loads and 240 V single-phase for use with small power and lighting loads.

Security of Supply and Emergency Power

Sonichar has offered to supply the Dasa site with between 8 and 9 MW power. To supplement this power and provide backup when Sonichar is not available due to maintenance, full diesel power generation is included in the site infrastructure to enable both the mine and mill to operate in the absence of grid power.

11 kV Switchboard

The 11 kV single busbar switchboard will be supplied with vacuum circuit breakers and protection relays. This board will consist of two incomer circuits, twenty-two feeder circuits of which three are spare, one bus-coupler, two busbar mounted voltage transformers (VTs) and five cable mounted VTs. The switchboard will include an arc flash protection system and have a remote switching panel. Two battery tripping units have been allowed for. The 11 kV switchgear, battery tripping units and remote switching panel will be installed inside the e-house which are of a concrete construction.

Power Factor Correction

Power Factor Correction (PFC) has been allowed for at the 11 kV switchboard level. The PFC design has been based on a total load of 13.1 MVA at a PF of 0.84. The power factor will be corrected to 0.96 lagging. The system will consist of two outdoor enclosed 1 MVAr PFC plants with two 500 kVAr steps and a 5th harmonic filter.

Low Voltage Reticulation

Secondary distribution is at 400 V. Star points of the distribution transformers will be solidly connected to earth. Transformer secondaries shall be rated at 420 V to allow for volt drop at full load.

Substation Configuration

All process plant Motor Control Centres (MCC's) will be housed in dedicated low voltage concrete substations. The MCC's will be air conditioned for added cooling. The substations are elevated for cable access (bottom entry).

Transformers

Electrical loads are allocated to MCC's, and associated transformers. These loads are grouped by process areas as far as practical, considering transformer loading and Voltage regulation. The MCC designs have been based on 1 600 kVA and 1 000 kVA transformers. Distribution transformers are 11000/420 V, vector group Dyn11.

Infrastructure lighting and small power would be fed from 11000/420 V, 315 kVA and 500 kVA mini substations, while plant lighting and small power will be fed from dedicated 400/400 V 100 kVA transformers.

A transformer loading schedule has been completed for each transformer.

Motor Control Centres (MCC)

The MCC's will be of brick-and-mortar construction, free standing, bottom cable entry, front access, and operation and fully compartmentalised. The operating Voltage will be 400 V, 50 Hz with a control Voltage of 110 V, 50 Hz supplied from an internal control transformer. The design fault level will be 50 kA at 400 V for all transformer fed MCC's. A power meter will be provided per MCC incomer and connected to the plant supervisory network.

Motor starters will be Direct-On-Line (DOL) for motor power less than or equal to 90 kW, unless otherwise required by the process. Motors above 90 kW will be started by Soft Starter or Variable Speed Drive (VSD) depending on the application. DOL starters are typically equipped with a triple pole, Moulded Case Circuit Breakers (MCCB), contactor (Type 1 coordination) and intelligent overload relay(s). Contactors will be rated for AC-3 duty.

All starter and variable speed drive (VSD) related information will be communicated over Ethernet network to the PLC / SCADA system.

Field Equipment

All drives will be equipped with local start-stop stations with latching e-stop buttons. These will be field mounted within robust steel frames and drip covers. Plant start-up sirens will provide a warning for conveyor and large equipment drives about to start. All emergency functions such as emergency stops are to be hard wired, but will also be monitored by the PLC.

Generally, all VSD's will be mounted within the MCC. The larger VSD's will be mounted in standalone cubicles with the MCC.

Motors

Low voltage (400 V) motors will be designed to IEC 60034, for continuous duty class S1. Insulation will be Class H. Temperature rise will be limited to 80 °C (Class B). Enclosures will be IP55 to IEC 60034-5. Premium efficiency (IE3) motors will be used throughout the design and shall be of the totally enclosed fan cooling type (TEFC).

Medium Voltage Motors (11 kV) will include Surge suppressors that will be fitted separately to the motor cable box. Temperature monitoring devices will be fitted to bearings and windings on MV motors.

Cable

Medium Voltage (11 kV) cables will be individually screened copper conductor three core XLPE/PVC/SWA/PVC 6.35/11 kV cable to IEC specifications. Single core cables will be of XLPE/PVC/SWA/PVC construction and be arranged in prescribed trefoil formation (gland plates will be of non-ferrous construction).

Low voltage cables will be copper conductor PVC/PVC/SWA/PVC 600/1,000 V cable to IEC specifications. Flame-retardant cable is to be utilised for surface installations. Power cables shall have four cores, the fourth core being utilised as an effective earth between the equipment (e.g., motor) and the substation earth bar.

Conductor sizes for 400 V motor feeders shall be sized to ensure reliable motor starting. Cables are sized for a maximum 5% voltage drop during full load condition. Start-up voltage drops are determined on a case-by-case basis based on the starting torque requirements.

Cable Racks

Cables will be mounted above ground on cable racks. Cable racking will generally be run in a horizontal orientation and be made of galvanized mild steel. In corrosive areas, stainless steel racking and fixings will be used. Buried cables will be in trenches and will be provided with cable markers on surface at 10 m intervals and changes in direction. Cable trenches will be backfilled with material to ensure effective heat transfer from the cable to the surrounding earth.

Earthing

Ring earth type electrode shall be installed around the perimeter of each building or plant structure, brought to surface for connection to the structures or down conductors at intervals defined by the class of lightning protection system (LPS) and at each earth spike termination point.

Earth spikes shall be installed along the ring earth conductor. The earth spikes shall be bolted to the structures or down conductors by dedicated insulated earthing conductors for ease of identification and testing. All plant ring earths shall be interconnected.

All electrical loads will be earthed via a dedicated earth conductor (either the fourth core of a power cable or separate earth conductor) to the relevant MCC earth bar. Plant equipment, vessels and structures will be connected (bonded) to the nearest ring earth.

Lighting and Small Power

Lighting will be achieved by using energy efficient LED luminaires using a combination of strip lights, bulkheads, and floodlights to achieve illumination levels required.

Emergency lighting has been allowed for in key locations. Emergency lights will be fed from dedicated UPS circuits. Photoelectric switches will control the exterior lights. Provision has been made for weatherproof 230 V 16 A switched socket outlets and 400 V 63 A welding socket outlets.

Water

The Dasa area is in the desert climate region of the Sahelian desert type, characterized by an arid climate with a wet season from June to September, and a dry season from October to May. Average annual rainfall over the last 20-years from the nearest weather station in Tchirozerine, shows an annual minimum of 10 mm, a maximum of 175 mm and an average of 109.8 mm. Consequently, natural precipitation and the associated runoff is unlikely to be sufficient to supply the mine year-round. The area is noted for excessive surface run-off and the mine portal, mining surface infrastructure, process plant and tailing storage facility have been positioned outside of the natural water courses and flood plains.

The area is characterised by 3 known water bearing aquifers and drilling and pump tests conducted in 2021 indicate that adequate and suitable water can be obtained from depths of 27 m to 47 m below ground level. The project capital cost estimate includes for 5 equipped boreholes within a 2 km radius of the plant, complete with overhead electrical lines and pipelines to bring the water to the raw water pond, which has a capacity of 3 000 m^3. The water demand for the process plant, surface infrastructure, and mining operation is approximately 80 m^3/hr.

Pump flowrate tests conducted during the hydrogeological investigation indicates that the planned 5 boreholes will be adequate to meet demands. The investigation also determined that during the first 5-years of development and mining, the underground workings will see inflows peaking in excess of 700 m^3/hr. This excess water will be pumped to the surface where it will be contained in a series of evaporation ponds. Alternative methods to handle the excess water, including re-injection into the aquifers will be pursued in the next phase of study.

In addition to the mine water evaporation pond a similar sized evaporation pond associated with the removal of heavy metal solids in the precipitation process is provided. This bleed stream is rated at 28 m^3/hr.

Tailings Disposal

Epoch Resources (Pty) Ltd (Epoch) was appointed by METC (Pty) Ltd (METC) to undertake the design of the Dry Stack Tailings Storage Facility (DSTSF) associated with Dasa. The design process is based on the design criteria summarised in Table 18-1 and the following guidelines:

- South African National Standards (SANS).
- South African National Environmental Management: Waste Act (Act 59 of 2008).
- International Atomic Energy Agency (IAEA) technical reports on the disposal and management of radioactive waste; and International Commission on Radiological Protection (ICRP) guidelines.

The total life of mine production of tailings will amount to 8.8 million tonnes over 24-years. The average rate of filter belt tailings reporting to the DSTSF is 1000 dry tonnes per day.

Table 18-1: Design Criteria for the Dasa Dry Stack TSF.

Item	Criteria	Value	Source
1	Ore type	Uranium	GAC/METC
2	Design Life of Facility	6-years	METC
3	Average Filter Belt Tailings Production Rate	1,000 tpd	METC
4	Average Filter Belt Tailings Deposition Rate to DSTSF	1000 tpd	METC
5	Total Filter Belt Tailings	8.8 Mt	METC
6	Total Filter Belt Tailings reporting to DSTSF	8.8 Mt	METC
7	Particle Specific Gravity	2.59	SGS
8	Average Particle Size Distribution	32% passing 75 µm	SGS
9	% solids to water ratio (by mass)	80	METC
10	Average Dry Density	1.66 t/m^3	SGS/Epoch
11	Delivery Method	Trucked	METC
12	Geochemistry	Potentially Acid Forming	The Moss Group

The geochemistry analysis of the Dasa tailings was conducted on representative non-neutralised filter belt tailings samples. The geochemical testing was undertaken by SGS Minerals Services (SGS) in Canada and classified by Moss Group according to the South African Norms and Standards for the Assessment of Waste for Landfill Disposal in order to determine the liner/containment barrier requirements for the DSTSF. The tailings classified as a Type 3 waste and according to South African legislation requires a Class C Landfill Liner. A Class C liner consists of multiple layers and systems as shown in Figure 18-2. Due to the 'dry' nature of the tailing product received at the facility, as well as the climatic conditions and geology of the area, it has been assumed that several the Class C liner layers can be removed or adjusted as follows (from top to bottom):

- Finger drains of geotextile covered aggregate – toe drains have been placed at the toe of the lowest wall in each compartment, therefore this layer will not be required.
- Protection layer of silty sand or geotextile of equivalent performance –this layer will be required to protect the HDPE geomembrane from the mobile equipment operating the facility (hauling, placing, compacting etc.) as well as exposure to UV which may cause decay; and
- Under drainage and monitoring system in base preparation layer – Since the tailings product will be 'dry' and be allowed to dry out before compaction, the underdrainage system will not be necessary. Any seepage within the tailings will flow to the toe drains and therefore no hydrostatic pressure head is expected to develop above

the liner (which would cause seepage should there be a hole or defect in the liner). This layer has thus been excluded from the design.

Based on the adjustments mentioned above, the basin and toe paddock liner system for the DSTSF will consist of the following:

- 300 mm think layer of liner cushion/impermeable layer (SC material found in borrow areas around the site) compacted in layers of 150 mm thick to 98% Standard Proctor Density.
- 1.5 mm (1500 micron) thick single textured HDPE Geomembrane; and
- 300 mm thick liner protective layer of sand (On the starter wall side slopes, the sand protective layer has been replaced by an A6 Bidim geotextile, or similar specified).

The above liner design was based on South African requirements based on the South African Waste Classification of the material.

It is the environmental team's responsibility to approach local authorities in the next study phase in order to confirm the above requirements and adjustments.

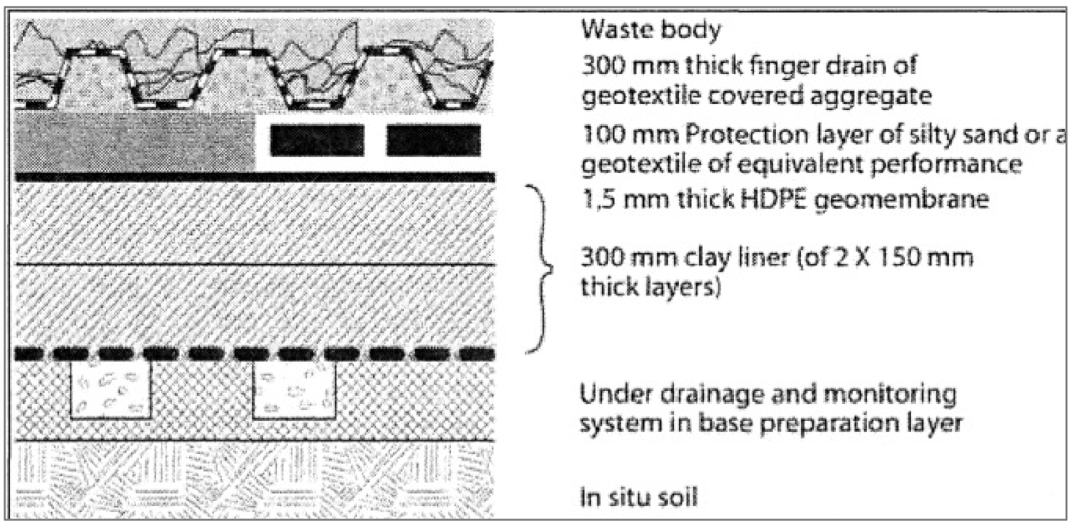


Figure 18-2: Engineering Design Requirements for a Class C Landfill (SA National Norms and Standards for the Disposal of Waste to Landfill

The preferred site for the initial DSTSF development was revised following the Site Selection study completed by Epoch in 2020. The final site selected for the first DSTSF is within the mine lease area and west of the process plant. The new site was closer to the plant which would result in reduced hauling distances and was positioned outside of the predominant wind direction in relation to the mine plant area. The suitability of the site for the development of the DSTSF was based on consideration of:

- Tailing deposition rates and required storage capacity.
- Topography.
- Proximity to the plant (i.e., haul distances).
- Exclusion zones (blast radius).

- Environmental impacts (flood lines, wadi area and predominant wind directions).
- Site properties (wall volumes, total height, footprint area and stormwater diversion).
- Potential for increase in capacity of extended life of mine.

A geotechnical investigation of the original DSTSF footprint (as identified in the Site Selection) was undertaken, comprising the excavation and profiling of 26 test pits, from which representative samples of the soil horizons were retrieved for laboratory testing. Following the repositioning of the DSTSF footprint, three core boreholes were drilled to supplement the investigation on the revised footprint where test pits were not located.

The site was found to have shallow rock at an average depth of 2 m, ranging from 0 m to 4.2 m deep. The rock was overlain by clayey sand (SC material according to USCS) in virtually all pits. The borehole drilling found deep clay layers at depths of 4 m to 10 m in contrast to the test pit investigation which indicated shallow soil depths and subsequent refusal during excavation. The borehole logs were obtained after the initial geotechnical investigation and no laboratory test work was conducted on the deep clays which may have different strength parameters to the clays already tested.

For the purpose of the FS study, in line with the low parameters obtained from the geotechnical investigation conservative assumptions were made regarding the site geotechnical conditions for the design and slope stability analysis of the facility.

The Dasa DSTSF comprises of the following:

A Tailings Deposit (TD) as follow:

- Three adjacent footprint compartments constructed in 2 tiers with side slopes of 1V:4H.
- Engineered, earth-fill starter walls of nominal height.
- A Class C liner consisting of an impermeable layer, HDPE geomembrane and liner protective layer.
- Starter wall toe drains (to reduce the potential build-up of the phreatic surface at the base of the starter walls.
- Toe drain collection manholes; and
- Toe paddocks.

Storm Water Diversion Trenches and Berms

The key parameters associated with the Dasa Dry Stack TSF are summarised in Table 18-2. The Dry Stack TSF 1 will be constructed as three phased compartments constructed with 2 Tiers. The three compartments are individually constructed to the first tier over the entire footprint area of the facility. The infrastructure layout for the three compartments is shown in Figure 18-3. The second tier will involve the simultaneous lifting of all three compartments to the DSTSF's final elevation. The phasing of the DSTSF will require that each successive compartment be constructed and equipped to receive tailings upon termination of the storage capacity of the former compartment to Tier 1. The phased development and operation of the DSTSF is illustrated in Figure 18-4.

Table 18-2: Key Parameters Associated with the Dasa DSTSF.

DSTSF 1 Parameter	Compartment 1 – Tier 1	Compartment 2 – Tier 1	Compartment 3 – Tier 1	Tier 2
Footprint area (ha)	4.7	4.1	4.2	-
Maximum tailings elevation (mamsl)	477.5	477.5	477.5	483.0
Maximum tailings height (m)	10.5	12	12.5	18.5
Tailings side slope	1V:4H			
Outer slope of starter wall	1V:2H			
Inner slope of starter wall	1V:2H			
Starter wall crest width (m)	3			
Access starter wall crest width (m)	12			
Years of tailings deposition	1.15	1.55	1.83	1.49
Cumulative years of tailings deposition	1.15	2.70	4.53	6.02
Tonnes of dry tailings stored in compartment (t)	418 520	564 865	668 280	543 386
Cumulative tonnes of dry tailings stored in the DSTSF (t)	418 520	983 385	1 651 665	2 195 051

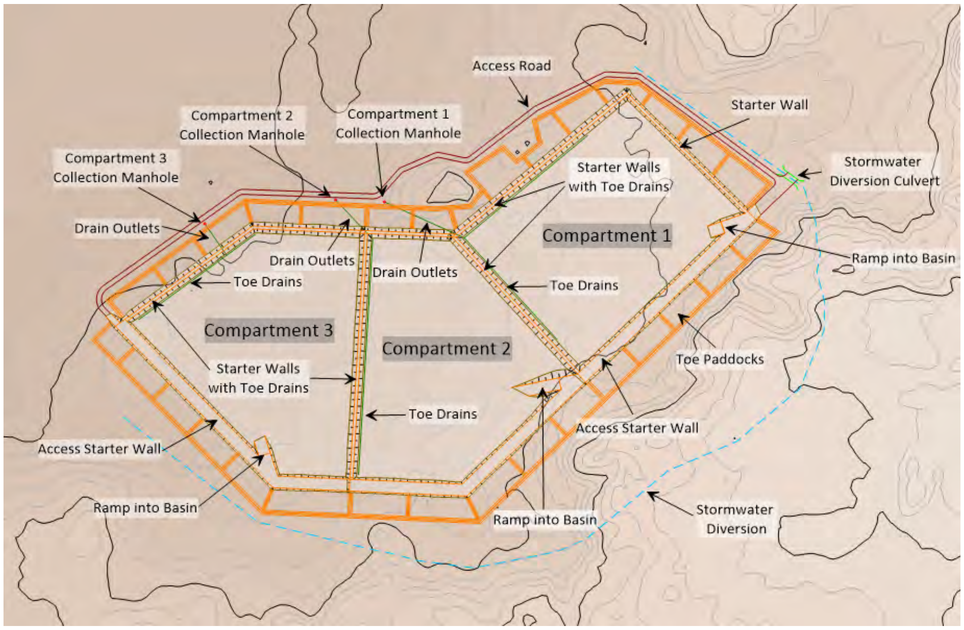
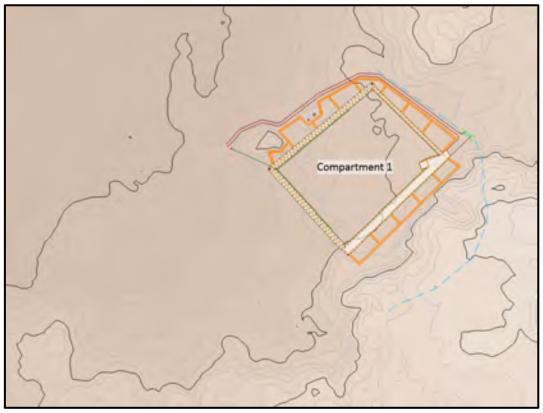
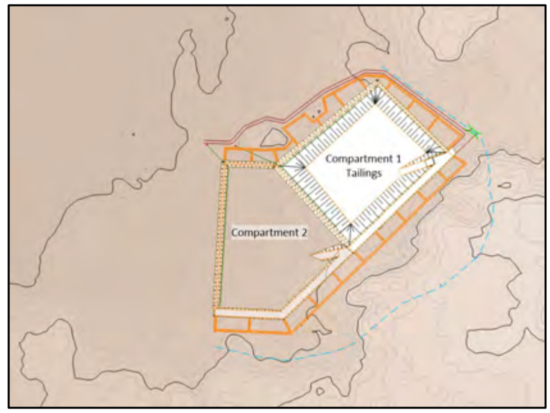


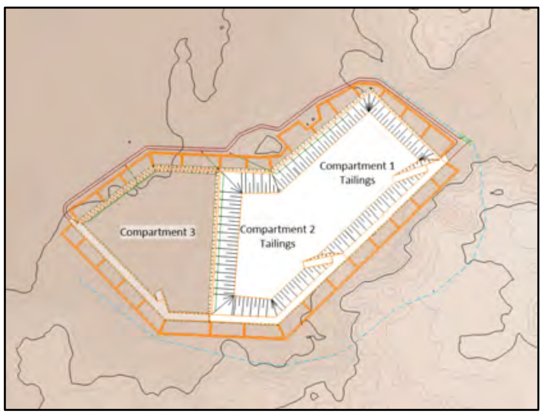
Figure 18-3: Infrastructure Layout of the Dasa DSTSF.



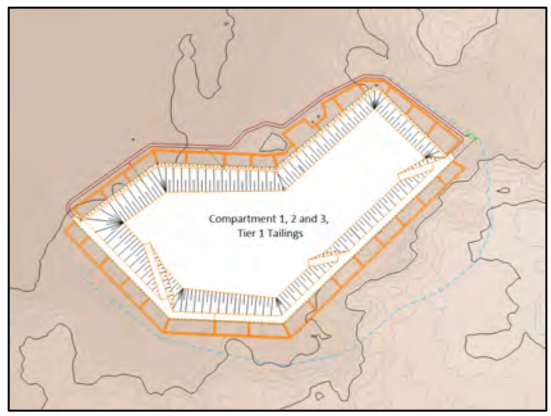
Compartment 1



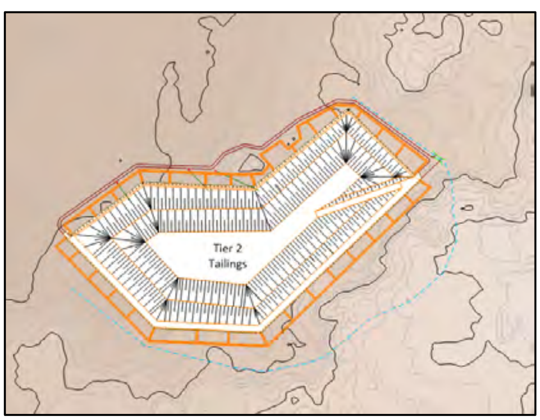
Compartment 2



Compartment 3



Compartment 1, 2 and 3 – Tier 1



Compartment 4 – Tier 2

Figure 18-4: Phasing of the DSTSF.

The delineation of the zone of influence and the safety classification of the DSTSF was carried out in accordance with the method specified by SANS 0286:1998. Based on the safety classification criteria detailed in the code of practice, the Dry Stack TSF has been classified as a Low Hazard facility.

Seepage and slope stability analyses were conducted on the DSTSF using the in-situ soil parameters received by the geotechnical site investigation and tailings values from test work conducted on a representative sample. Stability analyses were considered for the facility under operating conditions at the final height as well as post closure conditions. The results show that the TD is stable, with a factor of safety well above 1.5 all both static and pseudo-static conditions.

For the purpose of the FS study and to approximate the Life of Mine costs associated with the tailing's storage solution for the Dasa Mine, it was assumed that an additional 3 DSTSF would be required to provide adequate capacity for the 24-years of tailings production. The appropriate locations for the siting of these facilities should be identified in the subsequent phase of the project, and site-specific requirements for each facility be considered.

Communications

In Niger, the telephone system is a limited system of copper, radio telephone communications, and microwave radio relay links concentrated in the southwestern area of the country. There is a domestic satellite system consisting of three earth stations, with a fourth planned for future commissioning. Fixed-line and mobile-cellular penetration tele-density remains low at approximately 30% of the population.

The mine site will be initially linked to data and voice telecommunications network via a low earth orbiting satellite system, and once fibre is available and installed on site, the satellite solution will constitute communication redundancy.

Communications on site will link the public network to the various voice, data, and telemetry infrastructure systems within the Dasa Mine network will use fibre optic cable and Wi-Fi which will support both data and voice communications. A repeater system will be provided to enable handheld and mobile radio sets to communicate around the site.

Effluent

Sewage from the various plant, mine site and off-site buildings will be treated by means of a central sewage treatment plant located between the mining surface infrastructure and the process plant. An additional sewage treatment facility is included at the accommodation camp. Waste such as hydrocarbons from equipment maintenance and chemical waste from the laboratory will be collected and removed to the mines waste disposal facility, where it will be stored for collection and regularly removed by an appointed contractor and disposed of responsibly at a facility in the town of Agadez. Office waste, general waste and waste generated from the transportation and packaging of equipment and reagents etc. will be collected at the various sites and transported to the refuse disposal area where it will be sorted and prepared either for collection by an appointed contractor or incinerated. Waste materials that cannot be incinerated will be transported to and disposed of at the facility in Agadez.

The sulfuric acid plant follows World Bank and Niger standards, and is of double contact type, thus ensuring the highest sulphur to sulphuric acid conversions and low emissions within environmental regulations. On start-up, the off gases from the sulphuric acid plant will be treated by means of a caustic scrubber which will ensure that during the startup period, the emissions also remain below the applicable environmental emission limits.

The dust extraction and baghouse facility at the milling area will be designed such that the dust caused by the processing equipment in the milling area is kept to a minimum and any emissions from the baghouse plant conforms to the applicable environmental standards.

Similarly, the off-gas module which forms part of the product drying and packaging system will ensure that the effluent gases produced by the drying module conform to the applicable environmental standards.

Transportation

There are no railways in Niger or usable waterways. All regional transportation is provided by trucks that use a relatively dense road network in the south, leading to the north and east of the country. Most of the capital equipment required for the Dasa Project and consumables used once the mine is operational, will be imported from outside of the West Africa region. Historically, all goods arrived via the port of Cotonou in Benin and were transported by road from Cotonou through the border town of Gaya. There is a 1,200 km paved road between Gaya and Arlit. Other mining operations in the Arlit region use this route and despite the poor road conditions in some places, the local transporters and logistic providers get goods to site. Sanctions were placed on Niger in August 2023 and an alternative routing from the port of Lome through Togo and Burkina Faso has since been used. Recently, the sanctions were dropped so it is expected the Benin route will open again. Abnormal and fragile cargo likely to be required for the construction phase of the project will need to be carefully planned. The airports of Niamey and Agadez can receive large freight aircraft.

The proposed access road to the plant site is an existing sand piste (track) approximately 5.2 km in length which runs from the N25 main sealed road that connects the towns of Arlit and Agadez. The existing sand piste will be upgraded to an unsealed road suitable for frequent heavy load traffic and will be maintained by the mine site.

On site, a network of approximately 12.1 km of internal roads will connect the various infrastructure, buildings, accommodation camp, and mine entrances to the main access road. The construction of these roads will be of an unsealed, graded, and compact type with demarcation and drainage ditches.

Fuel Storage and Distribution

Diesel fuel storage, dispensing and distribution facilities will be provided to supply fuel to light vehicles, surface mobile plant and equipment, processing plant equipment and the diesel backup generators. The diesel storage tanks selected, are shipping container sized that are double skinned or “self-bunded” and are commonly referred to as “tanktainers”. They are fully equipped with transfer pumps as well as dispensing units for the different types of vehicles and equipment. All fuel required at the plant site will be delivered in tanker trucks by commercial suppliers. The fuel dispensing and vehicle park bay will be bunded to prevent spillage of fuel contaminating the site area.

Wash-Bay Facilities

Vehicle wash-bay facilities will be provided nearby the plant and mining workshops. They will comprise of bunded concrete slabs sloping to settling sumps. The sumps will be equipped with oil/water separators which will treat any dirty water collected in the sumps.

Each wash-bay will be equipped with washing equipment specifically designed for the cleaning of the designated vehicles which may include light vehicles, heavy mobile and earthmoving vehicles.

Workshops

A vehicle workshop will be constructed on surface near the mining surface infrastructure area and will service, maintain, and overhaul light vehicles, surface mobile plant and the mechanised underground mining machinery. The workshop will be sized to handle the range of equipment envisaged on the project and shall be equipped with an inspection pit, lifting equipment, tools, spares stores, offices, tire storage and oils, paints, and lubricant stores. A sump pump will transfer dirty water to an oil/water separator.

Maintenance workshops will be constructed adjacent to the processing plant and in the mining surface infrastructure area and will be divided into separate sections to accommodate each engineering discipline (fitting, boiler making, rigging, electrical and control and instrumentation). Each section of the workshops will be equipped with a spares store, office and the machinery, tools, and equipment, including cranes and hoists (where required), necessary for the different engineering disciplines to perform maintenance in and around the mine and plant site.

Buildings

Infrastructure buildings are classified as either architectural, control rooms or industrial. Architectural buildings include administration offices, ablution facilities, change houses, tea rooms and accommodation camps. Control rooms include the underground mining and the process plant control room and will be of the same type of construction as the architectural buildings. Industrial buildings include workshops, stores, and buildings that house processing equipment.

As far as reasonably practicable, the architectural buildings will be fabricated of concrete. All architectural buildings will be equipped with air lock doors to prevent dust ingress and will be equipped with air conditioning units.

Industrial buildings will be constructed predominantly from structural steel and sheeting as these building require overhead cranes. Pre-engineered buildings will be used where practical.

The process plant buildings that contain processing equipment will be constructed of painted mild steel. The paint coating will be applicable to the corrosion protection required, taking into account both the macro and microenvironments where they are situated. These buildings will be designed to be suitable to accommodate any static and dynamic loads generated by the processing equipment. Similarly, acid proofing will be applied to concrete structures, such as sumps, where required. All buildings will include suitable stairways, walkways, and platforms to enable all operational and maintenance functions to be performed.

Accommodation Camp

The existing accommodation camp is approximately 7 km from the mining operation. The available accommodations are being increased through a combination of prefabricated and locally built camp facilities to increase the total available housing to 700 persons. Facilities presently include a canteen with the requisite food storage, preparation, cooking, and housekeeping facilities servicing all staff. The canteen facilities are being expanded for the increased numbers. Recreation, security, access control, laundry and administration units are being provided at the camp. The camp is operated by an independent service provider.

The intention is to accommodate all staff in the accommodation camp on a rotation system, whereby they will be collected from their hometowns and then be accommodated on site for their work cycle, thereafter, being returned home for their rest period. This will minimise the need for a town to develop at the mine site. The accommodation camp will provide the majority of the requirements of the construction teams and as the construction teams decrease in numbers, they will be replaced by operational readiness teams and ultimately the operational crews to run the mine.

Security

All persons entering the process plant and mine facilities areas will be required to pass through the continuously manned boom gate adjacent to the administration building on the access road. For security reasons and due to the handling of a potentially hazardous product, security guards located adjacent to the administration building will control all entry and exit of vehicles and personnel. Search and inspection of personnel, bags and items leaving the plant will be carried out at this facility.

A standard medium security fence will be constructed around all Project facilities including the process plant, mine waste disposal facility, mine, and raw and process water reservoirs. Security fencing with lockable access gates will be installed locally around the remote pumping facilities and the explosives magazine.

Access to hazardous and flammable process will be restricted to authorised persons and equipment, and these areas too will be separately fenced and controlled by the security team.

Additional security fencing will be provided around the warehouse yard.

Fire Protection

a) General

Fire protection will consist of the provision of fire hydrants, fire hose reel cabinets and fire extinguishers placed strategically around the facilities in accordance with the requirements of the relevant regulations. Firefighting water will be supplied from a dedicated volume in the fire water tank from where the water will be pumped to the required area. Jockey, duty, and diesel-powered standby pumps will be provided for this purpose. Various types of fire extinguishers will be provided in areas where water as a means of fire control is undesirable.

b) Solvent Extraction (SX) Plant

General

All suppression systems will be equipped with dedicated cylinders, containing suppression agent in the form of a water and foam mixture. An Aqueous Film-Forming Foam (AFFF) will be used to further enhance the fire suppression capability. The AFFF provides a rapidly spreading foam and total insulation from air, ensuring quick extinguishing of a liquid fuel fire.

The suppression systems will utilise linear heat detection tubing as primary detection. Once the linear heat detection tubing is exposed to a rated elevated temperature, it will burst near the fire or heat source, causing the differential valve to open and discharge the water/foam mist through the discharge nozzles.

Settlers

Twelve nozzles will be installed on the inside of each settler tank to suppress any fire that could occur in the tank. These twelve overhead nozzles will be mounted internally in two rows of six nozzles to create a foam blanket in the event of a fire.

Mixer Tanks

The fire protection system for the mixer tanks will consist of twenty water deluge nozzles directing towards the tanks in the following manner: six nozzles evenly distributed around the top rings, six nozzles evenly distributed around the middle rings, six nozzles evenly distributed around the bottom rings of the tanks and two nozzles directed towards the motors on top of the tank. The bund wall will be required to contain any product spillage from the tanks and will be sloped towards the spillage pump.

Loaded Organic Tank – Surge Tank

Bund Suppression: Twenty-six nozzles will be installed around the bund wall to create a foam blanket covering any product spillage in the event of a fire. A fire at the spillage pump will also be suppressed by these nozzles. The bund wall will be required to contain any product spillage from the tank and will be sloped towards the spillage pump.

Tank Cooling: Three water deluge nozzles will be positioned on top of the tank to ensure proper cooling of the tank in the event of a fire.

Loaded Organic Tank - After Settler

Bund Suppression: Six nozzles will be installed around the bund wall to create a foam blanket covering any product spillage in the event of a fire. A fire at the spillage pump will also be suppressed by these nozzles. The bund wall will be required to contain any product spillage from the tank and will be sloped towards the spillage pump.

Tank Cooling: One water deluge nozzle will be positioned on top of the tank to ensure proper cooling of the tank in the event of a fire.

A galvanised support structure is installed across the tank to support the nozzle and piping.

Stripped Organic Tank After Settler

Bund Suppression: Six nozzles will be installed around the bund wall to create a foam blanket covering any product spillage in the event of a fire. A fire at the spillage pump will also be suppressed by these nozzles. The bund wall will be required to contain any product spillage from the tank and will be sloped towards the spillage pump.

Tank Cooling: Three water deluge nozzles will be positioned on top of the tank along the length of the tank to ensure proper cooling of the tank in the event of a fire.

A galvanised support structure is installed across the tank to support the nozzles and piping.

Stripped Organic Surge Tank

Bund Suppression: Twenty-six nozzles will be installed around the bund wall to create a foam blanket covering any product spillage in the event of a fire. A fire at the spillage pump will also be suppressed by these nozzles. The bund wall will be required to contain any product spillage from the tank and will be sloped towards the spillage pump.

Tank Cooling: Three water deluge nozzles will be positioned on top of the tank to ensure proper cooling of the tank in the event of a fire.

A galvanised support structure is installed across the tank to support the nozzles and piping.

18.2. Mining Infrastructure (Surface Complex)

The layout of the current temporary infrastructure utilised during the early works phase is indicated in Figure 18-5. All temporary infrastructure will be removed from the site and replaced by the planned permanent infrastructure.

The surface mining infrastructure and area access roadways that has been considered for the Dasa Project, is illustrated in the overall permanent site layout in Figure 18-5 and the matching legend is included in Table 18-3. All of the infrastructure and supporting facilities are located in close proximity to the decline portal. The workshops, offices, stores, diesel storage tanks, vehicle brake test ramp, vehicle parking areas and other facilities have been laid out to support the planned mining operations for the Dasa activities and are logically positioned to separate the heavy underground vehicle routes from other vehicle routes, as far as is practicable.

The mined ore will be hauled and dumped on an ore pad arrangement, situated at the at the process plant.

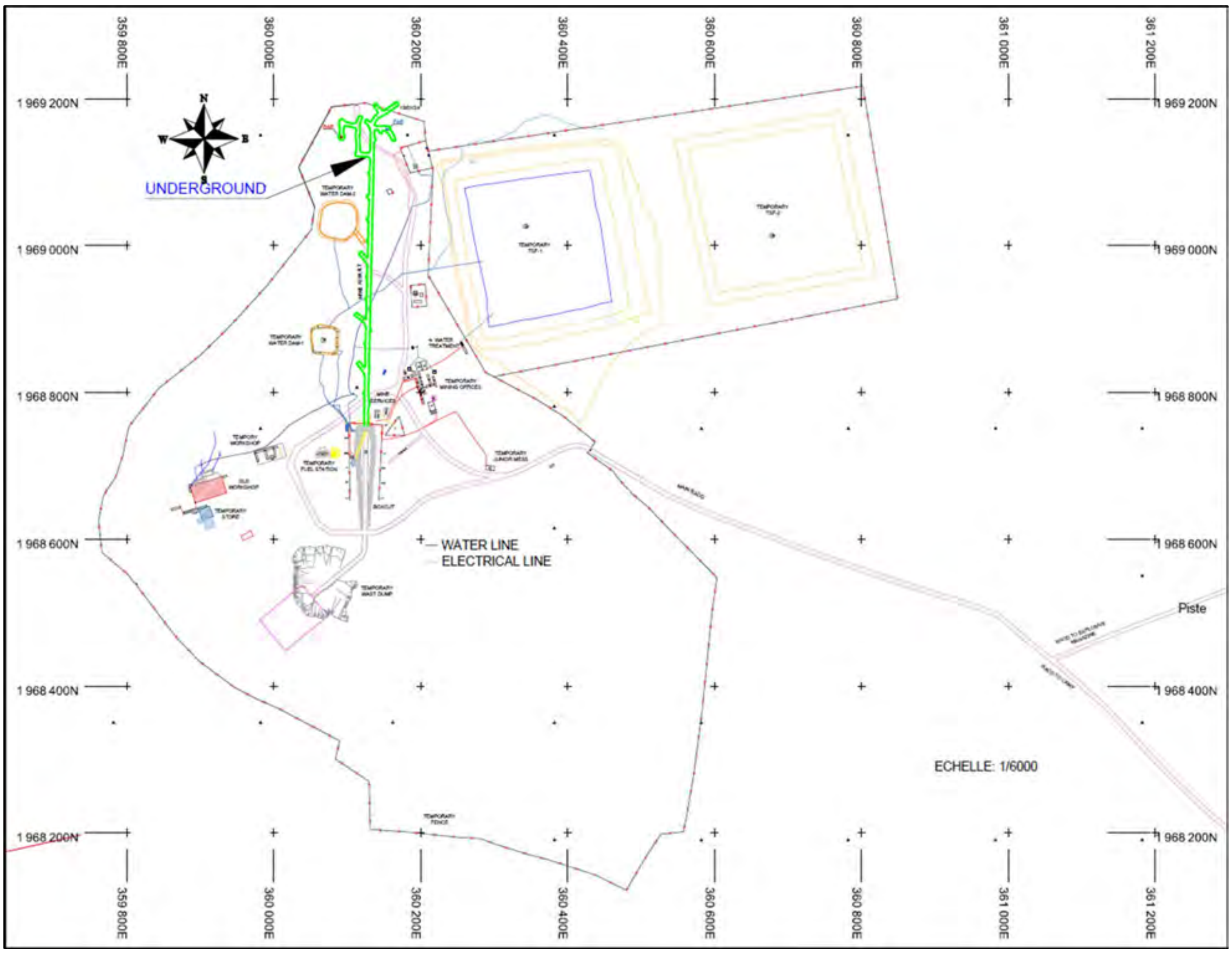


Figure 18-5: Layout of Temporary Mining Infrastructure.

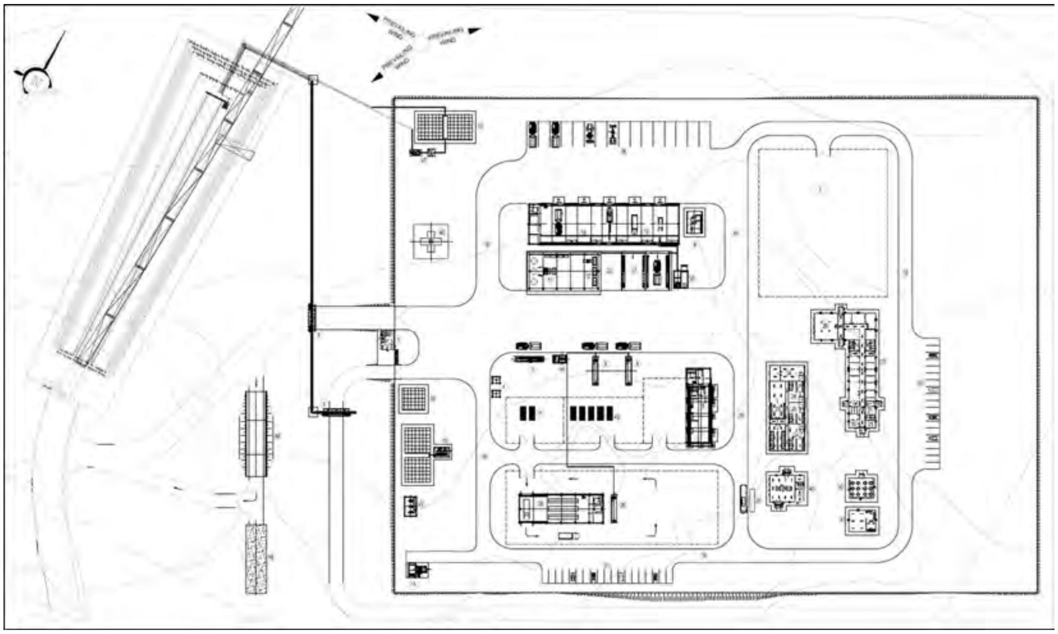


Figure 18-6: Proposed Layout of Surface Mining Infrastructure.

Table 18-3: Surface Mining Infrastructure Provisions for Dasa.

Mark No.	Description	Mark No.	Description
1	Main Gate Security Office	24	Change House – Workforce Ladies
2	Control Room	25	Change House – Management Men
3	General Laydown Area	26	Change House – Management Ladies
4	Emulsion Isotainers	27	Main Office Block
5	General Workshop	28	Open Plan Office
6	Pipe Gantries	29	Perimeter Fence
7	Lubrication Station	30	Canteen / Restroom
8	Diesel Refuelling Stations	31	Parking - Light Vehicles
9	Compressor House - Surface	32	Potable Water Storage Tank
10	Fire Water Storage Tanks	33	Oil/Water Separation Sump
11	Tyre Storage and Handling Bay	34	Laundry
12	Tyre Inflation Safety Cages	35	Mine Rescue Services Room

Mark No.	Description	Mark No.	Description
13	Service Water Storage Tanks	36	Parking - Mining Vehicles
14	Sub-Station	37	Personnel pick-up / drop off
15	Sewage Collection Sump	38	Mine Store
16	Internal Roads	39	Lamp Room
17	Mining Vehicle Wash Bay	40	Generators
18	Mining Vehicle Workshops	41	Diesel Offloading Transfer Pump
19	Support Vehicle Workshops	42	Compressor House - Underground
20	Support Vehicle Refuelling	43	Decline Portal
21	First Aid Station	44	Settling Facility
22	Support Vehicle Wash Bay	45	Brake Test Ramp
23	Change House – Workforce Men	46	Sand Pit

Access and Internal Roads

Provision has been made for unsealed surface roads with a 150 mm thick wearing coarse of G5 layered material for all internal roads within the mining cluster as well as within the explosive magazine area. The width of the access roads inside the mining cluster will vary depending on the type of vehicles that will be utilising that specific section of road.

Security and Fencing

Access to and from the mining cluster will be controlled by security personnel via two entrance gates. The one gate will be exclusively used for the trackless mining vehicles travelling to and from the decline portal, and the second gate will control access for all normal road vehicles used by staff, visitors, contractors and for deliveries etc. The mining cluster will be enclosed with a 2.4 m high diamond mesh fence. Provision has been made for concrete type security office with toilets and four turnstiles, which straddles the area between the two access gates.

General Laydown Area

The general laydown area is situated within the mining cluster, the area will be enclosed with 2.4 m high diamond mesh perimeter fence, having a secure single access point through a set of gates. The surface area will be levelled, excavated, backfilled, and compacted with G5 layered material to a thickness of 150 mm. Provision has been made for pole mounted, energy efficient LED type floodlights to illuminate the area.

Emulsion Storage

Bulk emulsion for underground blasting will be stored on-site is isotainers provided by the explosive's supplier. Provision has been made for a bunded concrete pad for placement of four isotainers which will provide storage capacity of 120 tonnes of bulk emulsion. This equates to approximately two months' worth of supply.

General Workshop

The general workshop will cater for the mine boiler making workshop, mobile equipment workshop which includes the fitting, electrical and instrumentation maintenance, and repair requirements for both surface and underground equipment. The general workshop floor area is divided into the three working sections by means of dividing walls to create a dedicated working space for each discipline.

The general workshop will be a steel structure with IBR cladding and a reinforced concrete floor. Provision has been made for overhead travelling gantry cranes, jib cranes, toilet facilities, a kitchen, storeroom, and offices.

Lubrication and Diesel Refuelling Station

The diesel refuelling and lubrication station is positioned to provide a logical route for the mining vehicles leaving the mining cluster to proceed underground. It is also positioned across from the tyre inflation station and wash bay. The lubrication and diesel refuelling station comprises of the following main components:

- 2 off 67,100 L Containerised and self-bunded tanks.
- 1 off Diesel offloading / Decanting facility.
- 2 off Diesel dispensing and metering pump units.
- 4 off 2,000 L Containerised and self-bunded tanks (Hydraulic oil, engine oil, transmission oil and engine coolant).
- 4 off Receiving pump units.
- 4 off Dispensing pump units.

Surface Compressor House

The surface compressor house will be a steel structure, with IBR cladding and a reinforced concrete floor. Two oil-injected rotary screw compressors, one duty and one stand-by, with a 12-bar air receiver (pressure vessel) will be installed in the surface compressor house. These compressors are designed to supply compressed air to inflate the trackless vehicle tyres and to provide air for the pneumatically driven lubrication facilities. It also supplies compressed air to all surface areas within the mining cluster, such as the trackless workshop and general surface workshop. The surface compressed air reticulation system will consist of 50 NB carbon steel piping, supplying air to all the required areas.

Fire Water Storage Tanks

A fire suppression system was provided for the mining cluster comprising of storage tanks, pump units and a buried pressurised water ring main system with fire hydrant connections. To ensure that an adequate supply of water may be dedicated to fire suppression, two separate pressed steel storage tanks have been provided for the storage of fire-fighting water. The fire water reticulation system will consist of a main ring with appropriately positioned take-off points. There are a total of 18 fire hydrants within the mining cluster area.

Tyre Store and Tyre Inflation Bay

The tyre store and tyre inflation bay will be a steel structure, with IBR roof cladding and a reinforced concrete floor. Provision has been made for tyre inflation cages to allow for the safe inflation of mining vehicle tyres, together with a 5 tonne overhead travelling gantry crane for lifting and handling the heavy, mining wheels and tyres.

Service Water Storage Tanks

A dirty water relay pumping system is planned for the handling and pumping of dirty water from the underground operations to surface. A portion of this water will be fed through a settling facility to produce clarified service water for re-use underground. The settled and clarified water will be piped to the service water storage tanks. This service water will then be gravity fed back underground for use as required. Two separate pressed steel type storage tanks have been provided for the storage of this recovered service water.

Mine Bulk Power Supply and Main Sub-Station

The mine main substation will be a containerised E-house and will be supplied with power from the processing plant which includes back up diesel generators providing the mining emergency power requirements. Power will be reticulated to the surface infrastructure, ventilation fans and underground areas using cables and overhead lines at 11 kV. The single line diagram for the MV reticulation is shown in Figure 18-7.

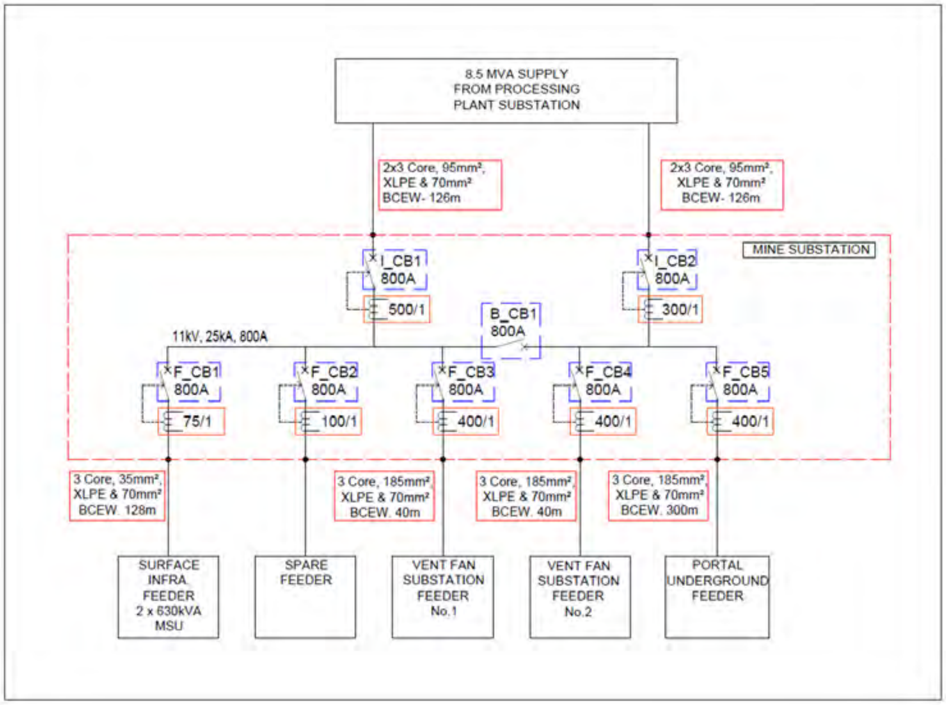


Figure 18-7: Dasa Mine MV Reticulation.

The calculated maximum demand for the Dasa mining operation is 7.3 MW, occurring during year 2030. The mining loads are provided in Table 18-4.

Table 18-4: Dasa Mining Load List.

Description	Connected Load (kW)	Operating Load (kW)
Surface Infrastructure	700	310
Surface Infrastructure - Vent Raise Shaft	120	90
Ventilation	3 800	3 220
Pumping	8 500	2 988
UG Infrastructure	500	256
Mining	1 600	440
Total	15 220	7 304
Peak Operating Load (MW)		**7.3**

Sewage Collection

Black and greywater from the offices, change house, and laundry will report to a sewage collection sump and transfer pump station, situated within the mining cluster area. The black and greywater will then be pumped from the sewage collection sump to the central sewage treatment plant. Underground employees will make use of portable chemical toilets. The drums of these toilets will be collected during routine services and transported to surface. A sewage transfer hopper for the discharging of these drums into the sewage collection sump has been provided for as well as a storage facility for clean drums to be taken underground again.

Trackless Mining Machinery Wash Bay

The TMM wash bay consists of three wash bays, separated by reinforced concrete walls. Two wash bays are dedicated for mining vehicles and the other one for light vehicles. The wash bay will be constructed with a reinforced concrete floor and a collection drain for collecting dirty oil and hydrocarbon residue. The collection drain will feed the oil/water separation sump, which is described elsewhere in the report. Provisions have been made for high pressure washing units, lighting and raised platforms on either side of the mining vehicle wash bays, to access all vehicles for washing.

Trackless Mining Machinery Workshop

The TMM workshop will be a steel structure with IBR cladding and a reinforced concrete floor. The layout of the workshop allows for the ability to meet the maintenance requirements of all the mining and other light vehicles. The building has five major repair bays and is equipped with offices and toilets for supervisory personnel, hydraulic repair facilities, electrical repair facilities, a 10 tonne overhead travelling gantry crane, tools, equipment, storage space and work benches.

The facility will be constructed with a collection drain feeding the oil/water separation sump to recover oil. The oil/water separation sump will be described in detail elsewhere in the report.

Change House and Laundry

The change house and laundry building are a prefabricated type building which will be assembled and installed on a reinforced concrete slab. Provision has been made for “airlock” facilities at all entrances to the building, to contain the cool, air-conditioned air inside the building, as well as to prevent dust and sand from blowing into the building when personnel enter and exit the building.

The change house and laundry building are located within walking distance from the main office building, lamp room and the personnel drop-off and pick-up facility. The facility is divided into four separate change houses to cater for the following split of personnel, and is based on the manpower schedule:

- Workforce male.
- Workforce female.
- Management / Visitor male.

- Management / Visitor female.

The change houses provide changing and ablution facilities for approximately 290 personnel and are equipped with showers, geysers, toilets, urinals, wash hand basins and Whirly bird type ventilation air extractors. Provision has been made for 4-tier type, lockable steel lockers, which will be allocated to individual employees, as required, for storing personal belongings.

Main Office Building

The main office building is an architectural type building which will be constructed on a reinforced concrete slab. Provision has been made for "airlock" facilities at all entrances to the building, to contain the cool air-conditioned air inside the building, as well as to prevent dust and sand from blowing into the building when personnel enter or exit the building. The following facilities are included in the main office building:

- Management single occupant offices.
- Technical Services and supervisory single occupant offices.
- Technical Services open plan office area.
- Toilets.
- Kitchen.
- Boardroom.
- Control Room.
- Server Room.
- Strongroom (Secure document storage).
- First Aid station.
- Training room.

Provision has been made to fully furnish the offices and facilities as required. Facilities will be equipped with air conditioners, double plugs, double fluorescent lights, and water heating equipment such as hydroboil units in the kitchens.

Canteen and Restroom

The canteen building is an architectural type building which will be constructed on a reinforced concrete slab. Provision has been made for "airlock" facilities at all entrances to the building, to contain the cool air-conditioned air inside the building as well as to prevent dust and sand from blowing into the building when personnel enter or exit the building. The canteen building will be equipped with double sinks, counter tops, air conditioners and water heating equipment such as hydroboil units. Provision has been made for four-seater canteen tables with chairs and electrical equipment such as a refrigerator and a microwave oven etc.

Potable Water Storage Tanks

Potable water will be required on surface at the main office, canteen, change house and laundry buildings, as well as the kitchens in the workshops and store. Potable water will also be required underground for drinking and also to provide a source of gland service water for the underground pump sets.

Provision has been made to store the peak usage per day, which is calculated to be approximately 330 m^3, in a pressed steel type tank positioned on surface at the mining cluster. The storage tank will be installed on purpose designed, 5 m high steel structure, to provide a gravity feed system with an inlet feed from the plant supply column. The potable water reticulation will consist of 50 NB carbon steel piping, feeding all the required areas.

Oil and Water Separation Sump

The oil and water separation sump will be fed from the vehicle workshop and wash bay by means of drains and a trench arrangement. The polluted oil / water and hydrocarbon residue contaminated water, separation process will be achieved by means of a rope mop skimmer unit. This machine will remove the floating oils from the polluted water collected and contained in the sump section, with the continuously running rope loop extracting oil by the skimmer. The dirty oil will then be transferred from the rope mop skimmer's separation tank into oil drums. Provision has been made for a storage area for these oil drums. The oil and water separation sump are also equipped with a grit trap, walkway grating and handrailing for safe access, as well as a vertical spindle pump to pump the oil free water to the dirty water storage tank.

Personnel Pick-up / Drop-off

Personnel will be transported between the accommodation camp and the mining cluster as well as to and from underground working areas. Provision has been made for a central personnel pick-up and drop off "bus shelter" type facility, which is located near the change house and lamp room buildings.

Mine Store

The mine store will be a steel structure with IBR cladding and a reinforced concrete floor, designed to service the needs of all the mine requirements including the underground mining activities. The building is equipped with offices and toilets for store management and clerical personnel. It is also provided with a storeroom, a 6 tonne overhead travelling gantry crane and heavy-duty storage racking. Provision has been made for a drive through facility for delivery vehicles to facilitate direct crane loading and offloading.

The location of the mine store, within the mining cluster, was based on ensuring minimum interaction between the traffic flow of delivery vehicles and mining vehicles. The facility has also been set out to accommodate the receiving and issuing of all goods, with a yard area for handling and storing larger items such as pipes.

Lamp Room

The lamp room is in built to the change-laundry building and thus forms part of an architectural concrete building. Provision has been made for "airlock" facilities at all entrances to the building, to contain the cool air-conditioned air inside the building, as well as to prevent dust and sand from blowing into the building when personnel enter or exit the building.

The lamp room will be equipped with access-controlled turnstiles at the entrance and exit of the building in order to capture and account for the movement of personnel going underground and returning to surface.

This would form part of the human resource management system and to ensure that the underground workings are clear before blasting takes place. It is equipped with charging racks for cap lamps, together with storage space, alongside, for each allocated self-contained self-rescue (SCSR) pack. A section is also provided for the storage and charging of gas monitoring and measuring instruments.

Power

Power is provided to the mine from the central power station for the site which uses grid power in the first instance, backed up by diesel generated power when grid power is not available. Further provision has been made for transformers and switchgear, both also housed in a concrete industrial MCC building. Diesel will be provided to the generators from the diesel refuelling station, situated near the generator station.

Underground Air Compressor Station

The compressed air requirement for refuge chambers underground will be provided by a separate compressor installation as opposed to the workshop compressor installation, as the duty is significantly different and dedicated. This compressor station will be a steel structure, with IBR cladding and a reinforced concrete floor. Three off, air cooled, rotary screw type compressors, two duty and one stand-by, will be installed in the underground air compressor station.

18.3. Underground Infrastructure

Underground infrastructure comprises all facilities, utilities and reticulation required to support the underground mining operation. These infrastructures include:

- Service water infrastructure and reticulation.
- Mine, dewatering infrastructure.
- Compressed air reticulation.
- Backfill reticulation.
- Electrical infrastructure and reticulation.
- Communication, control, and instrumentation

Service Water Infrastructure and Reticulation

The mine service water system will serve to provide raw water to production and development drill rigs for flushing water in addition to providing water for auxiliary tasks such as watering down and mixing cement.

The mine service water requirements have been calculated using empirical usage ratios. The system has been designed to service the peak production and development rates. Over a 30 day per month shift cycle and a usage ratio of 0.5 tonnes of water required per tonne of rock broken, a maximum of 0.94 ML/day of service water is required during peak production and development activities in 2028. With two drilling shifts at an effective 6 hours per shift, a peak system load of 21.8 l/s may be expected during this period. The mine service water usage calculation for 2028 is presented in Table 18-5 with the life of mine service water requirement presented in Figure 18-8.

Table 18-5: Mine Service Water Usage (2029).

Description	Total	Units
Physicals		
Ore Tonnes	372 221	tpa
Waste Tonnes	305 220	tpa
Shift Cycle		
Days per Month	30	day/month
Drilling Shifts per Day	2	shift/day
Drilling Hours per Shift	6	hr
Service Water		
Service Water Usage Ratio (Production)	0.5	tonne/tonne
Service Water Usage Ratio (Development)	0.5	tonne/tonne
Service Water per Month	28.23	ML/month
Service Water per Day	0.94	ML/day
Peak Service Water Drilling Shift	21.8	l/s

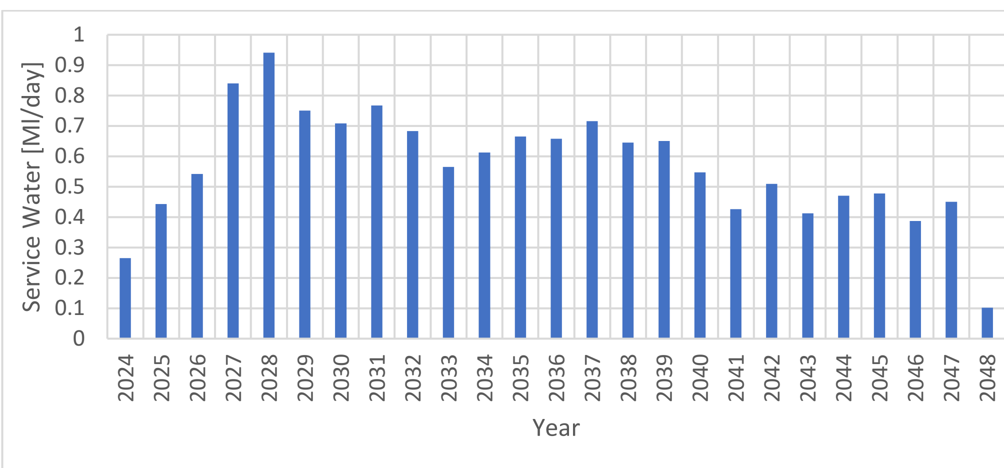


Figure 18-8: Mine Service Water Usage over LoM.

Service water will be obtained from excess water which is pumped from the underground workings and treated at the mine water treatment facilities situated at the surface complex.

Service water will be supplied to the underground mine by a service water header tank located on surface. From the service water header tank, service water will be transferred through the access portal, decline and service raises into the underground workings. The system pressure will be controlled through a series of cascade dam installed at specific location such that the overall service water pressure does not exceed 16 bar throughout the mine. At a certain location in the mine, a service water booster pump station is required to facilitate mining of Zone 5. A schematic of the service water system is presented in Figure 18-9.

The primary service water backbone is installed in the service raises and in the decline. The primary backbone will be 100 mm nominal diameter steel piping with 16 bar flanges. Water from the primary backbone will be reticulated to the development and production working areas by means of 80 mm nominal diameter steel piping with 16 bar flanges. All piping will be installed in the declines, drives and vertical intake service raises. Piping in the decline and horizontal sections will be supported by means of parrot hanger supports which are anchored into the hanging wall. Certain sections of the primary backbone will require bearer support such as the transition points from the vertical service raise to the horizontal development.

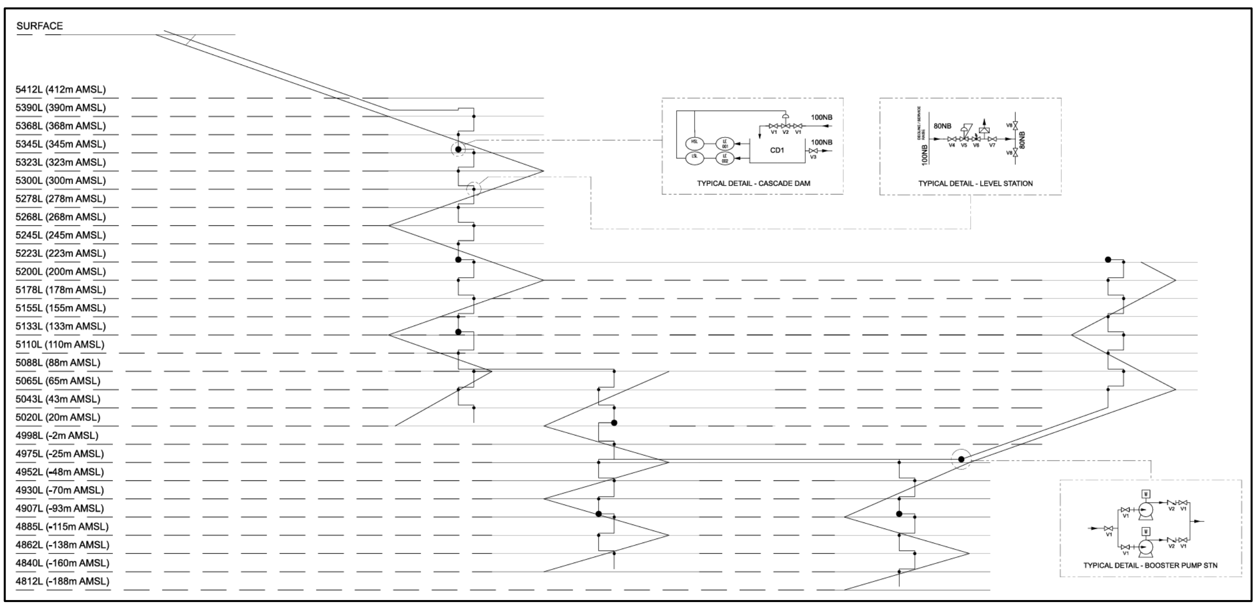


Figure 18-9: Mine Service Water Schematic.

Service water cascade dams' function to relieve the service water system pressure and are primarily used for mining Zones 1 to 4. The cascade dams will be situated in the decline or in cubbies near the service raises. The dams will be 5 m wide, 4 m long and will have a live height of 3 m. The water level in each dam will be controlled through float valves. In total, five (5) cascade dams are installed through the life of mine; their locations are presented in Table 18-6. A general arrangement of a cascade dam is presented in Figure 18-9.

Table 18-6: Cascade Dam Elevations and Servicing Levels.

Cascade Dam	Elevation	Elevation	Serving Strike Drives	
[]	[m AMSL]	[m BD]	[m BD to m BD]	
Surface	481	5345 L	5390 L	5300 L
1	345	5245 L	5278 L	5178 L
2	223	5223 L	5155 L	5067 L
3	133	5133 L	5044 L	4977 L
4	22	5022 L	4954 L	4864 L
5	-91	4909 L	4842 L	4804 L

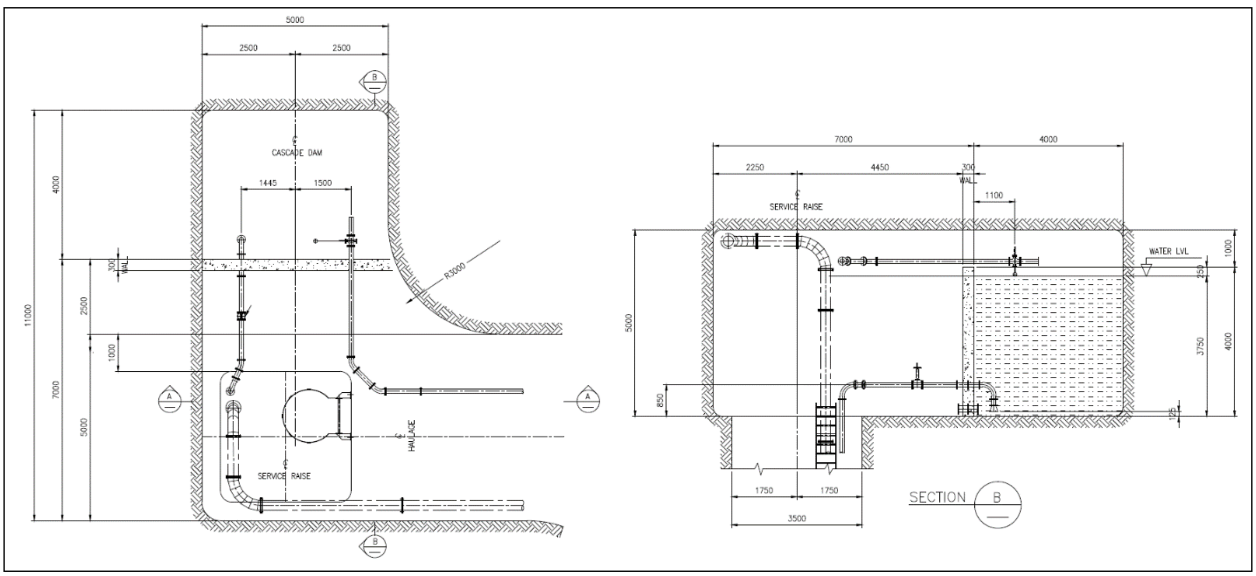


Figure 18-10: Arrangement of Service Water Cascade near Service Raise.

For mining Zone 5, a booster pump station is required to transfer water to a header dam at the top of the mining block. The booster pump station is located at top of Zone 4. Water is transferred from the fourth cascade dam system to the booster pump station, which comprises inline centrifugal pumps sets. The pump sets pump the service water to the header dam, situated at the top of Zone 5. From this dam, water is transferred to the Zone 5 workings by gravity.

The primary service water backbone is installed in the service raises and in the decline; the route is presented in Figure 18-11. The primary backbone will be 100 mm nominal diameter steel piping with 16 bar flanges. Water from the primary backbone will be reticulated to the development and production working areas by means of 80 mm nominal diameter steel piping with 16 bar flanges. All piping in the decline and horizontal sections will be supported by means of parrot hanger supports which are anchored into the hanging wall. Certain sections of the primary backbone will require bearer support such as the transition points from the vertical service raise to the horizontal development. Allowances have been made in this regard however, detailed pipe stress analysis must be undertaken at the detail design phase of work.

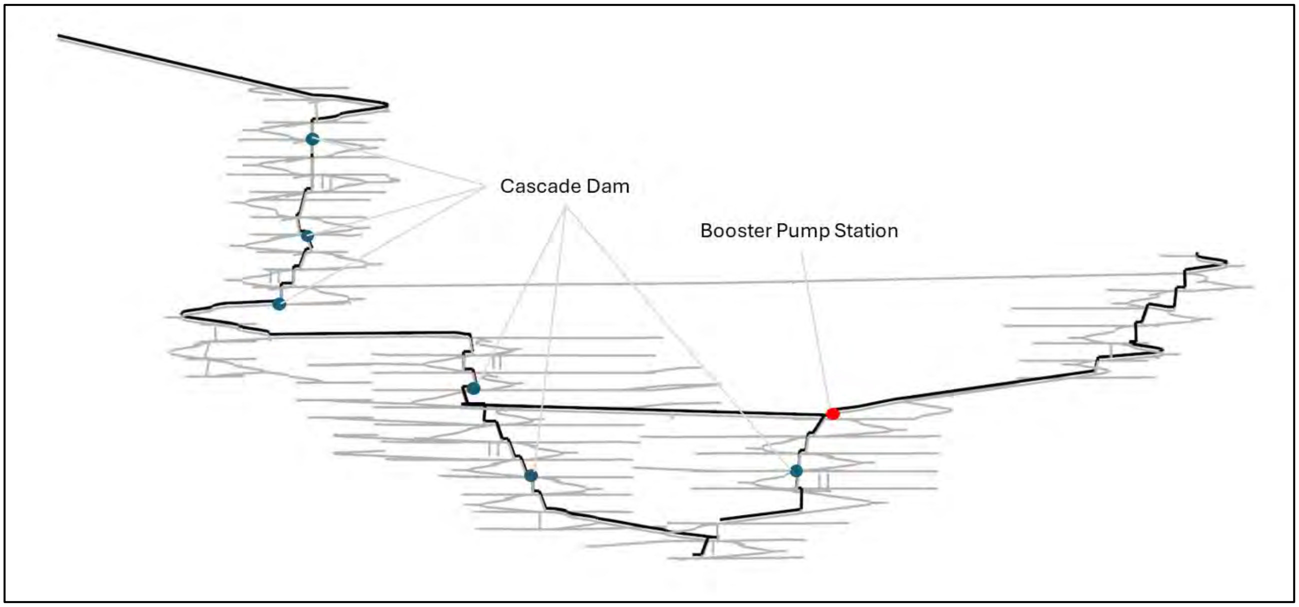


Figure 18-11: Service Water Pipe Routing, Booster Pump Station, and Cascade Dam Location.

Mine Dewatering Infrastructure

The mine dewatering system is a dirty water handling system and will serve to collect and transfer mine water from the production and development areas to the surface water treatment facility. The system has been designed to accommodate the service water load from the mining activities in addition to inflows from the backfill system and a constant ground water inflow rate as per the outcomes of the geohydrological study undertaken for the underground mine.

Table 18-7 presents the dirty water handling requirements for 2030, which is the point in the mines life where the pumping requirement reaches the maximum. The life of mine pumping requirement is presented in Figure Figure 18-13.

Table 18-7: Underground Water Handing (Pumping) Requirement 2030.

Description	Total	Units
Service Water		
Service Water per Day	0.68	ML/day
Peak Service Water Drilling Shift	15.8	l/s
Ground Water		
Ground Water Inflow	21.23	ML/day
Ground Water Inflow	245.72	l/s
Backfill		
Water contained in backfill	0.32	ML/day
Water released from backfill consolidation	0.18	ML/day
Pumping Requirement		
Total Pumping Requirement	21.91	ML/day
Pumping Duty	261.5	l/s
Water Balance		
Total Inflows	21.23	ML/day
Service Water (Recirculation)	-	ML/day
Ground Water	21.23	ML/day
Water In Backfill	0.32	ML/day
Total Outflows	1.18	ML/day
Footwall Loss	0.04	ML/day
Ventilation Loss	0.79	ML/day
Rock Hoisting Loss	0.21	ML/day
Backfill Loss	0.14	ML/day
Total Underground Makeup (-) / Excess (+)	**20.05**	**ML/day**
Total Underground Makeup (-) / Excess (+)	**835.44**	**m^3/hr**

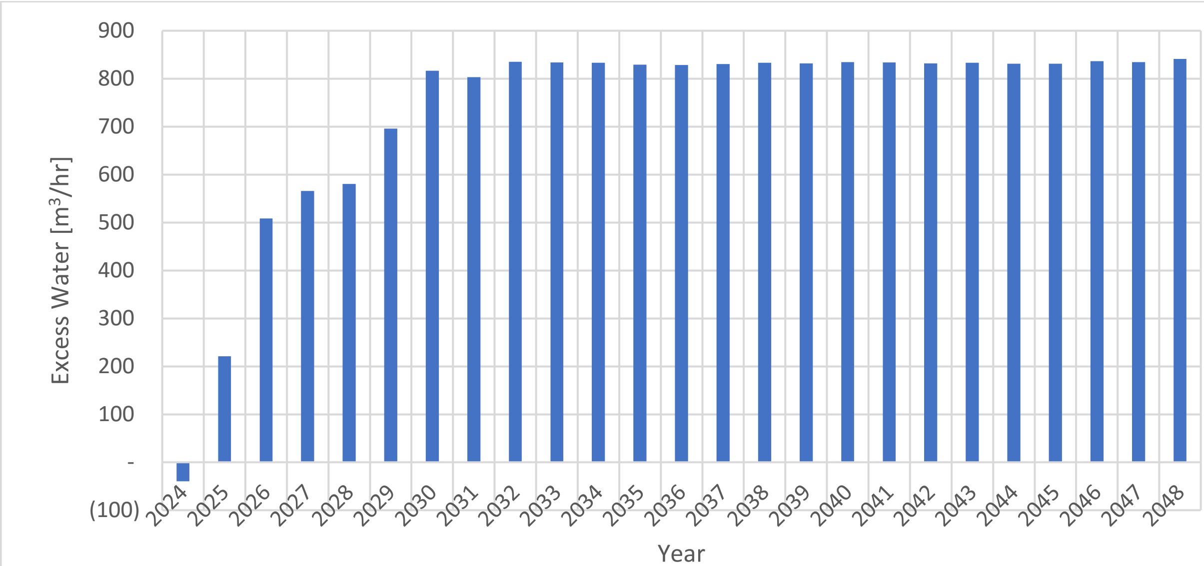


Figure 18-12: Pumping Requirement over Life of Mine.

It is evident from the underground mine water balance that the underground mine has significant quantities of excess water arising from the ground water inflows. The outcomes of the geohydrological study showed that a significant quantity of the ground water inflows is intercepted in the ramp during decline development, while quantities of ground water also extracted from the production workings. Specific dewatering design was employed as to reduce the inflows at the decline and horizontal development ends and manage the water at source.

The primary dewatering system includes five (5) dirty water pump stations, names PS1 to PS5, which transfer all water to the surface water treatment facility. The pump stations primarily service the Zone 1 to Zone 4 requirements in terms of water handling. Water handling requirements for Zone 5 is serviced by means of a drain column to PS5.

The primary dewatering reticulation is installed in the service raises and in the decline. The reticulation (Figure 18-8) includes 400 mm nominal diameter steel piping with 25 bar flanges for PS1 to PS4, 350 mm nominal diameter steel piping with 25 bar flanges for PS5. All piping will be installed in the declines, drives and vertical intake service raises. Piping in the decline and horizontal sections will be supported by means of parrot hanger supports which are anchored into the hanging wall. Certain sections of the reticulation will require bearer support such as the transition points from the vertical service raise to the horizontal development.

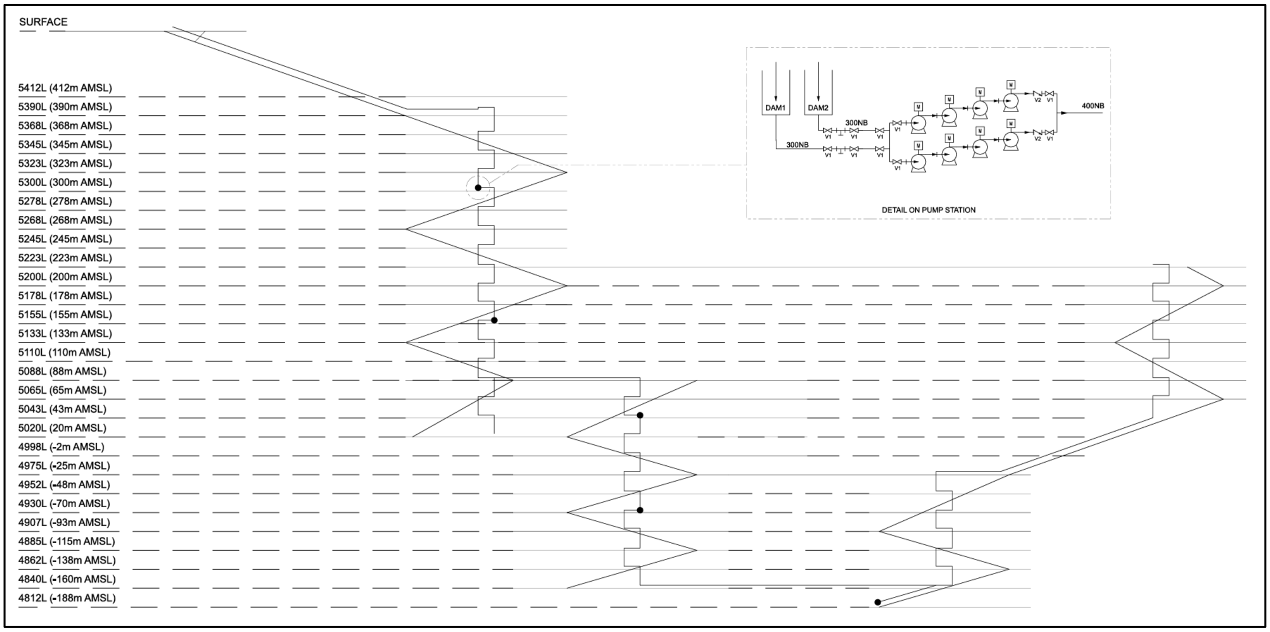


Figure 18-13: Mine Water Handling and Dewatering System Schematic.

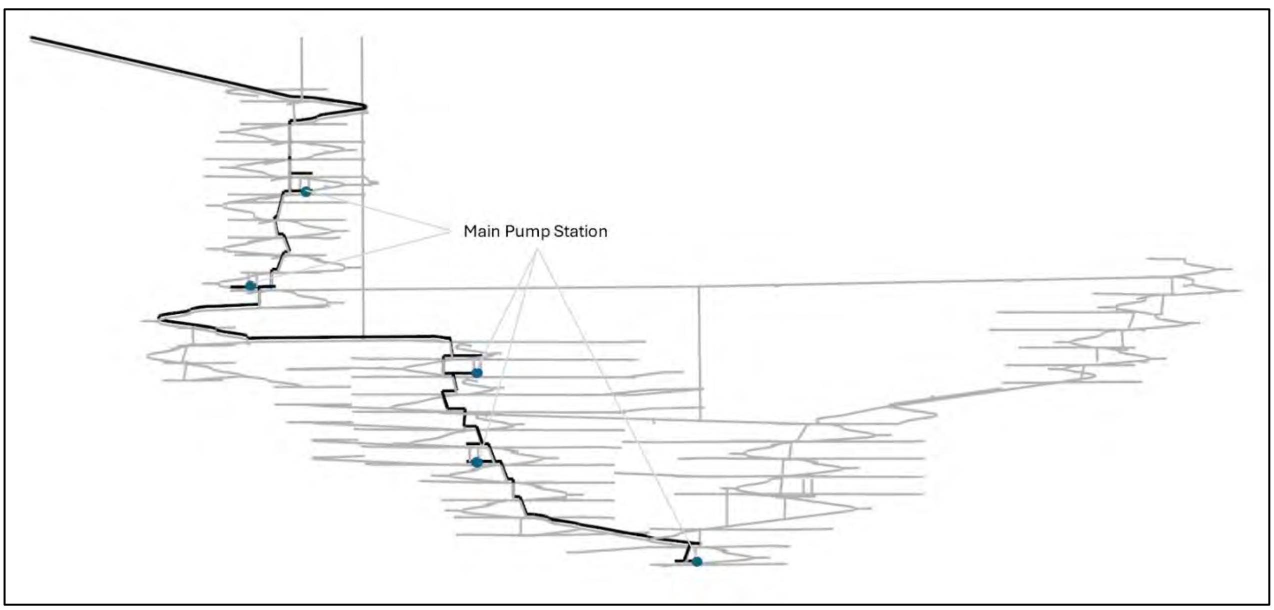


Figure 18-14: Dirty Water Pump Station Locations.

The pump stations will transfer, by means of cascade pumping, the dirty water to surface mine water treatment facilities. The location of the pump stations is presented in Figure 18-14. A general arrangement of the dirty water pump stations is presented in Figure 18-15. Pump stations 2 to 5 are identical in design, while pump stations 1 is similar in design but with an additional pump stage. The pump stations comprise two vertical dams, 5.5 m in diameter each. The dams have conical shaped bottoms to direct solids to the suction. The suction and delivery piping sizes have been selected to attain high fluid velocities as to prevent setting of solids in the system. The suction piping delivers water to a centrifugal pump set and an additional pump set is included as a standby set for planned or unplanned maintenance. The pump station duties are presented in Table 18-8: Dirty Water Pump Station Duties.

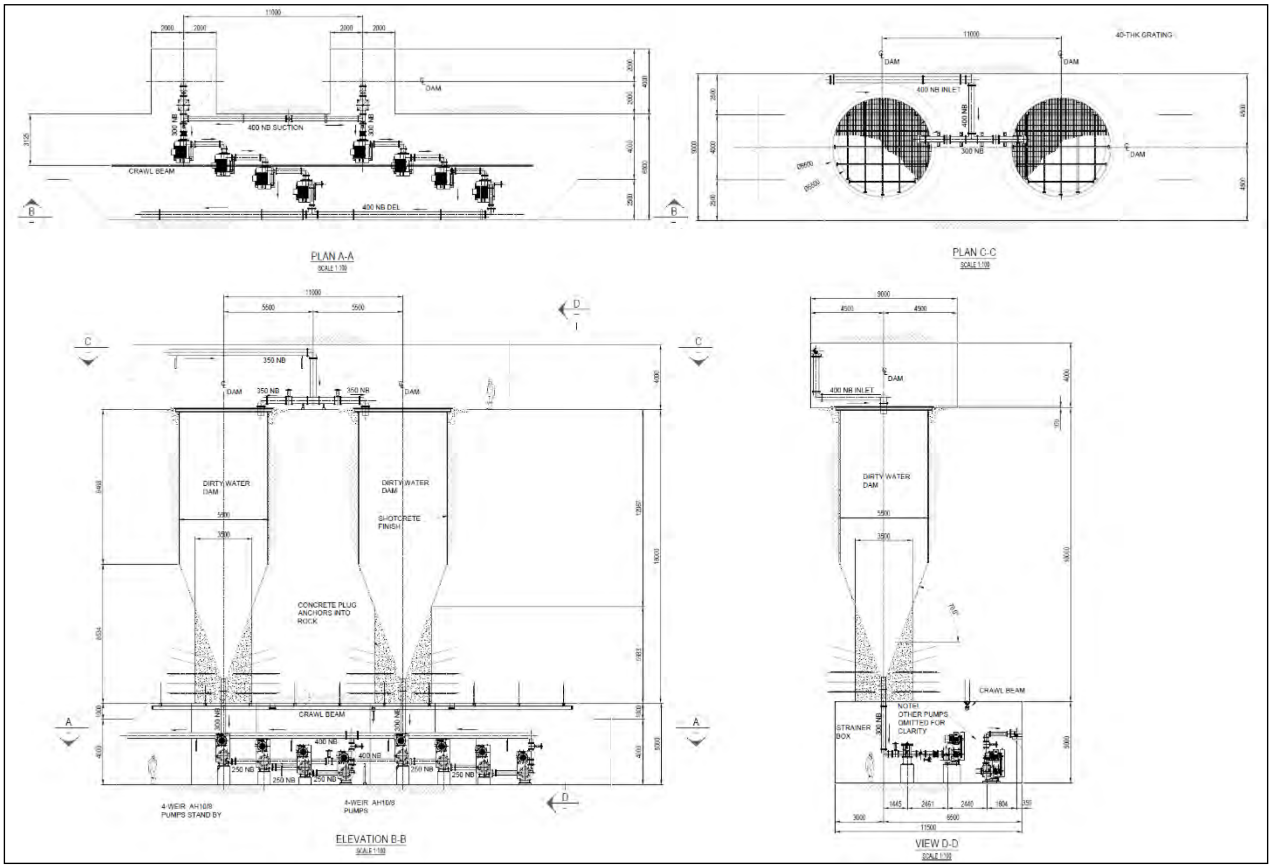


Figure 18-15: Dirty Water Pump General Arrangement.

Table 18-8: Dirty Water Pump Station Duties.

Pump Station	Unit	PS_1	PS_2	PS_3	PS_4	PS_5
Flow Rate	[l/s]	328	328	328	328	220
Density	[kg/m³]	1000	1000	1000	1000	1000
Dynamic Viscosity	[kg/m.s]	1.00E-03	1.00E-03	1.00E-03	1.00E-03	1.00E-03
Pipe Diameter	[mm]	400	400	400	400	350
Pipe Length	[m]	986.6	379.9	904.9	413.8	690.3
Pipe Outer Diameter	[mm]	406.4	406.4	406.4	406.4	355.6
Pipe Wall Thickness	[mm]	10	10	10	10	10
Pipe Inner Diameter	[mm]	386.4	386.4	386.4	386.4	335.6
Fluid Velocity	[m/s]	2.80	2.80	2.80	2.80	2.49
Pipe Roughness	[]	0.02	0.02	0.02	0.02	0.02
Friction Loss	[m]	12.75	4.91	11.70	5.35	8.44
Static Head	[m]	197.9	145.4	111.1	135.0	149.8
Additional Static	[m]	3	3	3	3	3
Frictional Head	[m]	12.8	4.9	11.7	5.3	8.4
Total Head	[m]	213.7	153.3	125.8	143.3	161.3
No in Series	[no]	4	3	3	3	3
No in Parallel	[no]	1	1	1	1	1
Standby Set	[no]	1	1	1	1	1
Total	[no]	8	6	6	6	6
Model	[]	AH 10/8	AH 10/8	AH 10/8	AH 10/8	AH 8/6
Absorbed Power	[kW]	224	214	177	200	147
Installed Power	[kW]	300	280	250	250	185

In addition to the primary dewatering system, the design includes various secondary infrastructures and equipment to collect and transfer water to the primary dewatering system. Dirty water from the production areas will be transferred to dirty water pump stations by means of vertical spindle pumps and drain columns. The ore drives are developed at an inclination of 1 in 200, such that all water from the stoping areas report to the entrance to the ore drive. Vertical spindle pump will transfer this water via the strike drive to a drain column situated in the service raise. The drain column reports directly to the dirty water pump station dam. The drain column is specifically sized for gravity feed and available head, with a drain column for each main pump station.

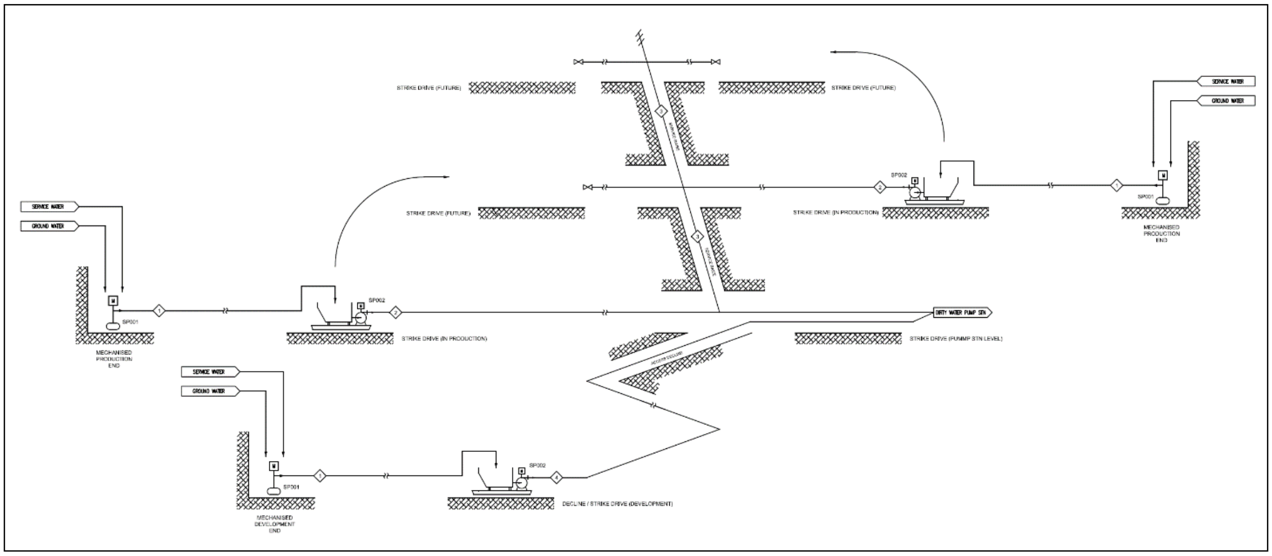


Figure 18-16: Secondary Pumping System Schematic.

Due to the expectation that the ground water inflows will report to the decline, a specific dewatering design needs to be employed as to reduce the inflows at the decline development end. The design includes cover drilling for water ahead of the decline development and the use of mobile skid dams to collect and transfer water to the primary dewatering system. In intervals of 100 m drill cubbies are developed and a 200 m horizontal or slightly inclined hole is drilled in the direction of the decline development from the cubby. The drill hole is cased with a perforated HDPE pipe and any water intercepted reports to the drill cubby where a mobile skid dam transfers the water to the primary pumping system. In total, 16 skid dam units are utilised at the peak dewatering requirement; this quantity is based on consideration of both the flow requirement and the potential location of the inflow with respect to the primary dewatering infrastructure.

Compressed Air Reticulation

Compressed air is solely used for the purposes of refuge bay ventilation as all mining activities will be undertaken by electro-hydraulic drilling. Compressed air requirements are therefore a function of the number of underground mine personnel that will use the refuge bays in an emergency situation. The maximum number of underground employees will occur during the day shift and will is approximately 78 people. It was calculated that in an emergency the refuge bays will be ventilated with approximately 421 CFM of compressed air. Compressed air will be reticulated from the surface compressor house to the underground refuge bays by means of 100 mm nominal diameter steel piping. All pipe reticulation was sized through compressed air flow modelling, such that pressure losses are acceptable throughout the system.

The compressed air reticulation is installed in the service raises and in the decline. All piping in the decline and horizontal sections will be supported by means of parrot hanger supports which are anchored into the hanging wall. Certain sections of the primary backbone will require bearer support such as the transition points from the vertical service raise to the horizontal development.

Backfill Reticulation

Backfill will be supplied to all mining sections via pipelines from the surface plant and discharging directly into the stope. Backfill will be supplied to all five mining zones, these being:

- Zone 1 - 5368 level to 5133 level.
- Zone 2 - 5065 level to 4947 level.
- Zone 3 - 5105 level to 4835 level.
- Zone 4 - 5037 level to 4812 level.
- Zone 5 - 5178 level to 5020 level.

Backfill will be supplied from a backfill plant on surface and will comprise of repulped tailings mixed with a cement binder. The backfill plant is located within the processing plant area, and it will be necessary to transfer the backfill overland to the mine for distribution underground. Restrictions have been set regarding the placement of equipment outside the fenced security area. In an ideal situation the backfill preparation plant would be placed close to the surface vent hole to allow for a pure gravity feed system to distribute the backfill underground. The piping system will operate as a combination of a pumped and gravity feed system using a pump located at the backfill plant to deliver the fill to the mine from where the available potential head will be used to transport the backfill to the stope. The system must operate to provide a steady continuous flow using pipe sizing and pump control to regulate the flow.

At the present time the characteristics of the slurry have not been properly defined and further work will be required to do this. The system flow modelling and design is based on the current expectation that the backfill slurry will be a free settling product with similar characteristics of backfill used in the Witwatersrand mines in South Africa. Changes in the properties of the slurry can affect the design and must be considered during the final design.

The design of a backfill barricade has been proposed based on a wire mesh and lacing design. The barricade will be in the stope access drive and will be used in the construction of a backfill plug. The barricade will allow water to flow through freely during the constructions of the plug. Water drainpipes will be used to drain water from the backfill once the plug has been formed.

Backfill Volume Requirements

Table 18-9 summarises the parameters that were used for calculation of the backfill supply requirements for the reticulation design.

Table 18-9: Backfill Supply Design Parameters.

Description	Value
Density of wet backfill	1.76
Density of solids	2.63
Stope fill factor	100%
Stope height	22.5 m
Stope width	16 m
Stope length	Variable – 26.4 m average
Backfill advance	1.5 m
Filling shifts per day	2
Stope production	1,000 tpd

Based on the daily production requirements it will be necessary to provide sufficient backfill to fill 424 m³ per day. A portion of the water in the backfill delivered into the stope will drain from the fill. This is expected to be approximately 30% of the total volume of the backfill slurry delivered to the stope.

A volumetric flow of 50 m³/h was requested in the mine design. The design for the backfill supply plant has been designed to supply backfill at a fixed rate of 41.9 m³/h. The fixed rate of supply does not match the requirements of the backfill reticulation systemmetallurgical test where variable conditions will occur.

Initial considerations were given to placing storage tanks to act as a buffer between the fixed supply from the plant and the variable flow through the underground distribution piping. This would require the backfill plant to transfer backfill to a tank, fitted with an agitator, ahead of transferring the backfill underground. A separate flushing water tank would also be required. Unfortunately, the required retention time in the tank would allow the binder in the fill to begin hydrating reducing its effectiveness and potential deposition in the pipe walls. This method of controlling the flow would therefore not be acceptable.

Two shifts are available per day. During each shift it will be necessary to make allowances for activities other than filling. These will include travelling hours to and from the workplace and general work preparation. Safety inspections will be required particularly with regards to the backfill barricade and the correct allocation of piping. Pre-filling and post-filling flushing of the backfill pipes are necessary to ensure that the piping system is clear of blockages. The time taken for filling is dependent upon the length of the pipe being flushed. For a 12 hour shift we have estimated that no more than 9 hours would be available for the physical filling procedure. For an estimated system utilization of 75% to 80% a nominal supply rate of 45m³/h would be required.

The designers of the plant have suggested that the plant will be capable of delivering up to 50 m³/h using variable speed drives on the backfill mixer and pump. To implement a backfill reticulation system that will be capable of operating with a limited supply it will be necessary to implement flow control within the piping system. Given the difficulties regarding the ability to accurately control the flow of backfill, a nominal design flow of 45m³/h has been used for the reticulation design. This will give some leeway in the control of flow.

Backfill Flow and Piping Design

Backfill Slurry Characteristics

Slurry characteristics are critical in determining the operating conditions and physical design of the reticulation system. The physical characteristics of the slurry are shown in Table 18-10.

Table 18-10: Backfill Slurry Physical Characteristics.

Slurry density (t/m³)	1.76
Solids density (t/m³)	2.63
Mass concentration	70%
Volume concentration	47%

No flow characteristics of the backfill slurry are currently available. In water systems the pressure losses resulting from pipe friction can be accurately calculated using standard friction loss formulae. In slurry systems it is much more difficult to determine the friction losses and, in addition to this, the settling characteristics of the solids within the slurry are critical to the design. Ideally characteristics are determined by means of loop testing where a bulk sample of the slurry is pumped around a pipe loop to determine friction losses and settling characteristics. Benchtop tests are also used to determine specific characteristics, using a smaller slurry sample, but cannot provide the accuracy of the full loop tests.

The characteristics for the feasibility design have been estimated based on the known qualities of the slurry. However, we recommend that suitable tests be conducted prior to final design and implementation.

During the initial phase of mining in Zone 1 there will not be any significant No flow characteristics of the backfill slurry are currently available. In water systems the pressure losses resulting from pipe friction can be accurately calculated using standard friction loss formulae. In slurry systems it is much more difficult to determine the friction losses and, in addition to this, the settling characteristics of the solids within the slurry are critical to the design. Ideally characteristics are determined by means of loop testing where a bulk sample of the slurry is pumped around a pipe loop to determine friction losses and settling characteristics. Benchtop tests are also used to determine specific characteristics, using a smaller slurry sample, but cannot provide the accuracy of the full loop tests. The characteristics for the feasibility design have been estimated based on the known qualities of the slurry. However, we recommend that suitable tests be conducted prior to final design and implementation.

During the initial phase of mining in Zone 1 there will not be any significant horizontal sections in the underground distribution system and the risk resulting from variations in the flow characteristics are expected to be low. During the initial period of backfilling, it will be possible to evaluate characteristics of the slurry by measuring actual flow rates and comparing these with the design flow. Using the results of the measurements it will be possible to adjust the design if necessary. This will be particularly useful for verifying the design for the potentially higher risk areas at zone 4 and zone 5.

Backfill Flow Modelling Characteristics

The basic underground piping installation consists of piping installed in near vertical boreholes and horizontal pipe sections on each level. The function of the vertical sections is to generate sufficient head to produce flow in the horizontal sections by overcoming friction losses.

The vertical sections generate flow through the differential height between the top and the base of each section. The vertical section in effect acts as a pump supplying the driving force to move the fluid. Friction will be generated in the piping as backfill flows through it. The friction will be dependent upon the characteristics of the fluid and the velocity of the fluid in the pipe. This friction will act against the potential head resulting in an overall reduction in flow and pressure. In the simplest case, of a vertical pipe section having no horizontal component, the potential head would be balanced by the friction head. The flow in the pipeline would reach its terminal velocity and the pressure (static) in the column would be zero. This situation results in the maximum flow that can be developed using gravity feed for a fluid in a specific pipe.

The horizontal pipe sections will dissipate energy in the form of friction losses. Increasing the length of the horizontal pipe will increase the pressure in the pipeline. This will result in a reduction of flow through the system. The available pressure head and the total system losses will balance to produce a steady state flow condition. For a fixed pipe length, reducing the pipe diameter will reduce the flow rate while increasing the diameter will increase the flow rate.

Backfill piping must operate in full flow conditions to make sure that the design flow rate is maintained, and that pipe wear is kept to a minimum. A gravity fed backfill installation is a self-generating flow system which will balance out at a specific flow. If the system is not supplied with backfill at the same rate, then 'free fall' conditions will develop. This occurs when low pressure vapour pockets are formed to replace the shortfall in backfill supplied. In this situation plugs of backfill tend to flow (fall) down the piping. Pipes operating in these conditions have a reduced operating life and are unsuitable for the transportation of backfill over long distances. To ensure that full flow conditions are met the pressure at the top of the shaft column must be positive. When supplying the backfill pipeline a positive pressure head must be available at the top of the pipeline on surface.

The piping system is required to deliver backfill to the stopes within defined limits. The flow modelling for the piping has produced pipe specifications that will be capable of operating at the required flow rates. Variations in the system operating parameters will change the flow rate. Therefore, it is important to ensure that the piping can still operate within the required limits without requiring any major design changes. Such variations can be due to the following parameters:

- Changes in pipe layout.
- Variations in backfill density.
- Pipe wear.
- Backfill settling.

Pipe wear will occur in all backfill pipes and will continue to increase over the life of the pipe. Increased wear on the pipes results in higher flows which can become excessive, particularly in this case where the supply is from a pump. Flow will increase gradually over time, and it will be possible to restrict the flow by installing smaller diameter piping in place of the standard piping. This will act as a choke restricting the flow to acceptable levels. Pipe wear, in the piping, can result in settling of the backfill due to reduced velocities. This can be expected but is not necessarily a problem provided the settling does not increase to an extent where the pipe becomes choked. If the flow rate in the new piping is acceptable then the settling at low velocity should not result in choking as any settling will effectively reduce the pipe diameter enabling the flow velocity to increase.

Characteristics of backfill can vary depending upon the mineralogy, method of preparation and quality of the backfill. It is important that the backfill can settle quickly when deposited in the stope. The backfill piping system must keep the backfill in suspension prior to deposition in the stope. Low flow rate will result in settling of the backfill in the pipes. The particle size distribution indicated that particles of up to 1 mm will be present and we have based the design on a minimum operating flow velocity of 2.5 m/s.

Backfill Pipe Routing

The mine will be accessed from the surface via a decline down to 5390 level. Ramps will then access each level down to 4840 level.

Flow modelling showed that installing backfill pipes in the ramps will result in excessive pressure losses due to the high horizontal to vertical ratio. To reduce the horizontal distances the pipes will be installed in boreholes wherever possible.

It is expected that only one backfilling operation will take place at any one time. However, two piping systems are proposed for this installation to provide an additional system that can be used as an emergency spare or to be available for routing to new stopes and to assist with maintenance requirements.

Figure 18-17: Schematic Showing Underground Backfill Piping Distribution. shows a schematic arrangement of the piping distribution underground. The system will consist of a 'backbone' piping, shown in orange, main which will supply the backfill from surface down to the lowest level requiring backfill. The backfill pipe will be installed from surface down to 5390 level. From here the pipe will be routed to a vent hole linking 5390 level to 5368 level. This will continue down to 5155 level for zone 1. For zone 2 the piping will be extended down to 5020 level. The piping will extend in a similar manner for zone 3 although the overall length of the horizontal sections will increase. The top of zone 4 is on the same elevation as zone 3 although the horizontal section on 4998 level is longer. Zone 5 will be accessed via a connecting drive on 5200 level.

Piping will be installed on the level where the backfill is to be delivered to a stope and is connected to the operating piping main. As only one pipe will be in operation the piping main will be separated on the level and connected to the on-level distribution piping supplying the stope. The pipeline will be dedicated to supplying only one stope at any one time. Once backfilling operations have been completed the on-level piping can be redirected to supply another stope on the same level or the piping main reconnected to supply another level.

Figure 18-17: Schematic Showing Underground Backfill Piping Distribution.

It has already been stated that the preferred layout would be a gravity feed from surface via the ventilation borehole connecting surface to 5390 level. In the current design this will not be used but has been included for comparative purposes. In this option the flow is only restricted to prevent the formation of low pressures within the system. Larger 100 NB piping has been used in addition to 80 NB piping to assist with the supply to zones 4 and 5.

The second layout requires the backfill to be pumped from the backfill preparation plant overland to the top of the borehole. From here the piping is installed in the ventilation borehole from surface down to 5390 level. Pipe flow has been restricted to limit the flow to 45 m^3/h to match the requirement of the pump. Other than the choke piping 80 NB piping has been used throughout.

In the third layout the backfill slurry is pumped from the backfill preparation plant overland to the top of the decline. The piping continues down the decline to 5390 level. Pipe flow has been restricted to limit the flow to 45 m^3/h to match the requirement of the pump. Other than the choke piping, 80 NB piping has been used throughout.

The piping below 5390 level is identical for all three layouts.

The piping will connect to a 75 mm HDPE pipe used to feed the backfill into the stope. The backfill should be delivered to the back of the stope to allow the water to drain to the front of the stope to where the backfill barricade is located on the level below and to the drainpipes at the front of the stope. A second HDPE pipe can be used to distribute the backfill towards the front of the stope when required.

Backfill System Flow and Pressure Analysis

The piping layouts were modelled using backfill slurry flow algorithms, system layouts and piping specifications to predict flow and pressures. A set of summary data sheets showing flow and pressure characteristic for the range of pipe installations are given in Appendix 8.7. The flow algorithms were developed from test data obtained from the analysis of backfill slurries that have characteristics similar to those expected at Dasa. During the modelling, the size and specification of the piping was adjusted to suit the design requirements of the system as closely as possible.

To cover the range of flows that could be expected for each of the mining zones the supply to the top and to the base of each zone was modelled.

Each of the potential layouts were modelled to determine the operating parameters and these are described separately in the following sections. Distribution piping pressures are given for the piping at levels below 5390 as the piping layout is identical for all three layouts below this level.

Backfill Gravity Feed from Surface Ventilation Borehole

The initial layout was based on gravity feed system capable of providing the initial required flow rate of 50 m^3/h. This design was not continued due to the limitations described previously. A summary of the flow and pressure characteristics for this layout are shown in Table 18-11 below.

Table 18-11: Backfill Operating Characteristics – Gravity Feed.

Zone	Piping NB	Level	Flow [m³/h]	Piping Pressure [kPa]
Z1	80	5390	58.6	683
Z1	80	5133	57.0	887
Z2	80	5110	54.8	1,234
Z2	80	4998	51.2	1,774
Z3	80	4975	47.3	2,377
Z3	80	4840	49.8	2,002
Z4	100/80	4975	57.5	6,006
Z4	100/80	4908	57.7	5,993
Z5	100/80	5200	60.5	3,025
Z5	100/80	4043	57.1	3,119

The flow rates are generally above the initial requirement of a supply rate of 50 m^3/h except for zone 3 where the flows are slightly below this value, although, with expected pipe wear it is likely that 50 m^3/h would be achievable.

For Zone 1, 2 and 3 the piping will be 80 NB from surface to the stope entrance. This will be the same piping system supplying all three zones operating in accordance with the mining schedule. The 80 NB piping may be extended to zone 4, this will however result in a drop in the flow, to 43 m^3/h.

To increase the flow above 50 m^3/h, for Zone 4, 100 NB piping has been modelled for the section from surface down to 4975 level. At these flow rates the flow velocity in the 100 NB piping, at 2.7 m/s, will be above the minimum design velocity of 2.5 m/s.

For Zone 5 it will be necessary to install 100 NB along the section connecting Zone 1 and Zone 5 on 5200 level. Minimum flow velocities are close to 2.7 m/s.

Backfill Pumped Feed from Surface Ventilation Borehole

A summary of the flow and pressure characteristics for this layout are shown in the below table.

Table 18-12: Operating Characteristics – Pump to Decline.

Zone	Piping NB	Level	Flow [m³/h]	Piping Pressure [kPa]	Pump Pressure [kPa]
Z1	80	5390	45.0	427	6,950
Z1	80	5133	45.0	1,760	6,650
Z2	80	5110	45.0	1,852	6,650
Z2	80	4998	45.0	1,907	6,650
Z3	80	4975	45.0	2,045	6,770
Z3	80	4840	45.0	1,921	6,650
Z4	80	4975	45.0	4,002	8,750
Z4	80	4908	45.0	3,765	8,500
Z5	80	5200	45.0	6,216	11,840
Z5	80	4043	45.0	5,967	11,600

Based on a nominal flow of 45 m³/h, the flow to all areas will be identical. 80 NB piping is used in all zones. In Zones 1 to 3, 50 NB choke piping is used on all sections, except 4975 level, to prevent the development of sub atmospheric pressures and this has resulted in a common pump pressure for these zones. The longer lengths of horizontal piping in Zones 4 and 5 produce higher pressure losses resulting in higher pressures at the pump.

The pump pressures for this layout are slightly lower than the layout to supply via the ventilation borehole, however the differences are not significant.

Flushing Water

Flushing water is required for both pre-flush and final flush operations. Pre-flushing will ensure that the pipeline is clear and ready for filling. The final flush is critical to flush the backfill out of the piping system. The water used for the flushing has different flow characteristics to the backfill resulting in changes in flow volume. Water has less frictional resistance but is less dense than the backfill, both characteristics will influence the system flow.

During the final flushing the water will begin to displace the backfill as both water and backfill flow along the pipeline. For a gravity fed system, the flow is dependent upon the static head in the system. The reduced pressure available from the water will initially cause the flow rate to reduce due to the less dense water displacing the fill. This is countered to some extent by the lower frictional resistance of the water. Eventually the reduced resistance of the water will overcome the resistance of the backfill and the flow rate will increase above the original flow rate of the backfill until only water is flowing through the pipe. It is important that during the slow down period the backfill does not settle, resulting in blockage.

A gravity system will provide sufficient water, supplied from tanks, to meet the changing system flow requirements provided the piping system linking the tank to the piping does not significantly restrict the flow rate.

A pumped system using a positive displacement pump will always deliver a fixed volume unless the speed of the pump is increased. As with the gravity system the flow characteristics of the system will change as the backfill is displaced by the water. The following sequence describes the pressure and flow characteristics for the flushing of backfill to a stope, at the bottom of Zone 1:

- Backfilling:
  Flow 45 m^3/h – pressure at pump 6,650 kPa.
- Flushing Water in Surface Pipe Section:
  Flow 45 m^3/h – pressure at pump 3,390 kPa.
- Flushing Water to Bottom of Decline:
  Flow 45 m^3/h – pressure at pump 1,760 kPa.
- Flushing Water 5133 Level:
  Flow 45 m^3/h – pressure at pump 1,745 kPa.
- Flushing Water to Stope:
  At a flow of 45 m^3/h sub atmospheric pressures will be developed in the pipeline. To prevent this, it will be necessary to increase the flow to 63 m^3/h at a pump operating pressure of 3,810 kPa.

For points 1 to 4 the flow will remain within the 45 m^3/h but will result in a reduced pressure at the pump due to the reduced frictional resistance in the piping. Towards the end of flushing the water will have displaced most of the backfill reducing the frictional resistance in the horizontal section on the backfilling level. For point 5 the development of vapour cavities will result in the formation of slug flow rather than a continuous flow. The flushing will continue however, and water hammer is likely to occur due to the irregular flow conditions. The introduction of vacuum break valves, located at suitable locations, will introduce air into the system helping to damp the water hammer, but it is unlikely that it will be able to prevent it. Increasing the water flow into the system will prevent this from occurring.

For zones 2 to 5 the outcome will be very similar although the longer horizontal sections leading to zones 4 and 5 will influence the point at which higher flushing flows are required.

To achieve full flow conditions additional flow will also be required for all zones. The current proposed arrangement at the backfill plant would not be capable of providing sufficient flushing flows to meet these requirements. It is not clear at this time if the slug flow during flushing will be a problem as this is only expected to occur during the final phase of flushing. Any problems are likely to be identified during the initial phase of backfilling when supplying Zone 1.

Other than increasing the flow capacity of the positive displacement pump, one method of providing additional flow is to use centrifugal pump to supplement the water from the positive displacement pump. A separate water pipeline would run in parallel to the backfill pipe on surface and connecting at a location where the pipeline pressure is low. Pump operating pressures would be low for the feed to zones 1 to 3. If the water is pumped to the top of the ventilation borehole or to the base of the decline pressures of typically 100 kPa would be required. For Zones 4 and 5 the use a centrifugal pump would be impractical due to the higher system pressures and the requirements for this should be reconsidered when there is better understanding of the slurry characteristics and system operation.

Flow Control

To achieve the required rate of flow from the pump it will be necessary to monitor the flow and pressures in the pipeline. Flow and pressure monitoring will be installed at the pump to determine the pump output conditions.

Additional pressure transducers will be required along the operating pipeline. The primary purpose of these would be to monitor for pressures in the pipeline and to provide feedback for pump flow control. To achieve full flow conditions the pressures in the pipeline must be always positive. Low sub atmospheric pressures are likely to occur at the end of a long horizontal section connecting to a vertical section. For this layout these would primarily be on surface prior to going underground and at the top of Zones 4 and 5.

Low pressures at any one of these locations will require an increase in pump flow to increase the pressure. Excessively high flow requirements, at low pressures, will require additional choke piping sections to be added.

Pipe Design

The calculations are based on the use of seamless piping manufactured to API 5L X42 or A106 Grade B specifications. The allowable working stress used for the pressure rating calculation for the material is 138 MPa. This is significantly below the yield strength of the piping material and will provide a factor of safety above the maximum operating conditions.

Slurry pipes tend to wear unevenly, and failures are likely to occur at particular locations, such as bends, even though the general wall thicknesses are acceptable. In horizontal pipes, the slurry tends to have a greater density at the bottom of the pipes leading to higher pipe wear in this area. The pipes must be monitored regularly by measuring the wall thickness with an ultrasonic gauge and replaced when wear is excessive.

Pipe wear has been estimated based on a wear rate of 20,000 tonnes of slurry per mm of pipe wear to provide an allowance for replacement piping.

Backfill Barricade

A barricade is required at all openings to the stope where it is necessary to contain the backfill. The barricade must be capable of safely retaining the backfill while allowing water to drain freely. Several types of barricade construction have been considered. These include:

- Reinforced concrete walls.
- Wooden barricades.
- Wire lacing and mesh.
- Steel fabrications.
- Waste rock walls.

Whichever method is chosen it is important to ensure that the barricade can be installed quickly and can be adapted to suit a variation in opening size.

Backfill Quantities Affecting Barricade

The tests carried out by Patterson and Cooke (P&C) on the backfill samples showed that for backfills with binder added, water drainage in the order of 56% can be expected. This would result in a placed backfill relative density of approximately 2.1. The backfill strength would be expected to reach its designed strength of 1 MPa after 28 days curing with 6% binder added.

Design Criteria

The barricade will be required to retain the fill entering the stope until a solid backfill plug can be formed at the base of the stope. To achieve this the barricade will be installed in the 4.5 m × 4.5 m stope access drive. The backfill will be build up in layers over several shifts until the access drive is covered. The backfill will then be allowed to cure until the plug has developed sufficient strength to allow the backfilling to continue for the remainder of the stope.

The plug effectively isolates the barricade from the filling area making it redundant as a pressure retaining barricade. This form of barricade has been used successfully in many mines and is the method considered most suitable for the Dasa application.

Barricade Types

The relative qualities of each of the types of barricades was considered:

Reinforced Concrete Walls

Concrete plugs have been used for numerous underground applications, notably underground water retaining bulkheads.

Concrete plugs need to be pinned to surrounding walls and reinforcing steelwork built to give strength to the concrete. Shutters are built outside the reinforcing mesh to form the outside surfaces of the plug. Concrete is pumped into the shuttering to ensure that the volume is completely filled. The concrete is allowed to cure for a set period to ensure that the plug has sufficient strength.

This method can provide a reliable effective bulkhead. There are a few disadvantages of this method. The concrete wall would take some time to build, as numerous pins need to be attached to the walls and the reinforcing structure. Shutters must also be built before the concrete can be poured. The main disadvantage is that the concrete must be given time to cure. This could typically be between 7 and 28 days.

Concrete plugs will prevent the free draining of water from the backfill and must rely on drainpipes alone during the formation of the plug.

Wire Lacing and Mesh

Wire lacing and mesh barricades use wire rope or de-stranded rope attached to eyebolts installed around the circumference of the opening. This creates an interlaced framework on which wire mesh and a geo-fabric material are attached.

This method is relatively inexpensive and will allow relatively free water drainage. Wire lace barricades are typically constructed as a flat barricade which can result in high loads in the wire rope and wall anchors.

Wooden Barricades

Barricades constructed from wood have been used for backfilling applications mainly in tabular ore bodies where paddocks and backfill bags are used. These barricades have been known to fail due to the poor water drainage and the limited strength allowed for by this type of design.

Steel Fabrications

Barricade constructed from steel fabrications can be used to form load bearing members onto which a backfill retaining structure can be attached. These structures need to be attached to the walls either by pinning or by embedding in trenches. This type of structure is relatively expensive and would need to be fabricated for each installation to allow for dimension differences. We are not aware of any barricade design of this type used for underground backfilling.

Waste Rock Piles

Rock can be tipped and pushed to form a rock plug. This method relies on the mass and frictional resistance of the rock pile to provide sufficient resistance against the pressure formed by the backfill.

While this method is relatively cheap, as waste rock can be used to create the pile, a large quantity of rock is required to provide sufficient weight. It is difficult to completely close the tunnel with a rock wall and the upper portion will be difficult to fill. The control of drainage water from behind the barricade could be difficult and may result in pressure build-up behind the barricade.

It has been found in practice that waste rock is not always available when required for the construction of the barricade.

Barricade Design

The primary requirements of the barricade are as follows:

- Capable of resisting the hydraulic head required to form a plug.
- To be free draining preventing pressure build-up behind the barricade.
- Relatively simple to construct.
- Ability to monitor loads on barricade.

Based on the above criteria it was determined that the structural steel construction or wire lacing would be the most appropriate options to be used in the formation of the plug. The cost of the structural option was found to be significantly greater than the wire lacing option. In addition to this the structural option would require components to be specifically sized for the actual excavation potentially increasing the installation time.

An arrangement drawing of the barricade is shown in Appendix 8.1. Eyebolts are installed at suitable locations with specified spacings. Wire rope is threaded through the eyebolts to form the load bearing backbone of the barricade. Expanded metal is then placed against the wire rope. The expanded metal will provide a surface which will support the geotextile fabric used to retain the backfill and permit water to drain from the settled fill.

The most significant feature of this design is the surface profile of the barricade. The barricade is designed to bulge in the middle extending to approximately 1 m beyond the eyebolt anchor points. This is critical in reducing the tensile loads in the wire ropes and anchors and acts in a similar manner to the ropes on a suspension bridge. Initial rope positioning is achieved by attaching the rope to two positioning ropes suspended between the hanging and footwalls. This will assist in attaching the expanded metal and geofabric and minimise the movement of these components during the filling and formation of the plug. The initial placement of the expanded metal should allow for an overlap between sheets of at least 200 mm to allow for expansion movement during filling. The expanded metal sheets and ropes should be secured in position with wire or heavy-duty cable ties.

Loads applied to the barricade can vary considerably during filling depending on the characteristics of the fill settling behind the barricade. The plug height will increase in layers depending upon the size of the plug and the volume of the backfill placed during a shift. Based on the expected flow parameters and the dimensions of a typical stope the rate of rise per filling shift will be less than 0.7 m. During the filling the solids will settle and begin consolidating. A portion of the water in the backfill slurry will remain in the fill while the remainder will be driven to the surface from where it can be drained. During the initial filling process, the loads acting on the barricade will be related to the hydraulic head of the fill. These horizontal loads will decrease as the fill consolidates however, the backfill plug will not be self-supporting until the action of the cement binder has had time to achieve the required strength.

For calculation purposes we have assumed that the friction angle will initially be low but will increase between filling shifts. During filling the barricade can be expected to expand during filling due to pressure loads potentially allowing water or slurry to act on the face of the barricade. This loading is likely to be transitory and is unlikely to act on the full face of the barricade however, this loading was considered and was found to be the worst case under normal conditions. The barricade is not considered to be a permanent structure and will be redundant once the plug has achieved its required strength. For this reason, the factors of safety are lower than would normally be considered for a permanent structure however, the design requires a minimum of factor of safety of two on the UTS of the ropes.

Seismicity can potentially result in liquifying the fill in the plug before it has cured resulting in increased loads. We understand that the risk of seismicity is low but has been considered in the calculations. These show that, in this situation, the factors of safety will be reduced but the barricade is unlikely to fail.

It is important to state that the barricade is designed for the formation of a backfill plug only. Once the plug is formed and is allowed to consolidate and cure to an acceptable strength the barricade should no longer required. The acceptable strength of the backfill plug must be sufficient to support the full weight of the backfill to be placed on top of it.

Barricade Installation Guidelines

An installation procedure should be drawn up by the construction team to suit their requirements and the site conditions. We foresee that the eyebolts and wire ropes will be installed using a telescopic lift. The expanded metal and geofabric must be installed behind the wire rope. The installation of these would be best achieved using scaffolding which would be dismantled on completion. A final section of expanded metal should be slid into position once the work requiring access behind the rope has been completed. The geofabric can be hung as a curtain suspended from the roof attached to eyebolts. The fabric can then be pulled into position using tape/rope attached to the fabric and passed through the expanded metal. Alternatively, an overlapping section of geofabric can be used to provide access during the construction. The geofabric should be sufficiently loose to allow for expansion and should overlap the walls by at least 1.5 m to prevent leakage.

Drainpipes wrapped with a geofabric material must be installed to allow water to drain from the backfill during filling above the plug. We have proposed that two 150 mm diameter, full perforated, twin wall drainage pipes be used although this should be modified to suit site conditions if necessary. The drainpipes are to be suspended between the access points at the top and base of the stope using wire ropes to support the weight. At least two drainpipes should be installed, one at each side of the access to minimise pooling of water. The stope drainpipes pass through the barricade through a hole cut in the expanded metal. Backfill delivered to the back of the stope will result in the formation of a beach profile sloping down to the front of the stope. Water forming on the surface of the fill will drain down the slope to the front of the stope where the drainpipes are installed.

Forming the Backfill Plug

Once the barricade has been completed and inspected the backfill can be poured.

It is recommended that the backfill used to form the plug be a higher-than-normal strength mixture to provide a solid plug.

The rate of filling will be dependent upon the size of the excavation and the settling characteristics of the backfill.

The water must be able to drain freely through the geofabric material without an excessive build-up above the top of the backfill. The percolation and consolidation rates limit the rate of filling. If the filling is faster than the rate of consolidation the fill will shrink away from the hanging wall.

The condition of the barricade must be observed during the filling to ensure that there are no excessive loads applied. Any distortion or failure of any components in the barricade will require the cessation of filling until the problem has been rectified.

The backfill is allowed to fill to up to 1 m above the top of the barricade to prevent leakage in subsequent filling.

Once formed, the plug must be allowed to cure to prevent any possibility of liquefaction. The strength of the backfill plug must also be sufficient to support the weight of backfill to be placed on top of the plug.

Water Drainage

The barricade is designed to allow water to pass freely during filling. Water must not be allowed to collect at the base of the barricade and should flow freely down the access drive away from the stope. A grit trap should be placed at a suitable location to prevent excessive grit from entering the mine's water handling system.

After the barricade has been covered most of the water draining through the pipes will be water that has collected at the top of the backfill.

The barricade must be monitored following completion of the backfilling of the stope. This is to determine whether water from fissure or other backfill stopes are draining into the redundant stope.

Alternative arrangements should be available to remove water from the stope should the drainpipes become blocked. This would typically be done using submersible pumps suspended from the top of the stope.

Monitoring Barricade Condition

Detailed procedures must be drawn up to assist in the construction, operation, and safety during the development of the backfill plug and subsequent filling. We recommend that no access should be permitted to any area affected by a potential failure of the barricade until it is considered that the risks are acceptable.

A full inspection must take place between each filling shift to ensure that conditions are acceptable before allowing filling to continue. It is important to ensure that water has drained from the fill before access to the barricade is permitted for the continuation of the inspection.

During filling the barricade will deform under load. Any significant distortion could be the result of excessive loading. In this type of situation, the backfilling must be stopped until the cause is determined, and corrective action taken. If the barricade is in danger of failing due to broken ropes or failed eyebolts, or if any of the elements are seen to have failed, then any areas operating close to and below the barricade must be evacuated until the area is declared safe.

Backfill Flow Modelling Characteristics

The basic underground piping installation consists of piping installed in near vertical boreholes and horizontal pipe sections on each level. The function of the vertical sections is to generate sufficient head to produce flow in the horizontal sections by overcoming friction losses.

The vertical sections generate flow through the differential height between the top and the base of each section. The vertical section in effect acts as a pump supplying the driving force to move the fluid. Friction will be generated in the piping as backfill flows through it. The friction will be dependent upon the characteristics of the fluid and the velocity of the fluid in the pipe. This friction will act against the potential head resulting in an overall reduction in flow and pressure. In the simplest case, of a vertical pipe section having no horizontal component, the potential head would be balanced by the friction head. The flow in the pipeline would reach its terminal velocity and the pressure (static) in the column would be zero. This situation results in the maximum flow that can be developed using gravity feed for a fluid in a specific pipe.

The horizontal pipe sections will dissipate energy in the form of friction losses. Increasing the length of the horizontal pipe will increase the pressure in the pipeline. This will result in a reduction of flow through the system. The available pressure head and the total system losses will balance to produce a steady state flow condition. For a fixed pipe length, reducing the pipe diameter will reduce the flow rate while increasing the diameter will increase the flow rate.

Backfill piping must operate in full flow conditions to make sure that the design flow rate is maintained, and that pipe wear is kept to a minimum. A gravity fed backfill installation is a self-generating flow system which will balance out at a specific flow. If the system is not supplied with backfill at the same rate, then 'free fall' conditions will develop. This occurs when low pressure vapour pockets are formed to replace the shortfall in backfill supplied. In this situation plugs of backfill tend to flow (fall) down the piping. Pipes operating in these conditions have a reduced operating life and are unsuitable for the transportation of backfill over long distances. To ensure that full flow conditions are met the pressure at the top of the shaft column must be positive. When supplying the backfill pipeline a positive pressure head must be available at the top of the pipeline on surface.

The piping system is required to deliver backfill to the stopes within defined limits. The flow modelling for the piping has produced pipe specifications that will be capable of operating at the required flow rates. Variations in the system operating parameters will change the flow rate.

Therefore, it is important to ensure that the piping can still operate within the required limits without requiring any major design changes. Such variations can be due to the following parameters:

- Changes in pipe layout.
- Variations in backfill density.
- Pipe wear.
- Backfill settling.

Pipe wear will occur in all backfill pipes and will continue to increase over the life of the pipe. Increased wear on the pipes results in higher flows which can become excessive, particularly in this case where the supply is from a pump. Flow will increase gradually over time, and it will be possible to restrict the flow by installing smaller diameter piping in place of the standard piping. This will act as a choke restricting the flow to acceptable levels. Pipe wear, in the piping, can result in settling of the backfill due to reduced velocities. This can be expected but is not necessarily a problem provided the settling does not increase to an extent where the pipe becomes choked. If the flow rate in the new piping is acceptable then the settling at low velocity should not result in choking as any settling will effectively reduce the pipe diameter enabling the flow velocity to increase.

Characteristics of backfill can vary depending upon the mineralogy, method of preparation and quality of the backfill. It is important that the backfill can settle quickly when deposited in the stope. The backfill piping system must keep the backfill in suspension prior to deposition in the stope. Low flow rate will result in settling of the backfill in the pipes. The particle size distribution indicated that particles of up to 1 mm will be present and we have based the design on a minimum operating flow velocity of 2.5 m/s.

Backfill Pipe Routing

The mine will be accessed from the surface via a decline down to 5390 level. Ramps will then access each level down to 4840 level.

Flow modelling showed that installing backfill pipes in the ramps will result in excessive pressure losses due to the high horizontal to vertical ratio. To reduce the horizontal distances the pipes will be installed in boreholes wherever possible.

It is expected that only one backfilling operation will take place at any one time. However, two piping systems are proposed for this installation to provide an additional system that can be used as an emergency spare or to be available for routing to new stopes and to assist with maintenance requirements.

Figure 18-17 shows a schematic arrangement of the piping distribution underground. The system will consist of a 'backbone' piping, shown in orange, main which will supply the backfill from surface down to the lowest level requiring backfill. The backfill pipe will be installed from surface down to 5390 level. From here the pipe will be routed to a vent hole linking 5390 level to 5368 level. This will continue down to 5155 level for Zone 1. For Zone 2 the piping will be extended down to 5020 level. The piping will extend in a similar manner for Zone 3 although the overall length of the horizontal sections will increase. The top of Zone 4 is on the same elevation as zone 3 although the horizontal section on 4998 level is longer. Zone 5 will be accessed via a connecting drive on 5200 level.

Piping will be installed on the level where the backfill is to be delivered to a stope and is connected to the operating piping main. As only one pipe will be in operation the piping main will be separated on the level and connected to the on-level distribution piping supplying the stope. The pipeline will be dedicated to supplying only one stope at any one time. Once backfilling operations have been completed the on-level piping can be redirected to supply another stope on the same level or the piping main reconnected to supply another level.

Figure 18-18: Schematic Showing Underground Backfill Piping Distribution.

It has already been stated that the preferred layout would be a gravity feed from surface via the ventilation borehole connecting surface to 5390 level. In the current design this will not be used but has been included for comparative purposes. In this option the flow is only restricted to prevent the formation of low pressures within the system. Larger 100 NB piping has been used in addition to 80 NB piping to assist with the supply to Zones 4 and 5.

The second layout requires the backfill to be pumped from the backfill preparation plant overland to the top of the borehole. From here the piping is installed in the ventilation borehole from surface down to 5390 level.

Pipe flow has been restricted to limit the flow to 45 m³/h to match the requirement of the pump. Other than the choke piping 80 NB piping has been used throughout.

In the third layout the backfill slurry is pumped from the backfill preparation plant overland to the top of the decline. The piping continues down the decline to 5390 level. Pipe flow has been restricted to limit the flow to 45 m³/h to match the requirement of the pump. Other than the choke piping, 80 NB piping has been used throughout.

The piping below 5390 level is identical for all three layouts.

The piping will connect to a 75 mm HDPE pipe used to feed the backfill into the stope. The backfill should be delivered to the back of the stope to allow the water to drain to the front of the stope to where the backfill barricade is located on the level below and to the drainpipes at the front of the stope. A second HDPE pipe can be used to distribute the backfill towards the front of the stope when required.

Backfill System Flow and Pressure Analysis

The piping layouts were modelled using backfill slurry flow algorithms, system layouts and piping specifications to predict flow and pressures. A set of summary data sheets showing flow and pressure characteristic for the range of pipe installations are given in Appendix 8.7. The flow algorithms were developed from test data obtained from the analysis of backfill slurries that have characteristics similar to those expected at Dasa. During the modelling, the size and specification of the piping was adjusted to suit the design requirements of the system as closely as possible.

To cover the range of flows that could be expected for each of the mining zones the supply to the top and to the base of each zone was modelled.

Each of the potential layouts were modelled to determine the operating parameters and these are described separately in the following sections. Distribution piping pressures are given for the piping at levels below 5390 as the piping layout is identical for all three layouts below this level.

Backfill Gravity Feed from Surface Ventilation Borehole

The initial layout was based on gravity feed system capable of providing the initial required flow rate of 50 m³/h. This design was not continued due to the limitations described previously. A summary of the flow and pressure characteristics for this layout are shown in Table 18-13 below.

Table 18-13: Backfill Operating Characteristics – Gravity Feed

Zone	Piping NB	Level	Flow [m³/h]	Piping Pressure [kPa]
Z1	80	5390	58.6	683
Z1	80	5133	57.0	887
Z2	80	5110	54.8	1,234
Z2	80	4998	51.2	1,774
Z3	80	4975	47.3	2,377
Z3	80	4840	49.8	2,002
Z4	100/80	4975	57.5	6,006
Z4	100/80	4908	57.7	5,993
Z5	100/80	5200	60.5	3,025
Z5	100/80	4043	57.1	3,119

The flow rates are generally above the initial requirement of a supply rate of 50 m^3/h except for zone 3 where the flows are slightly below this value, although, with expected pipe wear it is likely that 50 m^3/h would be achievable.

For zone 1, 2 and 3 the piping will be 80 NB from surface to the stope entrance. This will be the same piping system supplying all three zones operating in accordance with the mining schedule. The 80 NB piping may be extended to zone 4, this will however result in a drop in the flow, to 43 m^3/h.

To increase the flow above 50 m^3/h, for zone 4, 100 NB piping has been modelled for the section from surface down to 4975 level. At these flow rates the flow velocity in the 100 NB piping, at 2.7 m/s, will be above the minimum design velocity of 2.5 m/s.

For Zone 5 it will be necessary to install 100 NB along the section connecting zone 1 and zone 5 on 5200 level. Minimum flow velocities are close to 2.7 m/s.

Backfill Pumped Feed from Surface Ventilation Borehole

A summary of the flow and pressure characteristics for this layout are shown in Table 18-14 below.

Table 18-14: Operating Characteristics – Pump to Decline

Zone	Piping NB	Level	Flow [m³/h]	Piping Pressure [kPa]	Pump Pressure [kPa]
Z1	80	5390	45.0	427	6,950
Z1	80	5133	45.0	1,760	6,650
Z2	80	5110	45.0	1,852	6,650
Z2	80	4998	45.0	1,907	6,650
Z3	80	4975	45.0	2,045	6,770
Z3	80	4840	45.0	1,921	6,650
Z4	80	4975	45.0	4,002	8,750
Z4	80	4908	45.0	3,765	8,500
Z5	80	5200	45.0	6,216	11,840
Z5	80	4043	45.0	5,967	11,600

Based on a nominal flow of 45 m³/h, the flow to all areas will be identical. 80 NB piping is used in all zones. In zones 1 to 3 50 NB choke piping is used on all sections, except 4975 level, to prevent the development of sub atmospheric pressures and this has resulted in a common pump pressure for these zones. The longer lengths of horizontal piping in zones 4 and 5 produce higher pressure losses resulting in higher pressures at the pump.

The pump pressures for this layout are slightly lower than the layout to supply via the ventilation borehole, however the differences are not significant.

Flushing water

Flushing water is required for both pre-flush and final flush operations. Pre-flushing will ensure that the pipeline is clear and ready for filling. The final flush is critical to flush the backfill out of the piping system. The water used for the flushing has different flow characteristics to the backfill resulting in changes in flow volume. Water has less frictional resistance but is less dense than the backfill, both characteristics will influence the system flow.

During the final flushing the water will begin to displace the backfill as both water and backfill flow along the pipeline. For a gravity fed system, the flow is dependent upon the static head in the system. The reduced pressure available from the water will initially cause the flow rate to reduce due to the less dense water displacing the fill. This is countered to some extent by the lower frictional resistance of the water. Eventually the reduced resistance of the water will overcome the resistance of the backfill and the flow rate will increase above the original flow rate of the backfill until only water is flowing through the pipe. It is important that during the slow down period the backfill does not settle, resulting in blockage.

A gravity system will provide sufficient water, supplied from tanks, to meet the changing system flow requirements provided the piping system linking the tank to the piping does not significantly restrict the flow rate.

A pumped system using a positive displacement pump will always deliver a fixed volume unless the speed of the pump is increased. As with the gravity system the flow characteristics of the system will change as the backfill is displaced by the water. The following sequence describes the pressure and flow characteristics for the flushing of backfill to a stope at the bottom of zone 1:

- Backfilling:
  Flow 45 m³/h – pressure at pump 6,650 kPa.
- Flushing water in surface pipe section:
  Flow 45 m³/h – pressure at pump 3,390 kPa.
- Flushing water to bottom of decline:
  Flow 45 m³/h – pressure at pump 1,760 kPa.
- Flushing water 5133 level:
  Flow 45 m³/h – pressure at pump 1,745 kPa.
- Flushing water to stope:
  At a flow of 45 m³/h sub atmospheric pressures will be developed in the pipeline. To prevent this, it will be necessary to increase the flow to 63 m³/h at a pump operating pressure of 3,810 kPa.

For points 1 to 4 the flow will remain within the 45 m³/h but will result in a reduced pressure at the pump due to the reduced frictional resistance in the piping. Towards the end of flushing the water will have displaced most of the backfill reducing the frictional resistance in the horizontal section on the backfilling level. For point 5 the development of vapour cavities will result in the formation of slug flow rather than a continuous flow. The flushing will continue however, and water hammer is likely to occur due to the irregular flow conditions. The introduction of vacuum break valves, located at suitable locations, will introduce air into the system helping to damp the water hammer, but it is unlikely that it will be able to prevent it. Increasing the water flow into the system will prevent this from occurring.

For zones 2 to 5 the outcome will be very similar although the longer horizontal sections leading to zones 4 and 5 will influence the point at which higher flushing flows are required.

To achieve full flow conditions additional flow will also be required for all zones. The current proposed arrangement at the backfill plant would not be capable of providing sufficient flushing flows to meet these requirements. It is not clear at this time if the slug flow during flushing will be a problem as this is only expected to occur during the final phase of flushing. Any problems are likely to be identified during the initial phase of backfilling when supplying Zone 1.

Other than increasing the flow capacity of the positive displacement pump, one method of providing additional flow is to use centrifugal pump to supplement the water from the positive displacement pump. A separate water pipeline would run in parallel to the backfill pipe on surface and connecting at a location where the pipeline pressure is low. Pump operating pressures would be low for the feed to Zones 1 to 3. If the water is pumped to the top of the ventilation borehole or to the base of the decline pressures of typically 100 kPa would be required. For zones 4 and 5 the use a centrifugal pump would be impractical due to the higher system pressures and the requirements for this should be reconsidered when there is better understanding of the slurry characteristics and system operation.

Flow Control

To achieve the required rate of flow from the pump it will be necessary to monitor the flow and pressures in the pipeline. Flow and pressure monitoring will be installed at the pump to determine the pump output conditions.

Additional pressure transducers will be required along the operating pipeline. The primary purpose of these would be to monitor for pressures in the pipeline and to provide feedback for pump flow control. To achieve full flow conditions the pressures in the pipeline must be always positive. Low sub atmospheric pressures are likely to occur at the end of a long horizontal section connecting to a vertical section. For this layout these would primarily be on surface prior to going underground and at the top of zones 4 and 5.

Low pressures at any one of these locations will require an increase in pump flow to increase the pressure. Excessively high flow requirements, at low pressures, will require additional choke piping sections to be added.

Pipe Design

The calculations are based on the use of seamless piping manufactured to API 5L X42 or A106 Grade B specifications. The allowable working stress used for the pressure rating calculation for the material is 138 MPa. This is significantly below the yield strength of the piping material and will provide a factor of safety above the maximum operating conditions.

Slurry pipes tend to wear unevenly, and failures are likely to occur at particular locations, such as bends, even though the general wall thicknesses are acceptable. In horizontal pipes, the slurry tends to have a greater density at the bottom of the pipes leading to higher pipe wear in this area. The pipes must be monitored regularly by measuring the wall thickness with an ultrasonic gauge and replaced when wear is excessive.

Pipe wear has been estimated based on a wear rate of 20,000 tonnes of slurry per mm of pipe wear to provide an allowance for replacement piping.

Backfill Barricade

A barricade is required at all openings to the stope where it is necessary to contain the backfill. The barricade must be capable of safely retaining the backfill while allowing water to drain freely. Several types of barricade construction have been considered. These include:

- Reinforced concrete walls.
- Wooden barricades.
- Wire lacing and mesh.
- Steel fabrications.
- Waste rock walls.

Whichever method is chosen it is important to ensure that the barricade can be installed quickly and can be adapted to suit a variation in opening size.

Backfill Quantities Affecting Barricade

The tests carried out by Patterson and Cooke (P&C) on the backfill samples showed that for backfills with binder added, water drainage in the order of 56% can be expected. This would result in a placed backfill relative density of approximately 2.1. The backfill strength would be expected to reach its designed strength of 1 MPa after 28 days curing with 6% binder added.

Design Criteria

The barricade will be required to retain the fill entering the stope until a solid backfill plug can be formed at the base of the stope. To achieve this the barricade will be installed in the 4.5 m by 4.5 m stope access drive. The backfill will be build up in layers over several shifts until the access drive is covered. The backfill will then be allowed to cure until the plug has developed sufficient strength to allow the backfilling to continue for the remainder of the stope.

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Waste Rock Piles

Rock can be tipped and pushed to form a rock plug. This method relies on the mass and frictional resistance of the rock pile to provide sufficient resistance against the pressure formed by the backfill.

While this method is relatively cheap, as waste rock can be used to create the pile, a large quantity of rock is required to provide sufficient weight. It is difficult to completely close the tunnel with a rock wall and the upper portion will be difficult to fill. The control of drainage water from behind the barricade could be difficult and may result in pressure build-up behind the barricade.

It has been found in practice that waste rock is not always available when required for the construction of the barricade.

Barricade Design

The primary requirements of the barricade are:

- Capable of resisting the hydraulic head required to form a plug.
- To be free draining preventing pressure build-up behind the barricade.
- Relatively simple to construct.
- Ability to monitor loads on barricade.

Based on the above criteria it was determined that the structural steel construction or wire lacing would be the most appropriate options to be used in the formation of the plug. The cost of the structural option was found to be significantly greater than the wire lacing option. In addition to this the structural option would require components to be specifically sized for the actual excavation potentially increasing the installation time.

An arrangement drawing of the barricade is shown in Appendix 8.1. Eyebolts are installed at suitable locations with specified spacings. Wire rope is threaded through the eyebolts to form the load bearing backbone of the barricade. Expanded metal is then placed against the wire rope. The expanded metal will provide a surface which will support the geotextile fabric used to retain the backfill and permit water to drain from the settled fill.

The most significant feature of this design is the surface profile of the barricade. The barricade is designed to bulge in the middle extending to approximately 1 m beyond the eyebolt anchor points. This is critical in reducing the tensile loads in the wire ropes and anchors and acts in a similar manner to the ropes on a suspension bridge. Initial rope positioning is achieved by attaching the rope to two positioning ropes suspended between the hanging and footwalls. This will assist in attaching the expanded metal and geofabric and minimise the movement of these components during the filling and formation of the plug. The initial placement of the expanded metal should allow for an overlap between sheets of at least 200 mm to allow for expansion movement during filling. The expanded metal sheets and ropes should be secured in position with wire or heavy-duty cable ties.

Loads applied to the barricade can vary considerably during filling depending on the characteristics of the fill settling behind the barricade. The plug height will increase in layers depending upon the size of the plug and the volume of the backfill placed during a shift. Based on the expected flow parameters and the dimensions of a typical stope the rate of rise per filling shift will be less than 0.7 m. During the filling the solids will settle and begin consolidating. A portion of the water in the backfill slurry will remain in the fill while the remainder will be driven to the surface from where it can be drained. During the initial filling process, the loads acting on the barricade will be related to the hydraulic head of the fill.

These horizontal loads will decrease as the fill consolidates however, the backfill plug will not be self-supporting until the action of the cement binder has had time to achieve the required strength.

For calculation purposes we have assumed that the friction angle will initially be low but will increase between filling shifts. During filling the barricade can be expected to expand during filling due to pressure loads potentially allowing water or slurry to act on the face of the barricade. This loading is likely to be transitory and is unlikely to act on the full face of the barricade however, this loading was considered and was found to be the worst case under normal conditions. The barricade is not considered to be a permanent structure and will be redundant once the plug has achieved its required strength. For this reason, the factors of safety are lower than would normally be considered for a permanent structure however, the design requires a minimum of factor of safety of two on the UTS of the ropes.

Seismicity can potentially result in liquifying the fill in the plug before it has cured resulting in increased loads. We understand that the risk of seismicity is low but has been considered in the calculations. These show that, in this situation, the factors of safety will be reduced but the barricade is unlikely to fail.

It is important to state that the barricade is designed for the formation of a backfill plug only. Once the plug is formed and is allowed to consolidate and cure to an acceptable strength the barricade should no longer required. The acceptable strength of the backfill plug must be sufficient to support the full weight of the backfill to be placed on top of it.

Barricade Installation Guidelines

An installation procedure should be drawn up by the construction team to suit their requirements and the site conditions. We foresee that the eyebolts and wire ropes will be installed using a telescopic lift. The expanded metal and geofabric must be installed behind the wire rope. The installation of these would be best achieved using scaffolding which would be dismantled on completion. A final section of expanded metal should be slid into position once the work requiring access behind the rope has been completed. The geofabric can be hung as a curtain suspended from the roof attached to eyebolts. The fabric can then be pulled into position using tape/rope attached to the fabric and passed through the expanded metal. Alternatively, an overlapping section of geofabric can be used to provide access during the construction. The geofabric should be sufficiently loose to allow for expansion and should overlap the walls by at least 1.5 m to prevent leakage.

Drainpipes wrapped with a geofabric material must be installed to allow water to drain from the backfill during filling above the plug. We have proposed that two 150 mm diameter, full perforated, twin wall drainage pipes be used although this should be modified to suit site conditions if necessary. The drainpipes are to be suspended between the access points at the top and base of the stope using wire ropes to support the weight. At least two drainpipes should be installed, one at each side of the access to minimise pooling of water. The stope drainpipes pass through the barricade through a hole cut in the expanded metal. Backfill delivered to the back of the stope will result in the formation of a beach profile sloping down to the front of the stope. Water forming on the surface of the fill will drain down the slope to the front of the stope where the drainpipes are installed.

Forming the Backfill Plug

Once the barricade has been completed and inspected the backfill can be poured.

It is recommended that the backfill used to form the plug be a higher-than-normal strength mixture to provide a solid plug.

The rate of filling will be dependent upon the size of the excavation and the settling characteristics of the backfill.

The water must be able to drain freely through the geofabric material without an excessive build-up above the top of the backfill. The percolation and consolidation rates limit the rate of filling. If the filling is faster than the rate of consolidation the fill will shrink away from the hanging wall.

The condition of the barricade must be observed during the filling to ensure that there are no excessive loads applied. Any distortion or failure of any components in the barricade will require the cessation of filling until the problem has been rectified.

The backfill is allowed to fill to up to 1 m above the top of the barricade to prevent leakage in subsequent filling.

Once formed, the plug must be allowed to cure to prevent any possibility of liquefaction. The strength of the backfill plug must also be sufficient to support the weight of backfill to be placed on top of the plug.

Water Drainage

The barricade is designed to allow water to pass freely during filling. Water must not be allowed to collect at the base of the barricade and should flow freely down the access drive away from the stope. A grit trap should be placed at a suitable location to prevent excessive grit from entering the mine's water handling system.

After the barricade has been covered most of the water draining through the pipes will be water that has collected at the top of the backfill.

The barricade must be monitored following completion of the backfilling of the stope. This is to determine whether water from fissure or other backfill stopes are draining into the redundant stope.

Alternative arrangements should be available to remove water from the stope should the drainpipes become blocked. This would typically be done using submersible pumps suspended from the top of the stope.

Monitoring Barricade Condition

Detailed procedures must be drawn up to assist in the construction, operation, and safety during the development of the backfill plug and subsequent filling. We recommend that no access should be permitted to any area affected by a potential failure of the barricade until it is considered that the risks are acceptable.

A full inspection must take place between each filling shift to ensure that conditions are acceptable before allowing filling to continue. It is important to ensure that water has drained from the fill before access to the barricade is permitted for the continuation of the inspection.

During filling the barricade will deform under load. Any significant distortion could be the result of excessive loading. In this type of situation, the backfilling must be stopped until the cause is determined, and corrective action taken. If the barricade is in danger of failing due to broken ropes or failed eyebolts, or if any of the elements are seen to have failed, then any areas operating close to and below the barricade must be evacuated until the area is declared safe.

Electrical Infrastructure and Reticulation

Two separate underground feeders will provide supply redundancy to the underground operation with both rated to cater for the full load requirements. One of the feeders will be installed through the portal down the decline system to Zone 1. The second underground feeder will be installed down the down cast equipped ventilation shaft to Zone 1. From the production levels the MV reticulation will be installed in the service raises with each feeder accessing alternating main station levels.

Figure 18-19 provides an overview of the underground MV reticulation at the extent of the mining operation.

Suitably rated Mini-substation Units (MSU) and transformers will be installed at the load centres to provide the equipment operational voltage of 1000 V and 400 V. The mining and ventilation equipment will be supplied at 1000 V, with pumping, small power and lighting operated at 400 V.

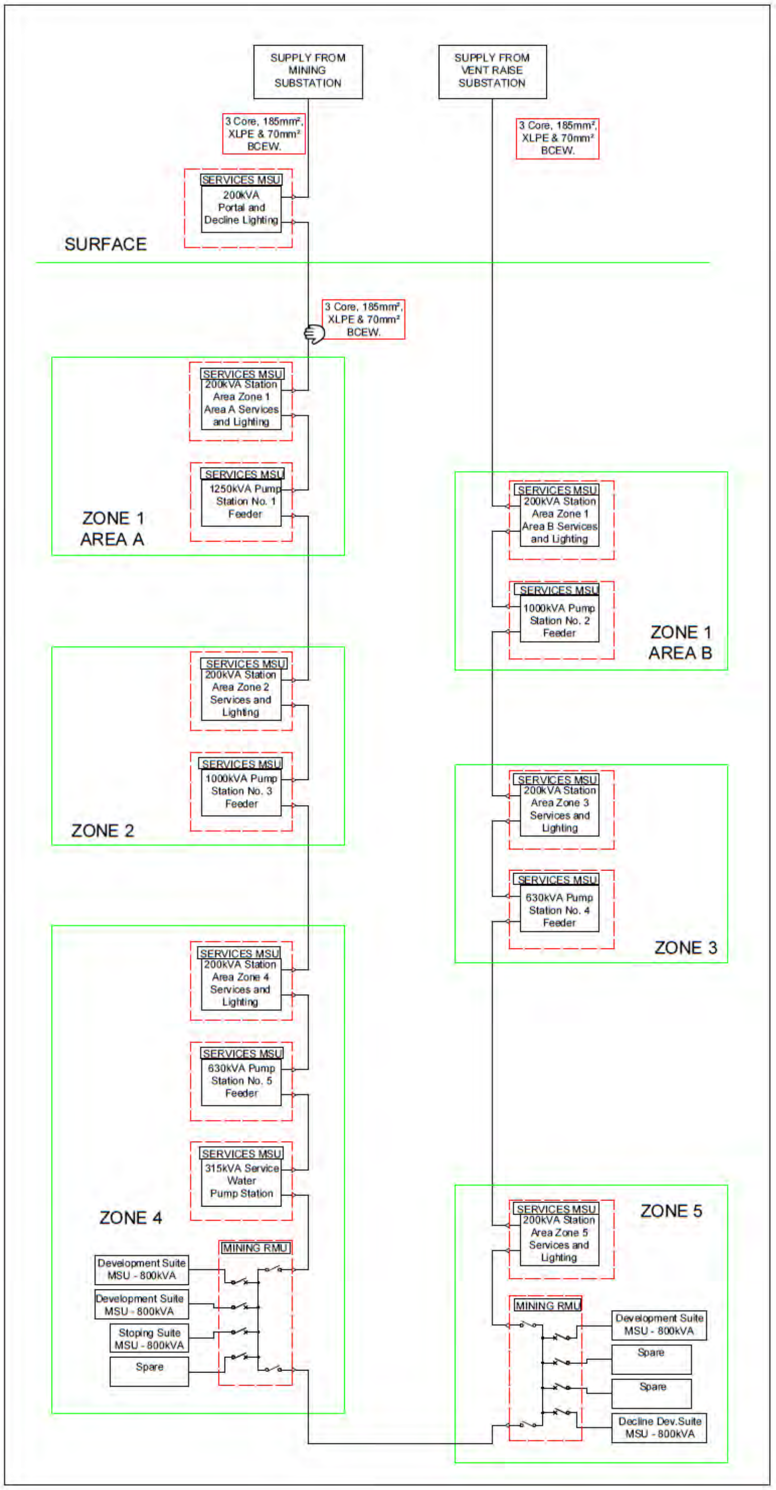

Figure 18-19: Dasa Underground Mine MV Reticulation.

Communication, Control, and instrumentation

A single mode fibre optic cable system will be installed providing the network backbone. All key areas will be connected to the fibre network providing a redundant communication system and data conduit for the mining operations. The fibre cabling system will provide connection of various systems, with dedicated fibre cores allocated for each of the communication, control system and IT sub elements, including the following:

- IP Telephone System.
- Refuge chamber hardwired telephone system.
- IT network.
- Access control, personnel, and asset tracking.
- Proximity detection system.
- Environmental monitoring systems.
- Centralised blasting system.
- PLC & SCADA control systems.

19. MARKET STUDIES AND CONTRACTS

19.1. Market Overview and Sales Strategy

Nuclear Power and Uranium Demand

Commercial demand for uranium is almost entirely defined by the fuel needs of the global nuclear power reactor fleet, however from time-to-time investor/fund demand (or secondary demand) can also be material. Currently there are 436 operable reactors supplying about 10% of the world's electricity, representing 24% of low carbon generation (Figure 19-1). Indeed, in advanced and emerging economies over the period 1971 to 2018, nuclear power has produced the greatest share of low carbon electricity (Figure 19-2). As the climate agenda gathers pace and with the new declaration made at COP28, nuclear power is well placed to increasingly support the decarbonisation of electricity grids around the world (Figure 19-2).

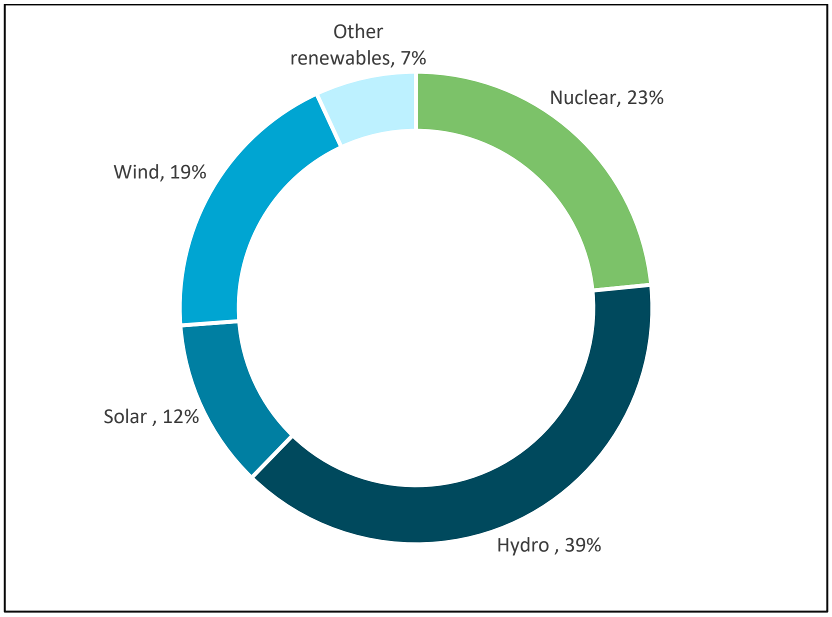

Figure 19-1: Low Carbon Electricity Contribution (2022).

Source: Ember Energy Institute.

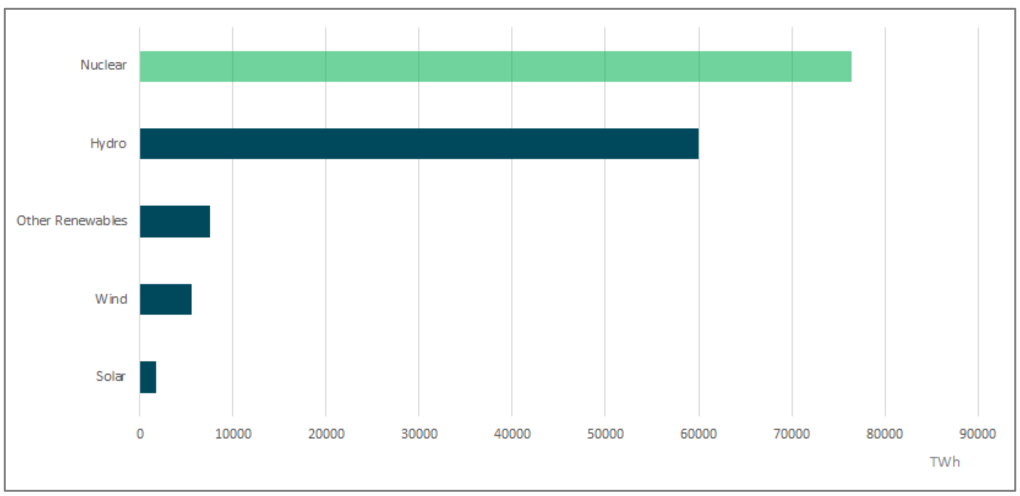

Figure 19-2: Contribution to Low Carbon Electricity in Advanced Economies (1971-2018).

Source: IEA.

There are currently 61 reactors under construction, including 26 units in China. At its peak in the late 1970's, following the oil shock, over 200 reactors were under construction, providing context for what might be possible under an analogous climate shock scenario.

The established fleet provides invaluable base load power supply to regional grids. A nuclear reactor is at its most efficient when running at full capacity and it therefore forms the cornerstone of grid supply, also termed baseload generation. As a result, uranium demand is resilient and predictable. The US average reactor fleet achieved a capacity utilisation of 93% in 2022, compared 35% for wind (Figure 19-3). On this basis 2.7 GW of wind generation capacity would need to be installed to match the output of a 1 GW nuclear reactor. For further context a 2.7 GW wind farm would require about 350 square miles of land compared to just one square mile for the same nuclear output.

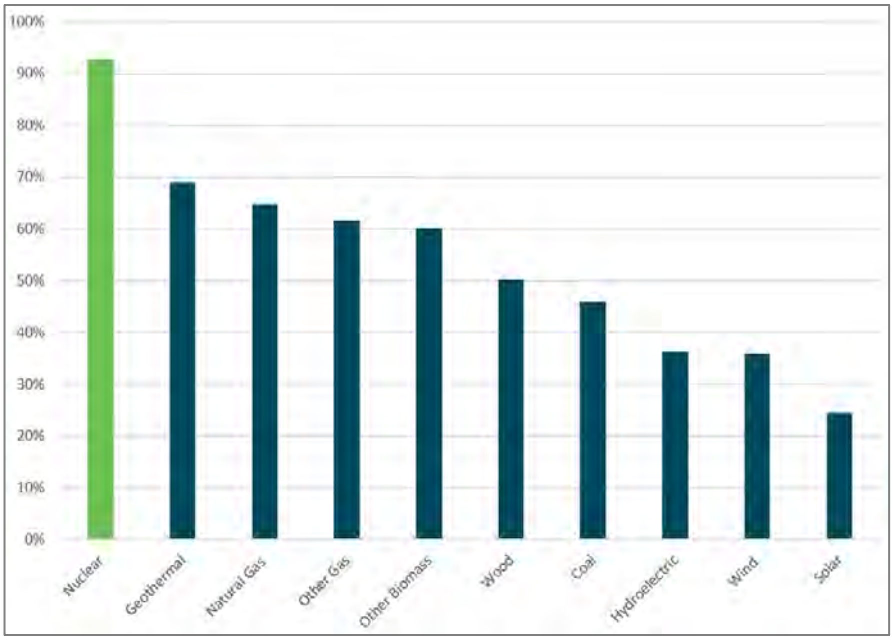

Figure 19-3: U.S. Capacity Factors by Source (2022).

Source: US EIA.

The high performance of nuclear power reactors and their long operating life (current licences run up to 80-years) means that the poor optics of high capex costs can in reality be depreciated over an extended period yielding a competitive Levelized Cost of Electricity (LCOE) (Figure 19-4). Nuclear LTO represents Long Term Operation, and the cost of extending the licenced lifespan of an existing reactor.

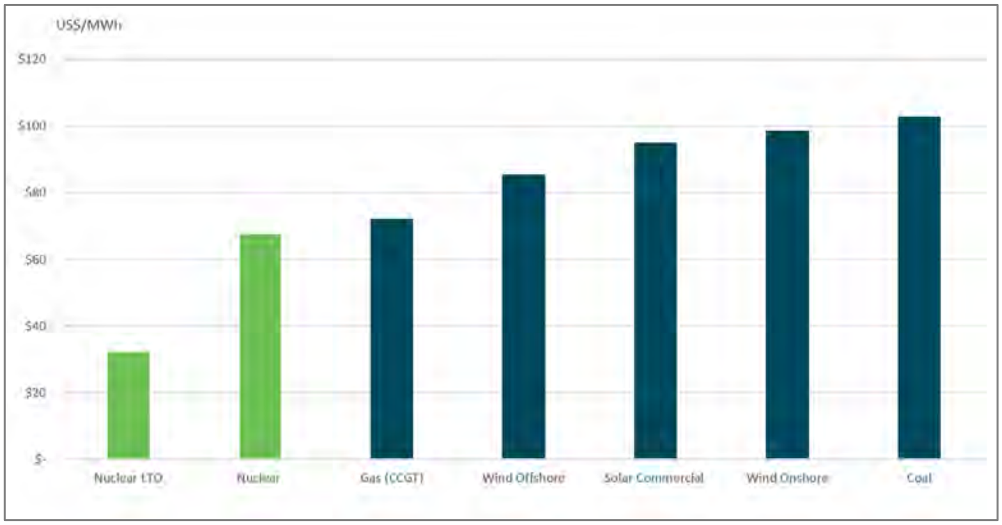

Figure 19-4: Levelized Cost of Electricity with a 7% Discount Rate.

Source: US EIA.

Financing is consequently a pivotal variable in defining the relative competitiveness of electricity generation. Importantly, the increasing interest rate environment has brought an end to the year-on-year decline in wind and solar costs. High finance rates have combined with escalating material costs – especially battery metals required to stabilise grid supply. The result is a notable reduction in competitiveness from the renewable sector.

Moreover, the advent of new technologies will provide upside to the nuclear sector. Small Modular Reactor (SMR) technology has gained considerable support and sponsorship from regional governments. These units are typically less than 300 MW in size and are quicker and cheaper to build. The flexibility of SMR's should not only improve the appetite for nuclear sourced grid supply, but also open new markets off-grid - be that for micro-grids in remote areas, industrial use, hydrogen production, district heating or shipping. The first SMR units are expected to be deployed in the early 2030's.

The 2023 COP28 Climate Summit in Dubai, UAE has triggered increasing commitments to accelerate new nuclear build as it has become clear that nuclear power will play a vital role in decarbonising global electricity grids. The US lead a pledge, along with more than 20 countries, to triple nuclear power capacity by 2050. It was backed by nations including the UK, France, Sweden, Finland, South Korea, and the UAE. It is hoped that this will pressure the World Bank to include nuclear energy within its lending policies. Separately, France declared that it would decide by the end of 2026 whether to build a further eight large (1600 MW) reactors in addition to the six that it plans to construct. President Macron had previously stated that France may build as many as 14 new reactors by 2050 as it pursues its carbon neutral target.

Further, the China Nuclear Energy Association (CNEA) has stated that the government expects to greenlight six to eight new nuclear units per year. Nuclear power is expected to contribute about 10 percent of power generation by 2035 and 18 percent by 2060 – targeting 400 GW. For context, this is larger than the entirety of the current global fleet.

The World Nuclear Association (WNA) in its reference case uranium demand forecast defines a 62% increase in reactor needs by 2035. Figure 19-5, below, factors in the reality that China has and continues to build strategic inventory beyond reactor needs, inflating demand over the near term.

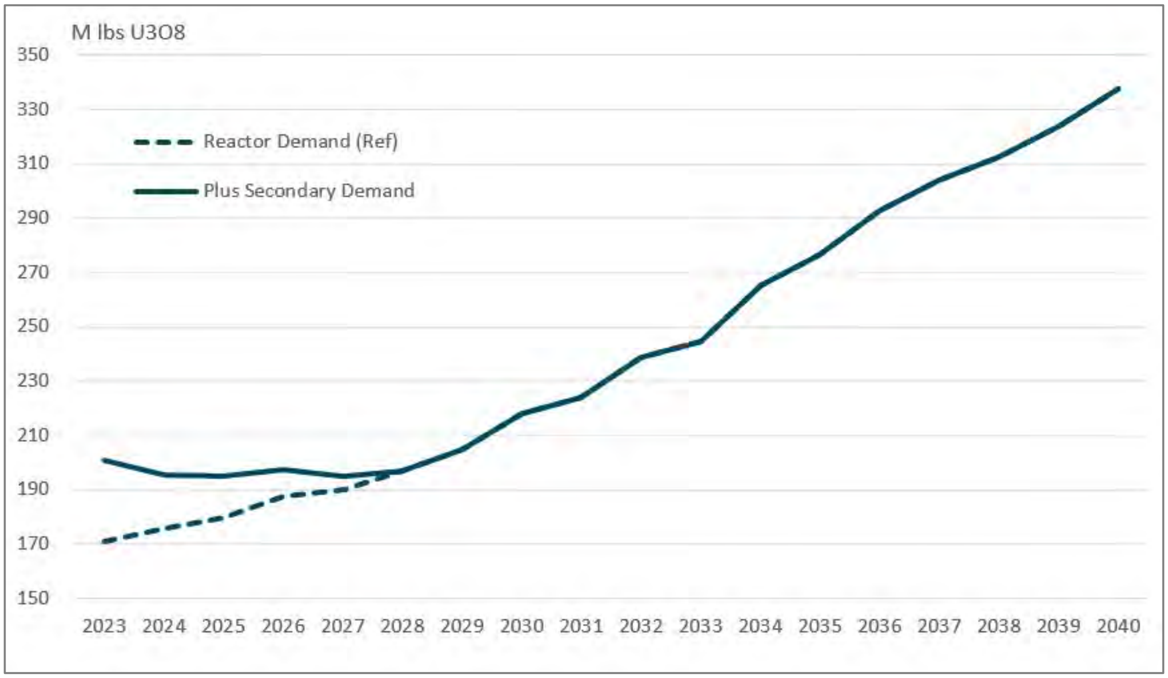

Figure 19-5: Uranium Demand.

Source: WNA Reference / Fuel Link.

19.2. Uranium Supply

Secondary Supply

Reactor demand is met through a combination of primary supply (mined production) and secondary supply (non-mined supply). Secondary supply is typically price insensitive and is deemed to find the market first. The key components of secondary supply are as follows:

MOX/RepU: Mixed Oxide or Reprocessed Uranium represents the ability of spent fuel to be reprocessed to extract unused uranium and/or blended with plutonium. Only 3 percent of spent fuel is technically waste, but its presence inhibits the performance of the fuel. Reprocessing uranium is expensive but is pursued by some utilities and regions for strategic reasons. It technically creates the 'closed fuel cycle' and helps to manage the spent fuel liability.

Re-enriched Tails and/or Underfeeding: These sources 'generate' uranium from the enrichment sector (Western and Russian). As part of the fuel cycle manufacturing process natural uranium (0.71% uranium) needs to be enriched to 4-5% uranium. Where excess enrichment capacity exists in the market (as has been the case in the past), more enrichment can be applied to less uranium to generate the required level of enriched uranium. This offset process has generated regular uranium supply to the market. However, following the Russian invasion of Ukraine, there has been a dramatic shift in enrichment demand from Russia to western sources. The availability of spare western enrichment capacity going forward will therefore be diminished. Indeed, the underfeeding situation could reverse. Should enrichment capacity become limited relative to uranium supply, then 'overfeeding' could manifest, whereby less enrichment is applied to more uranium (thereby increasing uranium demand).

Inventory: This tranche of supply is the opaquest component in the uranium market and represents

- The extent that utilities and supplier's drawdown their strategic inventory and
- Where historic excess supply has been carried on the balance sheets of producers, utilities, traders and financiers for future use or sale.
- Investor inventories, which have accelerated over the past 2-years, could return to the market at some stage in the future. The larger funds do not allow redemptions, and as such this supply is not assumed in current projections.

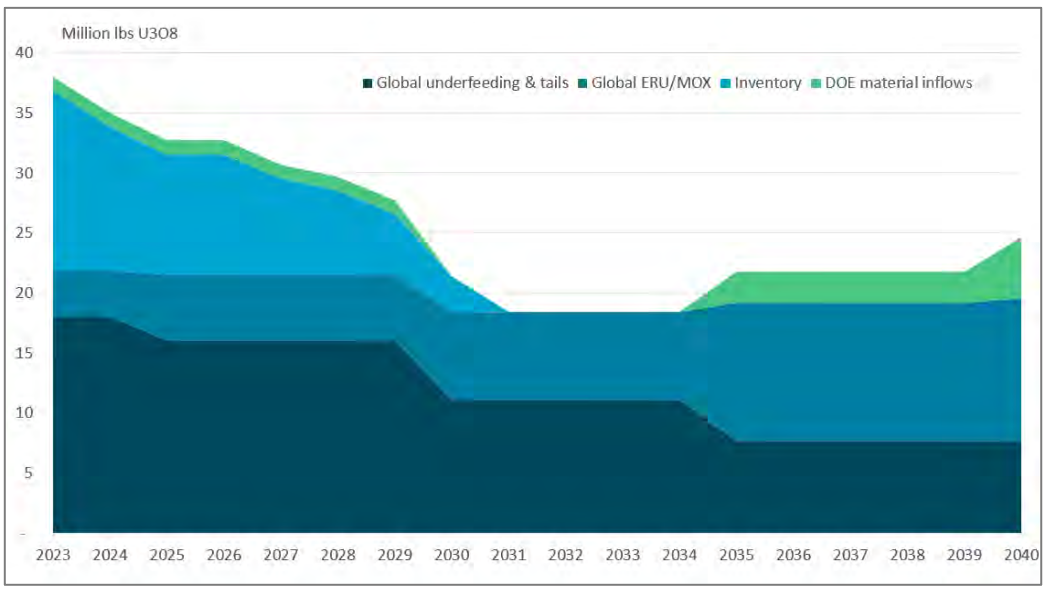

Figure 19-6: Secondary Supply.

Source: WNA Reference / Fuel Link.

Deducting secondary supply from uranium demand defines the call on primary production – Figure 19-7, below.

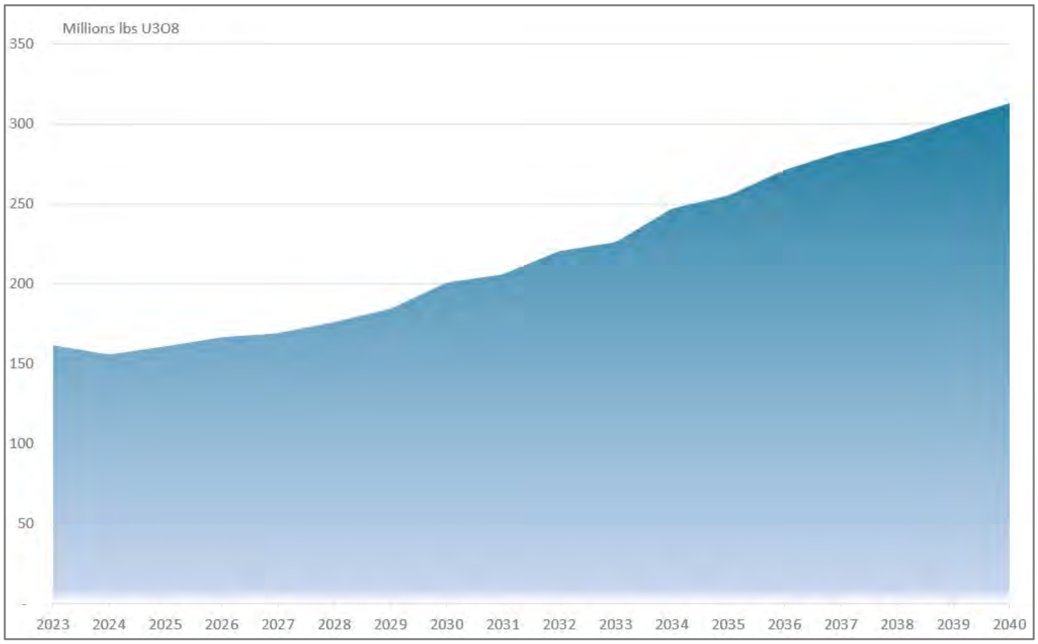

Figure 19-7: Call on Primary Production.

Source: WNA / Fuel Link.

Primary Production

Uranium is prevalent in the earth's crust, but as with all commodities the challenge is to extract it economically. The main established mining regions are Kazakhstan, Canada, Africa, and Australia although there are a number of other countries that produce the material. Conventional mining consists of open cast and underground operations, whereas over the past 15-years, In-situ Leach (ISL) production in Kazakhstan has increasingly dominated output. ISL production involves pumping an alkaline or acid solution into the ore body and then recovering uranium from the pregnant lixiviant.

Uranium production is generally price inelastic. Many countries that are new to uranium mining lean on the International Atomic Energy Agency (IAEA) for expertise and assistance in creating the necessary infrastructure and the legal, environmental, and regulatory framework to facilitate operations. The IAEA notes that it often takes 10 to 15-years to establish a uranium mine.

Once established, operations are typically underwritten with forward contracts. This mitigates forward business risk, but importantly it satisfies the desire from nuclear fuel buyers for forward supply security and predictable fuel costs. Through this forward hedging process, a proportion of supply becomes insulated from the prevailing spot price.

Consequently, structural oversupply can manifest and result in a protracted bear market, as characterised by the period up to 2022. This led to reduced investment in uranium assets. Over the past few years, legacy contracts have been expiring and uranium production has been falling (Figure 19-8), while at the same time uranium demand has been gradually firming. During 2022/23 as secondary demand from investors extracted significant spot supply, the market finally came into balance yielding a commensurate price reaction – however, the mining base will again be inelastic. It is evident that idled capacity exists. Once this is utilised new production will typically take many years to come online.

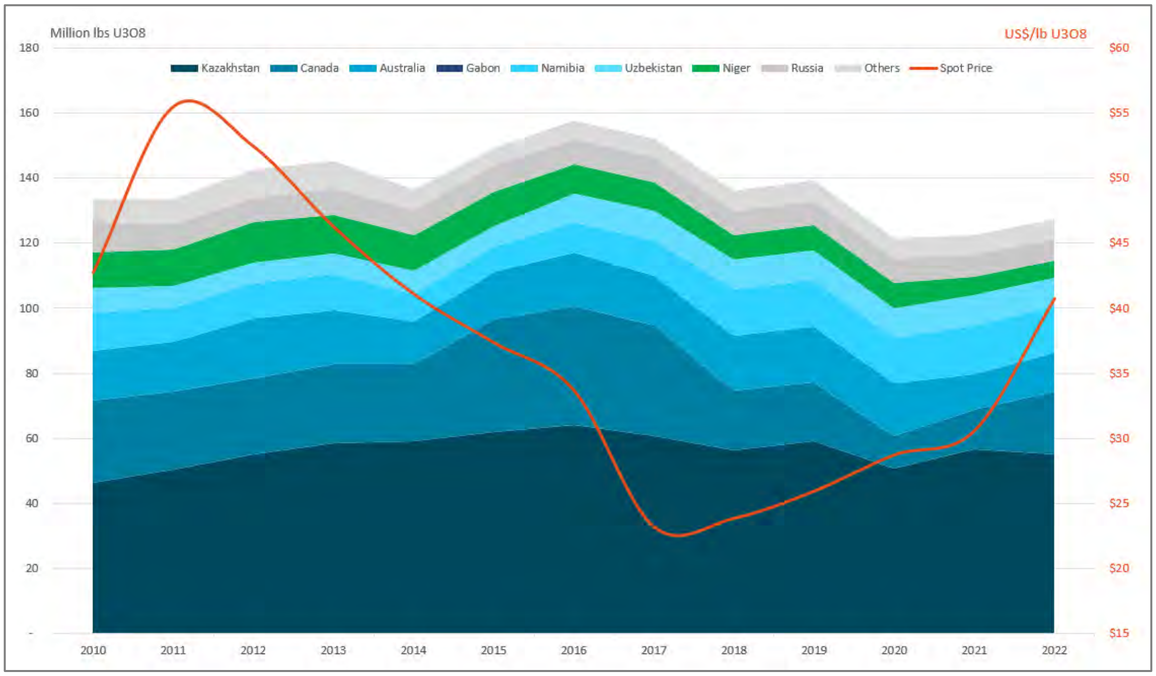

Figure 19-8: Historic Production and Uranium Price.

Source: WNA / TradeTech.

While the WNA identifies the potential for over 120 million pounds U_3O_8 of new production capacity (Figure 19-9), it is important to note that the vast majority of this output will require sustained high prices. Moreover, the 'Prospective' mines will need to pursue a typically lengthy permitting and licensing process.

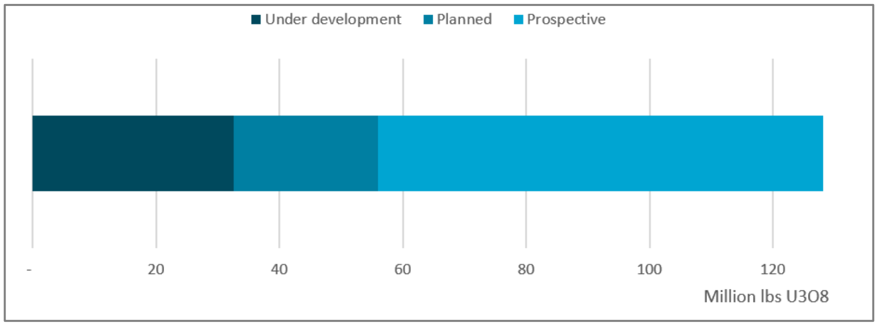

Figure 19-9: New Production Capacity 'Under Development', 'Planned' and 'Prospective'.

Source: WNA.

The overall supply and demand profile is presented in Figure 19-10, below. Based on WNA Reference case data, this illustration captures investor demand over the near term, which in large part is satisfied by inventories previously dedicated to forward demand. Inventory owners then replace this volume with forward purchases, decreasing the availability of future supply. Separately, as we move into the next decade, market fundamentals define a dramatic shortfall in supply.

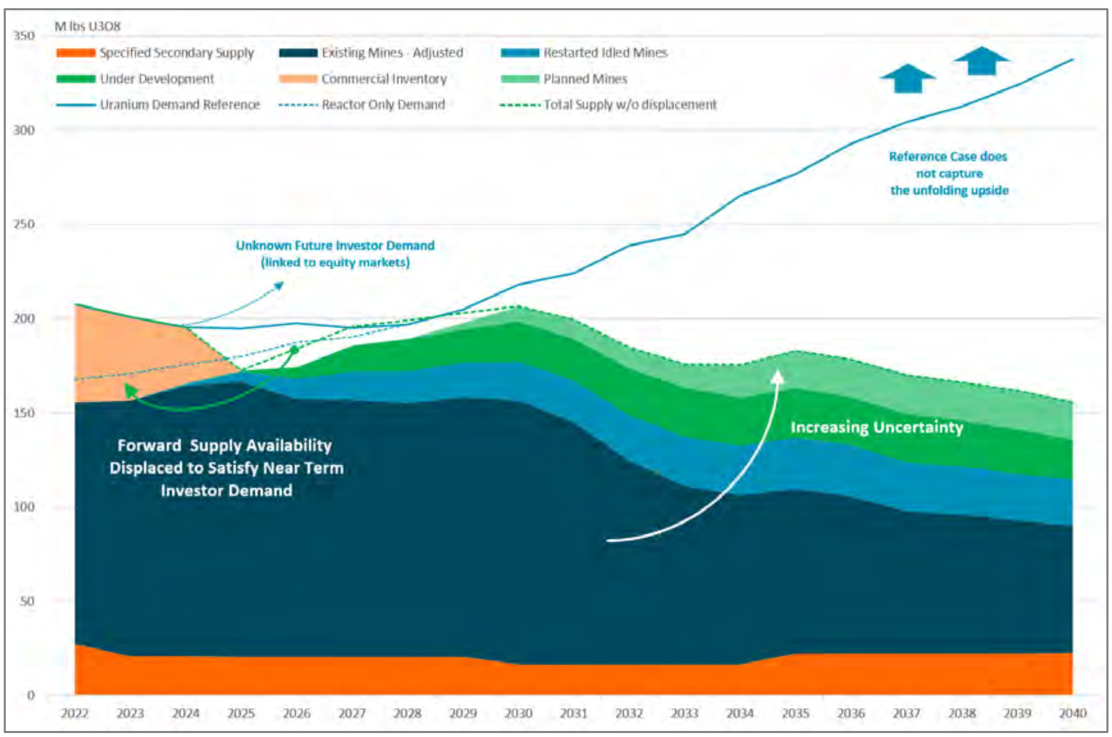

Figure 19-10: Uranium Supply and Demand Projection.

Source: WNA / Fuel Link.

There is a consensus view amongst industry analysts that uranium prices will need to demonstrate sustained improvement to:

- Prevent the further erosion of the established supply base and incentivise the introduction of new output.

Figure 19-11: Term Price Forecast

Source: TradeTech.

19.3. Uranium Sales and Contracting

The nuclear fuel marketing process requires involved dialogue: a two-way education process between the fuel buyer and the supplier to ensure that.

- The fuel buyer has sufficient confidence and trust in the supplier and
- The supplier is fully aware of specific needs of the utility.

Although price is important, there are a number of other attributes a fuel buyer will seek. Often there can be price parity within a tender process and non-price variables become the determining factor.

A lot of effort is invested into ensuring that potential partner companies share the same values. A buyer will want to know the background of a mining company, the stability of its operating environment, the quality of its resource, and its ability to produce effectively. Similarly, a supplier will want to know about the utility's uncovered requirements, its bid evaluation criteria, and if there were any delivery aspects that could be utilized to differentiate it from the competition. With a strong relationship, and the many involved discussions that will have been invested into that relationship, creative ideas are often forged that can satisfy requirements.

The uranium market involves a relatively small number of parties with approximately 40 potential end-user counterparties globally. Historically, utilities would rarely enter the market more than once a year for long-term business (contracts of 3 to 10-years in duration beginning 3 years forward). Indeed, it could be a number of years before a utility is “out” in the market.

Contracting can result from an open competitive tender or from strategic off-market negotiation. The contracting route will often be defined by utility preference.

Over the past 10-years, the market has evolved into a more liquid arena, and utilities will now enter the market more frequently, but for shorter-term contracts. These ‘mid-term’ contracts will typically begin 1 year forward and cover a 1 to 3-year period (although traders and banks are becoming more comfortable taking longer positions). The volumes involved are relatively small when compared to the traditional long-term contract and are often directed towards intermediaries. Utilities have been able to capitalise on the low cost of capital available to traders to carry surplus spot supply into the future. These transactions are termed carry trades, and they have become less viable over the past year, as finance costs have increased, and the uranium spot price has risen relative to the forward long-term price.

Direct contracting between utilities and producers dominates the transaction volumes and continues to be longer term and strategic in nature. Consequently, relationships remain of high importance.

Uncovered Demand

Consistent with the forward contracting behaviour of the market, openings for uranium increase over time. The US Energy Information Agency (EIA) and Euratom Supply Agency (ESA) publish the contract coverage numbers for the USA and EU, respectively. Figure 19-12 below displays the pace of US contracting over the 2020-2022 period. Activity increased in 2022 reflecting increased geopolitical risk, and tightening supply availability. In Europe contracting has been lighter reflecting the greater level of established coverage. The negative coverage in the earlier years (i.e. buying beyond reactor needs) indicates a desire to build inventory over that period.

In addition, there will be notable contract openings in Asia and the rest of the world. China is a significant buyer requiring approximately 30 million lbs U_3O_8 per year, rising to 60 million lbs U_3O_8 by 2030. In 2009, China began importing volumes well ahead of its needs in anticipation of its nuclear build campaign. Despite accumulating a significant strategic inventory in the interim, its appetite for continued deliveries remains undiminished. Other key buyers include Korea Hydro Nuclear Power (KHNP) with total annual needs of approximately 10 million pounds U_3O_8, plus the UAE and India with growing reactor fleets. Moreover, unaccounted for here is upside demand for initial cores, and the ongoing need from new builds, as decarbonisation plans are established.

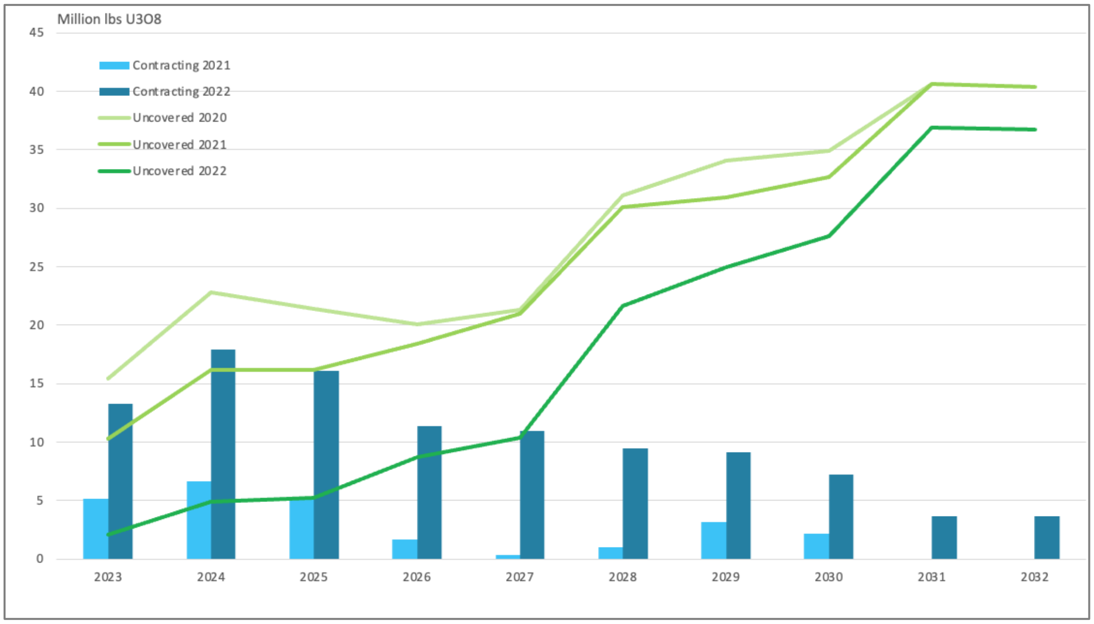

Figure 19-12: US Annual Contracting and Uncovered Demand.

Source: US EIA.

Figure 19-13: EU Annual Contracting and Uncovered Demand.

Source: ESA.

19.4. Supply Competition

Within the uranium supply base there are a limited number of uranium supply companies and even fewer producing jurisdictions. Over the last 5-years geopolitical issues have intensified making geographical supply diversity a key consideration for fuel buyers. Production has increasingly become dominated by Kazakhstan. Africa has always played a key role in balancing the market. However, in recent years African assets – which are often low grade and high cost - have been purchased by Chinese utilities. Consequently, this supply is not generally available to the open market. Figure 19-13 demonstrates the fundamental supply available to the open market from the existing supply base in 2025, and the reality – reflecting marketable material. The introduction of supply from the Dasa project would increase the availability of supply from Africa by 50%.

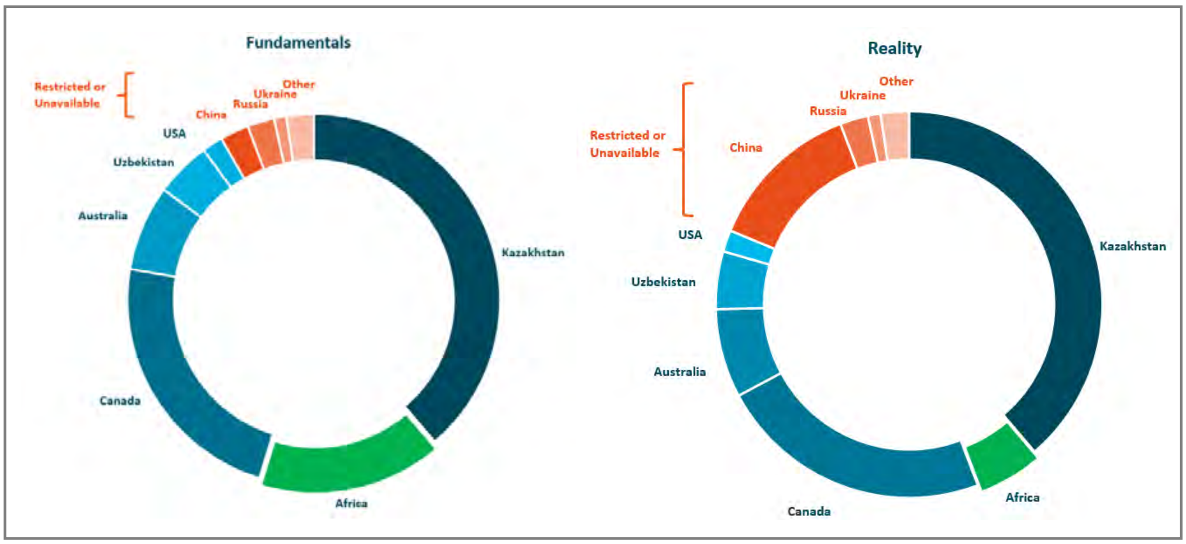

Figure 19-14: 2025 Existing Production & Geographical Diversity.

Source: Fuel Link / TradeTech.

Moreover, there are only a handful of major producers that wield notable market power in the market. Utilities are keen therefore to support supply competition from economic supply sources.

In 2020, low market prices combined with COVID disruptions to reduce 2020 primary production. Since then, the market has firmed, and production plans indicate a notable increase in capacity utilisation through 2025 (Figure 19-14). Producer company utilisation is set to increase from 60% in 2020 to almost 90% in 2025. Beyond this, the Dasa Operation is currently the only large green-field conventional uranium mine under construction.

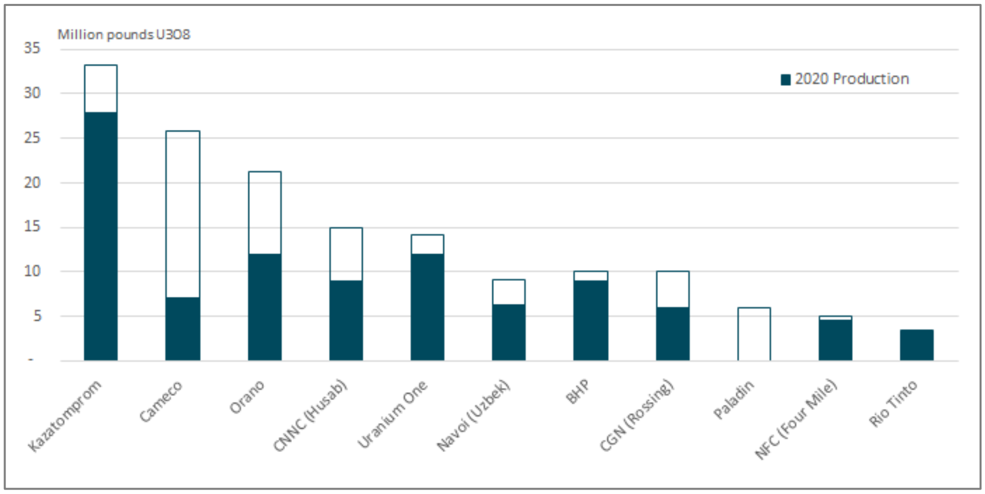

Figure 19-15: Production Capacity Utilisation 2020.

Source: Fuel Link.

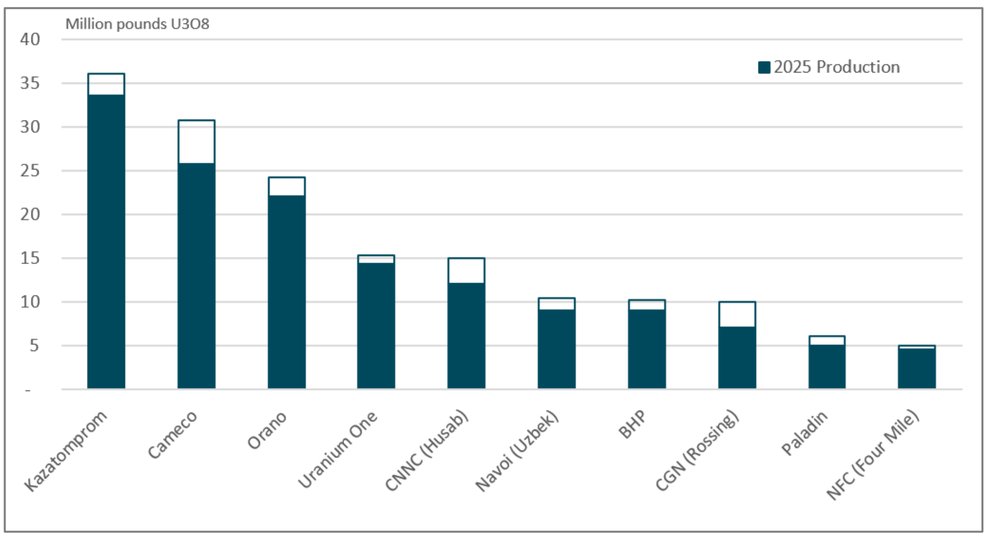

Figure 19-16: Projected Production Capacity Utilisation 2025.

Source: Fuel Link.

19.5. Pricing Mechanisms

Fuel buyers and sellers have a number of pricing mechanisms and approaches available to them that capture varying degrees of market risk. They generally fall into two main categories: Market Related or Specified Pricing. There exists a general desire from both sides of the transaction (miner and utility) for price predictability. This is particularly the case for utilities, allowing them to control their generating costs and onward powers sales. Indeed, the forward nature of sales (generally committing supply up to 10-years forward) affords utilities crucial supply security. Miners will also desire sufficient predictability to underpin positive cashflow. Higher cost operations typically are more risk averse and are likely to lock in prices for longer. Both sides, however, will also value market exposure. Utilities will not wish to be locked into ‘above market’ prices and like to benefit from lulls in the market. Similarly, miners will be keen to demonstrate to stakeholders that they can benefit from market upside. Defining a portfolio of contracts that deliver the correct balance of risk and reward for both parties, requires careful evaluation and execution.

Specified Pricing

This provides the greatest level of income certainty. Contracts can be written to define hard ‘fixed’ prices for each U_3O_8 delivery; or ‘base escalated’, where a delivery price is defined in today’s terms and then escalated by either i) an inflation indicator or ii) a defined escalation rate. Fundamentally, the escalation rate is employed to ensure the real terms value of the contract is maintained. Typical escalators include the US Consumer Price Index or US Gross Domestic Product price inflators (in US dollar priced contracts), and the date at which these escalators would begin is also defined in the contract. The base price agreed by the parties is the price that is reflected in the reported TradeTech LLC and Ux Consulting Company LLC Long Term-Price Indicators. These long-term price indicators typically trade at a premium to the spot price and over the period 2019-2021, they were typically 20 percent above the spot price. However, the term price is significantly less liquid than the spot price, and during periods of market firming the premium reduces - as the term price lags the spot price. During periods of market softening the reverse occurs and the premium generally increases.

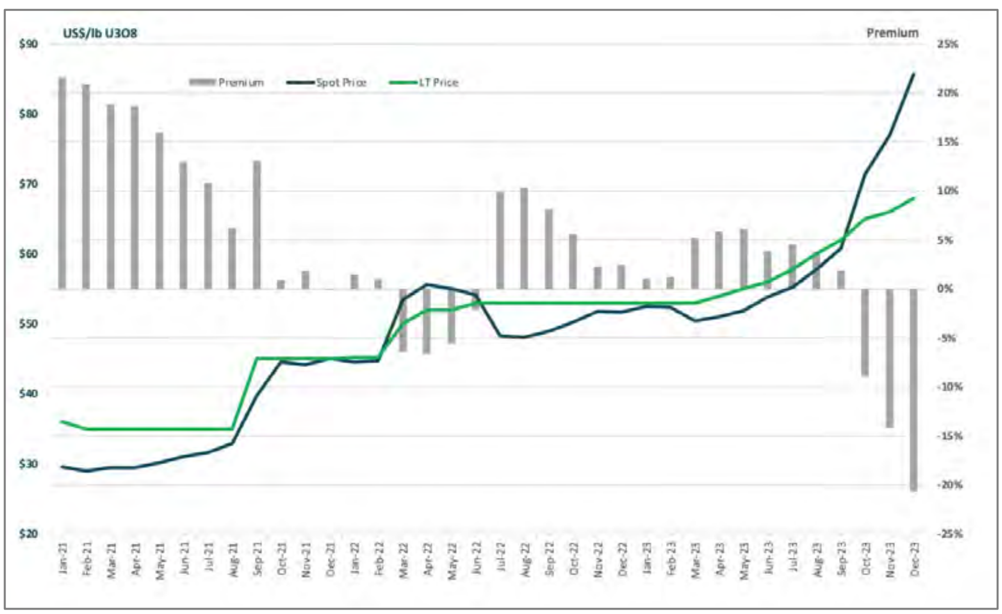

Figure 19-16: Spot Price vs. Long Term Price.

Source: TradeTech.

Market-Related Pricing

Also termed ‘floating’, market related pricing typically captures the uranium spot price at the time of delivery. Other pricing indicators can also be employed, depending on the appetite of the parties involved. Historically, long term price indicators as well as government published import/export numbers have been utilised, including a combination of those indicators. However, there are few parties that are comfortable with entirely floating delivery prices. Floor and ceiling prices are often also required to balance market exposure. A floor price will protect the supplier’s cost base, while a ceiling price will protect the utility’s fuel cost. Should the market index employed travel beyond the floor and ceiling levels (also termed a collar), then the floor and ceiling price would apply. Collar prices, like specified prices, can be fixed or escalate. Indices can include the TradeTech Exchange Value, Ux Spot Price, TradeTech Long-Term U_3O_8 Price Indicator, Ux Long-Term Price, US DOE Spot and/or Long-Term Price, Euratom Spot and/or Long-Term Price. The indicators utilised are mostly dictated by the nature of the market at the time of contracting.

Other Pricing Mechanisms

Although dominant throughout history on a worldwide basis, specified and market-related price mechanisms are not the only types that have been utilized. There exist three additional broad categories of price mechanisms, including “negotiated,” “hybrid,” and “cost-related” pricing.

Market Dynamics: Secondary Demand

The primary demand numbers discussed above relate entirely to utility needs. However, investor demand has become a dominant feature of the market. Relative to utility demand, investor volumes remain low but, importantly, this demand impacts the illiquid and shallow spot market; as opposed to utility demand, which falls into the deeper forward market.

A number of traders, banks, hedge funds, investors and uranium developers have taken positions in the physical market, and as the availability of spot supply thins, their influence increases Listed funds have been established to enable retail and institutional investors to gain exposure to uranium prices by proxy. Two funds were established: Yellow Cake PLC, listed on the London Stock Exchange; and the Uranium Participation Corporation (UPC), listed on the Toronto Stock Exchange.

In July 2021 Sprott Asset Management - with at the time approximately $15 bn under management in Gold, Silver, Platinum, and Palladium - took over management of UPC, and immediately raised the profile of the fund (renamed Sprott Physical Uranium Trust [SPUT]). Importantly, they introduced an at-the-market raising process allowing SPUT to avoid lumpy equity raisings, and instead incrementally gain funds at times when the fund trades at a premium to its NAV. They consequently became a more consistent buyer in the spot market.

Sprott’s timing was excellent as it coincided with post-pandemic fiscal stimulus, targeting decarbonisation strategies, and improving global projections for nuclear power. It was later able to capture positive market sentiment after Russia’s invasion of Ukraine. Since the war, western utilities have been scrambling to move away from Russian origin fuel in anticipation of sanctions. This mainly impacts the markets of conversion and enrichment, where Russia is much more dominant, but will have a knock-on influence on the uranium market.

Generally, the lack of liquidity in the uranium market means that it is not capable of compounding information and ‘rational expectations’ into the price i.e., it is not efficient. Equity markets, however, are much more efficient. A disconnect was therefore created between uranium related equities and the physical market, as equities were buoyed by the positive sentiment, while uranium prices remained largely unaffected (Figure 19-17) below.

Figure 19-17: Uranium Spot Price and Fund Inferred Price 2020-2021.

Source: TradeTech / UPC.

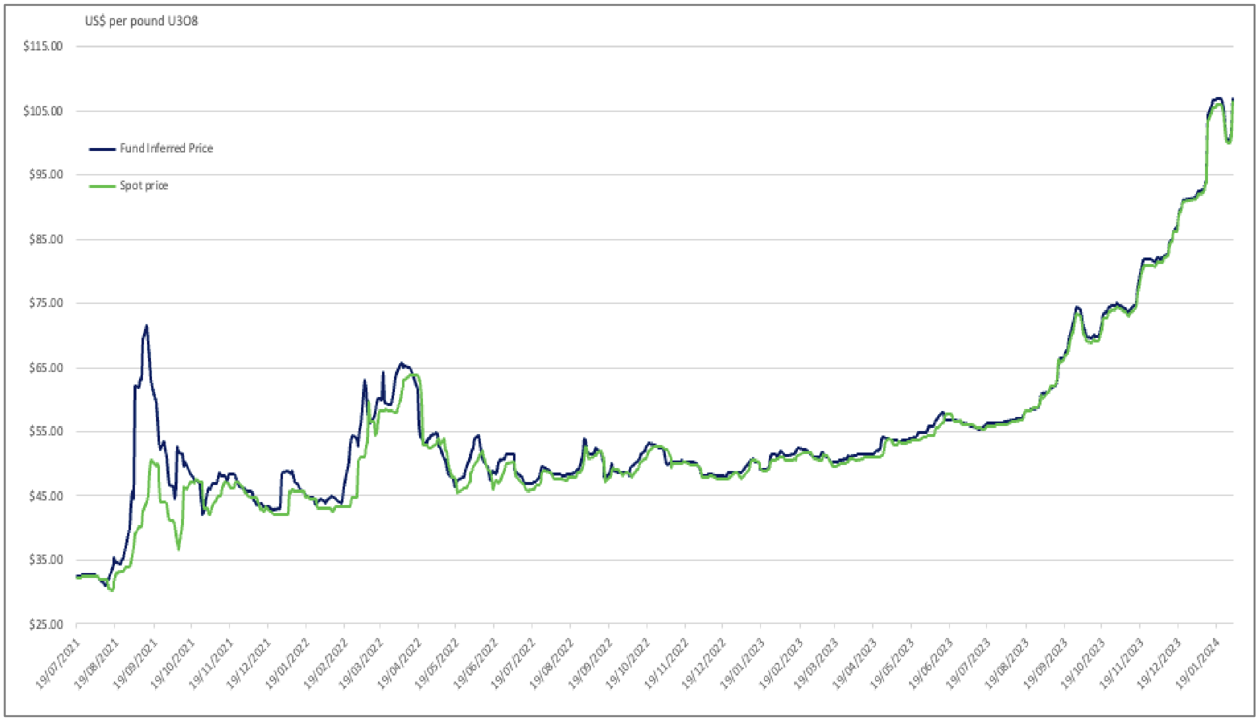

Figure 19-18: Uranium Spot Price and Fund Inferred Price 2021-2024.

Source: TradeTech / Sprott.

SPUT, however, became a vector for investment interest, channelling capital into physical uranium; thereby creating self-fulfilling investment returns. The more funds SPUT raised, the more uranium it purchased, the higher the uranium price increased, the more investment interest was created. The uranium price consequently increased from a $30/lb level in July 2021 to a $65/lb level by April 2022. As the equity markets softened, so did SPUTs ability to raise funds, and the spot price subsequently subsided to the $50/lb level – still a lot healthier than without SPUTs involvement.

Through most of 2023, SPUT has traded at a notable discount to NAV. However, in its absence a number of hedge funds have been active. Although buying on a much smaller, the thin market has ensured their demand has made a notable impact. As of January 2024, the uranium spot price breached $100/lb for the first time in 16-years. The uranium long term price – reflecting more substantial volumes booked between producers and utilities has increased from $35/lb in July 2021 to the $70/lb level as of January 2024.

The revised market level is now much healthier for the industry, reflecting pricing that will be more supportive of a diversified and stable production base.

Looking forward, it is not possible to forecast investor interest, but given the reduced level of liquidity, market players will be cognisant that only limited, well timed purchases will be required to generate tangible price increases – inflating their underlying fund values.

This combines with structural undersupply projections as energy policies embrace nuclear power - for decarbonisation and energy independence purposes. The consensus view is that market prices are set to remain both firm and volatile.

19.6. Marketing Strategy

In March 2021 Global Atomic engaged, Fuel Link Limited to provide marketing services for the company. Fuel Link is a UK based nuclear fuel trader, brokerage, and consultancy with over 20-years of market experience. A marketing strategy and sales plan was developed and approved by the board. The strategy has remained dynamic and dovetails with the financing requirement of the project. Fuel Link initiated a market introduction program to educate the fuel buying community on the attributes of Global Atomic. The Dasa operation is now well known and understood within the industry. The continued marketing objectives are to convey the compelling characteristics of Global Atomics' uranium supply capability, ensuring the company is included on utility Request for Proposal (RFP) bidder lists, and to progress off-market opportunities where possible. Term uranium sales typically have a long gestation period, as confidence in counterparty credibility is established, and common ground is found on commercial terms. Proposal iterations can take months and years to finally be agreed, but the ultimate deal value is typically high and often in the hundreds of millions of dollars.

Crucially, Dasa is unique within Africa exhibiting the following characteristics:

- Cashflow from an established business.
- Management track record of bringing new development projects into operation on schedule and on budget.
- Fully delineated Phase 1 high-grade, low-cost orebody with a sub $20/lb U_3O_8 AISC.
- Fully licenced, permitted and under construction.

These characteristics are demonstrated in Figure 19-19.

The marketing strategy aims to balance business security and finance support with market exposure. While the market fundamentals point to firm prices, the level of inventory overhang is opaque. Ultimately, the future is unknown, and it makes commercial sense to balance exposure to future firm prices with some downward price protection. This supplements the inherent needs of power generators, seeking fuel cost predictability.

The marketing plan therefore sets out a process of layered contracting, allowing the company to enter production as scheduled and benefit from a firming market over time. Furthermore, contracts will generally employ hybrid pricing, combining market related pricing with specified (fixed) prices. As mentioned, market related pricing will typically include a floor and ceiling, to protect both parties. A portfolio approach to sales will balance geographical markets, pricing mechanisms and contract length.

It is important to note that there is a distinct commercial/marketing advantage to being in operation. Existing suppliers will be deemed lower risk with utilities unlikely to provide large volume contracts to developers.

Global Atomics' ability to establish 'seed' contracts in the current market environment, which are then leverageable in the future, creates a competitive differentiator in the market.

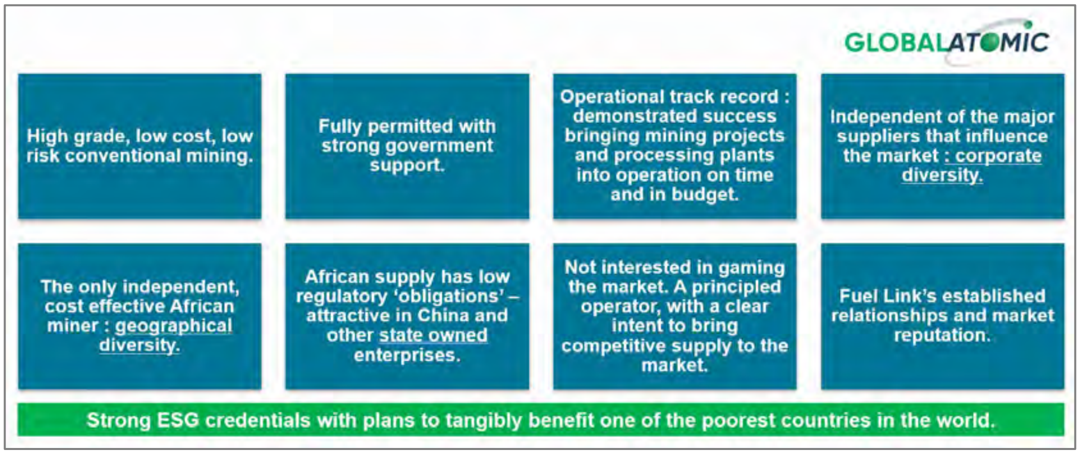

Figure 19-19: Global Atomic Uranium Supply Characteristics.

The marketing strategy will remain flexible, be reviewed regularly, and adapt to the changing market environment.

Booked Sales

The sales strategy described above has delivered significant results for the company. In the face of notable competition from existing and developing producers, utilities have chosen to support Global Atomic and the Dasa Operation. This reflects the confidence fuel buyers have in the management team, the quality of the Dasa deposit, as well as a desire for competitive supply diversity.

Sales so far have been North America focussed with three strategic contracts finalized plus one letter of intent with a European utility.

Delivery volumes total up to 9.5 million pounds U3O8 over the period 2026-2032. Per the strategy, pricing utilises a mix of defined prices and floating prices yielding a combination of income protection and market exposure. These sales have been booked over the past year as the term price indicator increased from the $50 level to the $80 level. Due the term nature of contracting there will always be an element of legacy pricing imbedded within the sales portfolio. However, this will be balanced by the floating priced component as well as the ability to sell further pounds into a firming market. While the future market environment appears particularly buoyant, the company is mindful of a need to hedge against uncertainties, as well as a need to underwrite debt. Current booked business, assuming a $95 market, has revenue potential of US$ 772 million or an average of $81 per pound U3O8.

Military Coup Impact

The military coup of July 2023 was a setback to the marketing effort. An increased risk perception to Niger supply was immediately felt, hindering new business. Further, the delay to mine development made delays to initial deliveries under the first two signed agreements likely. Regarding the signed agreements, start up flexibility terms allow the delivery schedules to be adjusted - and the entire delivery schedule under those contracts will now be delayed by 12 months.

From a new business perspective, the uranium market – with its strong institutional knowledge of mining disruption - was quick to digest the political developments and recognise the continued long term supply potential of the country.

An agreement by the new military leadership to democratic elections, leading to a removal of sanctions and the opening of the Benin border will go a long way to supporting continued supply credibility. Nevertheless, in a strong vote of confidence in the country, the company, and the asset, a third uranium term transaction was announced in October 2023.

In November 2023, it was announced that the project finance banks were re-engaged, that the new Niger government was highly supportive of the project, and that new supply chains have been implemented allowing development mining to restart.

Consequently, procurement interest is healthy, and further sales are being pursued.

Safeguards, Physical Delivery and Book Transfer

All nuclear fuel transactions are governed by the IAEA. Sales can only be made to known parties within the nuclear power industry, who in addition are contracted to comply with the nuclear fuel 'peaceful purposes' limitation. The IAEA supervises the 'fuel cycle' – the term used for the fuel manufacturing process (Figure 19-20). Nuclear fuel component deliveries (including uranium) can only be made within the fuel cycle.

Fuel is delivered to a facility that is engaged in the next stage of the fuel cycle, for natural U_3O_8 this is a converter. There are three western converters: Orano (France), ConverDyn (USA) and Cameco (Canada). Conversion facilities are also established in Russia, China, and India. Global Atomic sales will be focussed on deliveries at the western converters. The company will therefore establish storage accounts at those facilities. The process of establishing accounts has been initiated and requires U_3O_8 samples from the Dasa project to be tested for specification compliance.

In practice Global Atomic will store working inventory at the three converters. Utility customers, per their contract with Global Atomic, will provide notice of which converter they require their U_3O_8 delivery, and accordingly material will be 'book transferred' from the Global Atomic storage account to the utility account. Title of the uranium passes with the book transfer, triggering payment (typically within 30 days).

For sales to China, Russia and India, the process can differ, and title may transfer to the utility (or relevant fuel company) at the port or border. Transactions of this nature are termed ‘physical delivery’. These parties may also take delivery at the western converters by book transfer.

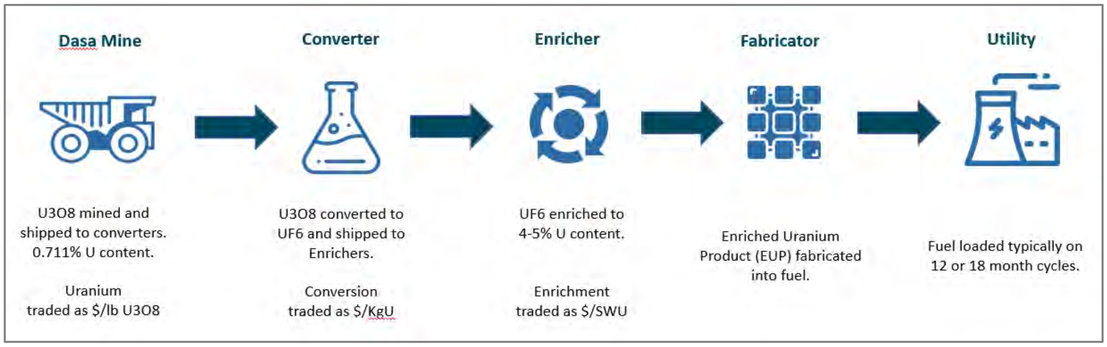

Figure 19-20: Front End Nuclear Fuel Cycle.

Source: Fuel Link.

20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1. Environmental and Social Studies and Approvals

In 2011, GAC commissioned Niger engineering and environmental consultancy Groupe Art & Genie to complete an Environmental Characterization Study (“ECS”) and establish environmental and socio-economic baseline information for the Dasa Project. GAC also retained Groupe Art & Genie to conduct hydrology and hydrogeology studies during the period 2012 - 2016.

In 2020, GAC retained Groupe Art & Genie to conduct an Environmental and Social Impact Assessment (“ESIA”) for the Dasa Project Phase I Mine Plan. The ESIA updated previous environmental and socio-economic baseline information in support of the Company’s Mining Permit application for the Dasa Project. GAC engaged with the Ministry of Environment and Sustainable Development (“MESUDD”) to confirm the Terms of Reference and scope of both the ESIA and the Environmental and Social Management Plan (“ESMP”) at the start of the ESIA process.

The 2020 ESIA served to update the ECS completed by Groupe Art & Genie in 2011 and included hydrological and air quality baseline data compiled during the interim period, as well as addressing environmental, social, and economic impacts associated with construction of the mine, mining operations and reclamation and closure.

The ESIA included an analysis of alternative design and operating scenarios to reduce environmental and social impacts associated with the Mine. A key element of the ESIA was public consultation to inform stakeholders of the proposed Project, and technical studies related thereto. Community meetings were held in area villages and small towns as part of the ESIA process and were organised by MESUDD and attended by Government and GAC personnel. GAC has maintained regular engagement with surrounding communities since the start of exploration activities in 2008 and provides on-going support to these communities in the areas of water supply, food security, healthcare, education and training, local business support and procurement (see Section 2.4 for further detail).

A key component of the ESIA is the ESMP. The ESMP governs the Company's activities from construction through operations and mine closure and establishes the protocols for Project monitoring and reporting to government ministries and agencies. The ESMP also outlines the management procedures, mitigation, monitoring and reporting protocols that will be used to avoid, reduce, and manage environmental and social impacts during the construction, operations, and closure phases of the Dasa Project.

The ESIA was submitted to the MESUDD for review in October 2020 in accordance with Law No. 98-56 of 29 December 1998 on the framework law on environmental management and Law No. 2018-28 of 14 May 2018 setting out the fundamental principles of environmental review. The MESUDD approved the ESIA and ESMP in November 2020, and officially advised the Ministry of Environment and Ministry of Mines that GAC had successfully completed the ESIA process and was eligible to receive a Mining Permit for the Dasa Project. The Mining Permit was issued by Presidential Decree effective December 23, 2020. The Mining Permit was issued for an initial period of ten years, subject to automatic five - year renewals until the deposit is depleted.

GAC also entered into two agreements with the MESUDD in December 2020: the "Partnership Agreement", which establishes the framework for the implementation and monitoring of the ESMP, and capacity building and budgets for the relevant ministries and agencies and, the Cahier des Charges ("CCES") (Environment and Social Charges Book), which establishes implementation, monitoring and reporting protocols related to the ESMP. The BNEA issued its final authorization, the Certificate of Environmental Conformity, on January 23, 2021. The Dasa Project has received all Permits required to commence construction and operations.

GAC is committed to undertake its operations in line with the Equator Principles ("EP4"), an international financial industry benchmark for determining, assessing, and managing environmental and social risks. Pursuant to the above, in 2022 GAC commissioned a new ESIA by Firme d'Expertise en Environnement et Développement ("FEED Consult"), a Nigerien environmental consultancy and subject-area specialist.

In 2023, GAC compiled an ESIA Addendum report ("Addendum"), which was designed to incorporate information from the 2023 revision of the Feasibility Study, as well as to summarize both the government - approved 2020 ESIA and the FEED Consult ESIA.

The Addendum summarizes environmental and social management measures put in place to ensure the Project is undertaken in accordance with both government requirements and good international industry practice ("GIIP") to include the International Finance Corporation Performance Standards on Environmental and Social Sustainability ("IFC PS"), IFC Environmental, Health and Safety ("EHS") Guidelines, and the guidance of the International Atomic Energy Agency ("IAEA"), of which Niger is a member state.

20.2. Existing Environmental and Social Setting

Location, Climate and Terrain

The Dasa Project is located within a sparsely inhabited region characterized by the presence of small villages. Three larger, permanent villages (Agatara, Teguef Nakh, and Tagaza) are located along the highway which runs approximately 5 kms west of the Project site. The highway links the regional town of Arlit, 105 kms to the north, with Tchirozérine and Agadez, 60 kms and 95 kms, respectively, to the south.

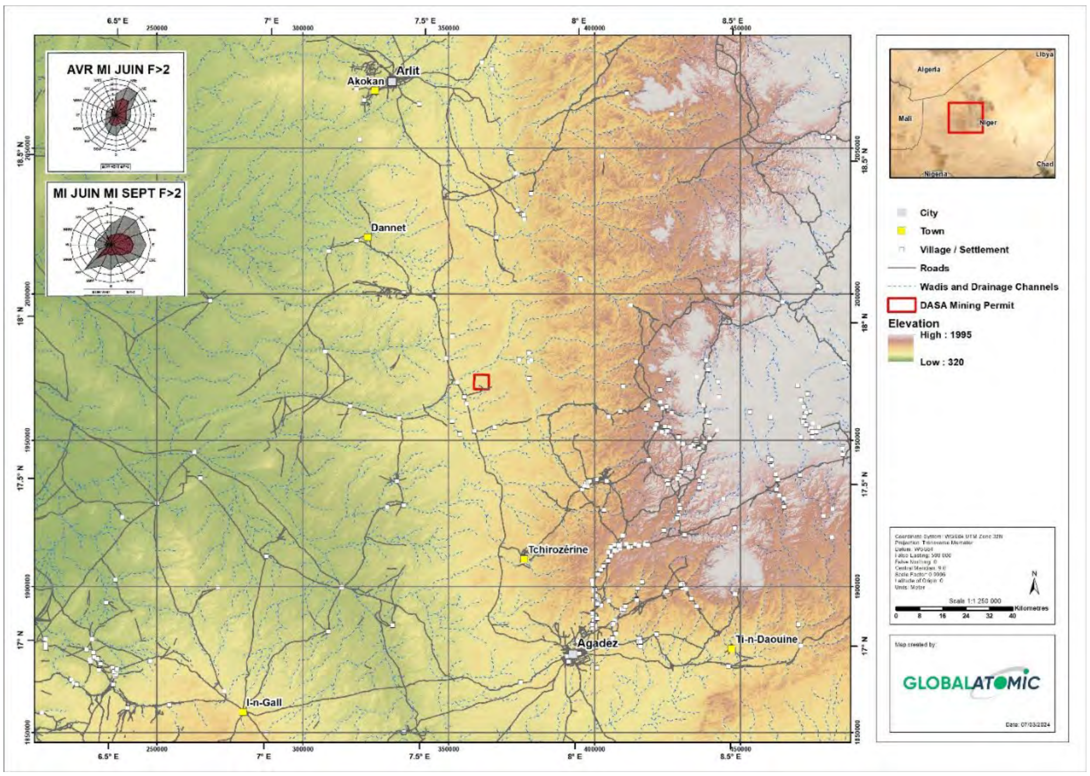

Figure 20-1: Regional location of the Dasa Mining Permits.

The Project is located within the Sahel-Saharan desert climate zone, which is characterized by a six-month warm season (April to September) and a six-month cold season (October to March). Within the warm season there is a short rainy season lasting from June to September. In the warm season, the temperature varies between 31 °C and 50 °C; in the cold season it varies between 0 °C and 20 °C. Analysis of 20-years (2000 – 2019) of rainfall data from the Tchirozérine weather station indicated annual rainfall varying between 77.5 mm and 332.5 mm, with an annual average of 180.2 mm. During the dry season, the prevailing winds are from the north-east and north-northeast - these are the Harmattan winds. During the rainy season, there is a more significant component of winds from the south-west.

The Dasa Project is located on the eastern edge of the Tim Mersoï Basin. The terrain is a generally flat, sandy plain at about 500 m above sea level, with elevations decreasing gently to the west. The Aïr Mountains, located some 30 km to the east, reach over 1,800 m above sea level. The sandy plain has occasional rock outcrops and is traversed in places by koris (seasonal watercourses).

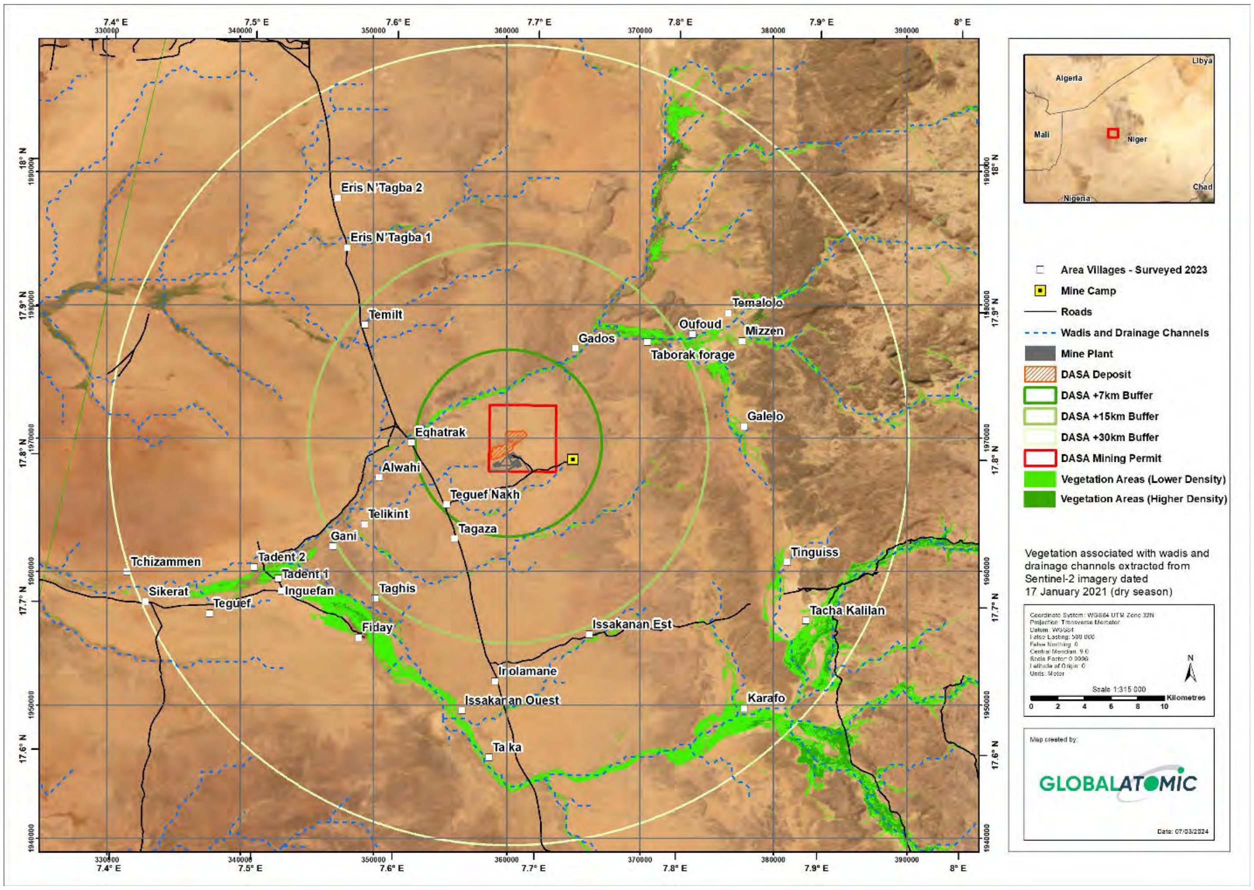

Figure 20-2: Map of the Project Area Showing Buffer Zones and Proximal Communities.

Air Quality

The current sources of airborne emissions in the area include construction work and traffic associated with the Dasa Project but are largely natural and consist of windblown dust. Naturally dusty conditions prevail in the region due to the soil type, desert climate, strong and hot winds, and relatively flat topography. Anthropogenic emissions in the wider region are associated with the coal-fired power station outside Agadez and from existing uranium mining operations to the north around Arlit, albeit at some distance from the Project.

Air quality measurements carried out in the Project area have shown levels of particulate matter (i.e. fine dust) that sometimes exceed WHO health standards. This is largely a natural phenomenon resulting from wind-blown dust from the desert surface, as opposed to anthropogenic sources.

Noise and Vibration

The remote location and absence of human habitation and industry in the Project area mean that anthropogenic noise levels are currently low; no significant sources of vibration exist. The Dasa Project is a significant distance from sensitive receptors and as a result, noise studies have not been included in baseline studies carried out to date.

Hydrology and Hydrogeology

There are no permanent water courses in the Project area. There is, however, a network of koris (ephemeral watercourses), the main flow direction of which is west-southwest from the Aïr Mountains in the east. The Project area lies between the Agatara and Tagaza koris which run approximately 4.5 km to the north and 1.5 km to the south of the mine site, respectively. A smaller, unnamed kori runs just north of the Project location.

The kori channels are characterized by short duration, high flow events in response to heavy seasonal rainfall. They remain dry for most of the year, but flash floods can occur as a result of local storm events.

The Project is located within the Tim Mersoï sedimentary basin, part of the much larger Iullemeden Basin that stretches into Mali, Algeria, Benin, and Nigeria. The Tim Mersoï Basin strata have a shallow westerly dip caused by the uplift of the Air Massif. The rocks within the Project area are predominantly clastic sediments with minor carbonates, which originated from the Air Massif. The Dasa site corresponds to a structural intersection of two major structures, the Adrar-Emoles flexure and the In Asouza-Arlit fault, which resulted in the creation of the Dasa Graben.

The hydrogeological units comprise a sequence of alternating layers of high to moderate permeability sandstones and low permeability siltstones and mudstones. The Teloua unit is considered a regionally important aquifer and the main water bearing unit within the Project area. The shallower Tchirezrine 1 and 2 units are considered secondary aquifers, important on a local scale for domestic and agricultural supplies. The Precambrian basement rocks are conceptually considered a basal aquiclude.

The geometry of the aquifer units is strongly influenced by the tectonic events that have occurred in the region. Regional and local faults have influenced the depth, thickness, extent, and interconnection of the aquifer units throughout the area and locally in the vicinity of the proposed mine. Fault zones can act as barriers to flow, resulting in compartmentalisation of units, or conversely, can act as preferential flow paths resulting in greater hydraulic connection between different units.

Three principal phases of hydrogeological investigations have been completed for the Dasa Project, in 2013-2014, 2020 and 2021. A total of eleven hydrogeological boreholes have been drilled and tested within the Project area. Extensive hydraulic testing (pumping test) programmes have been completed on the boreholes.

Groundwater levels in the Project area were recorded at the time of borehole drilling/testing, and regular groundwater level monitoring was initiated in June 2021. The depth to groundwater within the boreholes monitored in 2021 ranged from 27 to 47 m bgl. The regional groundwater flow direction is generally from east (Air Mountains) to west. Groundwater recharge is considered to be negligible, with previous investigations suggesting recharge of less than 5 mm per year. Recharge, when it occurs, will primarily occur via infiltration along koris.

Groundwater Quality

Groundwater samples were collected from 23 borehole locations within the Project area between 2013 and 2021. Groundwater was found to be generally of good chemical quality, with the majority of parameters tested falling below World Health Organisation ("WHO") drinking water standard limits, with a few exceptions relating to nitrite and fluoride. In addition, bacteria were detected at certain boreholes.

One round of radiological testing was carried out in March 2020 across 15 borehole sample locations. Parameters tested included gross alpha activity, gross beta activity, potassium 40, and dissolved potassium. The results indicated that radioactivity levels in groundwater exceed WHO guideline limits at certain borehole locations:

- The WHO limit for gross alpha activity in drinking water (0.5 Bq/l) was exceeded in 8 out of the 15 boreholes sampled; the exceedance values ranged from 0.9 to 5.31 Bq/l; and,
- The WHO limit for gross beta activity in drinking water (1 Bq/l) was exceeded at 2 out of the 15 boreholes sampled; the exceedance values were 1.55 and 2.45 Bq/l.

In 2022, nine water samples were collected from boreholes in the Project area, and subjected to laboratory analysis for global alpha activity, global beta activity, potassium-40, and dissolved potassium density. The gross alpha results ranged from 0.18 to 5.40 Bq/l and the gross beta results ranged from 0.20 to 2.90 Bq/l.

The implications of the groundwater quality with respect to human exposure to radioactivity are discussed later in this chapter.

In 2023 water samples were taken from 12 locations on a quarterly basis from the villages of Agatara, Tagaza and Elagozan as well as locations in the camp and mine site. The 2023 program confirmed results from earlier programs and provided visibility as to results over the course of the year. The 2023 program further expands SOMIDA’s database of radiation and chemical property baseline data of area water sources.

Soil

Soils are poorly developed and frequently eroded by wind and intense rainfall events. The soils present consist largely of sand, the result of water erosion in the Aïr Mountains and wind erosion in the Ténéré; gravel, mainly the result of erosion in the Aïr Mountains and found in the kori beds; and detrital clay, fine particles washed down by the koris and also resulting from erosion of local shales. Soil cover is typically 0.5 m thick.

Biodiversity

Biodiversity surveys have been carried out at the site by both Groupe Art & Génie and FEED Consult for the 2020 and 2022 ESIAs. Most recently, in 2023, a Critical Habitat screening assessment was carried out by Treweek Environmental Consultants Ltd and Abell Geospatial Consulting Ltd. The Critical Habitat Assessment was undertaken with the aim of aligning the Project with the requirements of IFC PS6.

There are seven legally protected and/or internationally recognised sites of importance for biodiversity within the region, including the Aïr et Ténéré Man and Biodiversity Reserve, a UNESCO World Heritage Site. However, these sites are located more than 100 km from Dasa, and the Project is not considered likely to impact any of them.

Ecological surveys undertaken in the Project area during the dry season recorded 29 floral species, with 17 being herbaceous and 12 woody. In the wet season, 38 species were recorded: 25 herbaceous and 13 woody. Of the woody species, five were identified as protected in Niger. It was possible to recognise distinct groupings of species according to three types of terrain: koris, plains, and rocky plateaux.

For mammals, there were direct observations of Dorcas gazelle (six individuals seen), squirrel, Golden jackal, and Cape hare. There were indirect observations of Aoudad (barbary sheep), Ratel (honey badger), Fennec fox, Pale fox, and African wild cat. Of these, Dorcas gazelle and Aoudad are classified as Vulnerable on the International Union for the Conservation of Nature's Red List of Threatened Species (the "IUCN Red List").

Observed reptiles included Horned viper, Cobra, Sand boa, Uromastyx and common lizards. The Uromastyx (spiny-tailed lizard) is listed as Near-Threatened on the IUCN Red List.

The Dasa property is located on an avian migration route. Thirty-four bird species were observed during the surveys, including the Egyptian vulture and Lappet-faced vulture, both of which are listed as Endangered on the IUCN Red List; and the Tawny eagle, which is classified as Vulnerable.

As defined by IFC PS6, a critical habitat is an area of high biodiversity value due to the presence of endangered, endemic, restricted-range, or migratory species; and/or its supporting key evolutionary processes (IFC PS6 contains detailed qualifying criteria).

Although the surveys identified the presence of several species listed as threatened on the IUCN Red List, it is considered unlikely that these will trigger a critical habitat determination, because they are all wide-ranging species. Based on GIS spatial assessment, desktop critical habitat screening, historical and recent fieldwork, there is no critical habitat in the Project's area of influence (defined as a 50 km radius around the mine site).

Several of the observed flora and fauna species are used by local communities for food, fuel, or medicinal purposes and, as a result, biodiversity is under threat in the Project area. Degradation and destruction of wildlife habitat and climate change are additional pressures. During interviews with communities, reference was made to certain species that have disappeared from the area completely, such as Dama gazelle, Oryx, and the Common ostrich.

Acid Rock Drainage ("ARD") and Metals Leaching

Test work conducted by SGS, Lakefield, Ontario in March 2011, on a 100 kg bulk sample of “ore” included Modified Acid Base Accounting ("ABA") and Net Acid Generation ("NAG") testing. The following results were reported in Table 20-1 below.

Table 20-1: Acid Rock Drainage and Metal Leaching Results.

Description	D1 Comp	D2 Comp	D3 Comp	Overall Comp
Net NP (t $CaC0_3$/1000 t)	10.6	4.79	3.99	5.99
CO_3 Net NP (t $CaCO_3$/1000 t)	0.44	0.11	0.95	<0.01
NAG @ pH 7.0 (kg H_2SO_4/t)	0	0	2.0	0

In addition to the above, elemental analysis conducted by SGS Canada confirmed that elemental iron, sulphide mineralization and sulphur are not present in significant amounts in the Dasa samples (elemental sulphur was reported as 0.02% for the overall composite sample). The results above indicate sufficient neutralising potential is present in the ore such that ARD generation and associated metals leaching is highly unlikely to occur.

Testing of neutralized tailings samples in 2022 also confirmed low potential for acid generation and low concentrations of potentially toxic elements. Leachate testing indicate the tailings will not generate hazardous leachate, although it was noted that the concentration of uranium in the leachate, at 7.93 mg/L, is close to the 10 mg/L reference standard (being the limit generally imposed in Canada). In any case, at Dasa there will be no discharge of liquids from the DSTSF; any seepage will be collected and routed back to the DSTSF.

Radiological Exposure Baseline

The Project area is located in a region of elevated background radiation due to the natural presence of high concentrations of uranium in on and below surface rocks and soil. Key exposure (dose) rates for workers and local residents are based on external atmospheric radiation; external radiation received from the ground; inhaled dust and gases; ingestion of radionuclides on foodstuffs; and radionuclides contained in drinking water.

Baseline surveys carried out during the period 2020 - 2023 focused on measurements of external exposure dose rates at several points around the deposit; measurements of uranium 238 concentrations in soil samples; and measurements of gross alpha and gross beta volumetric activity in samples from water supply sources (wells and boreholes) in the villages and towns within a 20 km radius around the Dasa deposit.

The 2020 ESIA provides the results of a total of 33 soil sample points around the deposit for dose rates, as well as 13 water samples including the GAC camp and water supply borehole. Village water supplies sampled included Tilkin, Taden, Guifayen Digui, Adaley, Belaten, Tegazaou Tziliyaman, Teragan, Gani, Inolamane, Tagaza and Elagozan "Jardins".

Based on these results, the annual natural external exposure dose which would be received by a member of the general public living in the area was determined to vary between 2.80 milli Sieverts (mSv) at a continuous dose rate of 320 nSv/h) and 0.53 milli Sieverts (mSv) at a continuous dose rate of 60 nSv/h).

From June 2021 to December 2023, a program of continuous sampling was undertaken to establish the natural background radiological level of the area. The program comprised the use of passive detectors (thermoluminescent dosimeters) to assess external exposure doses due to natural gamma and beta ionizing radiation; and continuous sampling and radiological analysis of atmospheric air to determine average volume concentrations of radon potential alpha energy (Rn220, Rn222) and alpha activity of long-lived Uranium 238 and Thorium 232 in airborne dust.

Four monitoring stations were established: at the accommodation camp, the mine site, and at Tagaza and Agatara villages.

Results were processed to determine the annual dose received by a standard member of the public, as tabulated below.

Table 20-2: Air Monitoring Results.

Air Recording Station	External exposure dose mSv/year	Internal exposure dose			Total dose mSv/year
		EAP Rn220 mSv/year	EAP Rn222 mSv/year	EAVL mSv/year	
Station 1 (Camp)	0.88	0.20	0.51	0.011	1.60
Station 2 (Mine Site)	1.70	0.25	0.76	0.011	2.72
Station 3 (Tagaza)	0.71	0.15	0.62	0.005	1.48
Station 4 (Agatara)	0.74	0.20	0.80	0.014	1.75
Average	1.08	0.20	0.67	0.01	1.96

mSv/year = milli Sievert per year; EAP = potential alpha energy; EAVL = long-lived alpha emitter

Table 20-3: Monitoring Results – Air and Water.

Stations / Doses	Station 1 Camp mSv/year	Station 2 Mine Site mSv/year	Station 3 Tagaza mSv/year	Station 4 Agatara mSv/year
Cumulative Dose mSv/an	1.6	2.72	1.48	1.75
Cumulative Dose intégrant ITD mSv/an	1.75	2.781	1.495	1.852

ITD = Indicative Total Dose

The radon-220 / thoron gas effective doses are considered very low, at 0.15 to 0.25 mSv/year. The radon-222 effective doses, though three times higher than those obtained for radon-220, are also low from the point of view of radiation protection of members of the public. The effective doses for long-lived alpha emitters are of the order of a few hundredths of a mSv/year and can effectively be considered as zero; there is practically no uranium-238 or its long-lived alpha emitting progeny in the atmospheric air of the area.

The average cumulative dose in the study area is 1.96 mSv/year, which is lower than the global average background level of 2.4 mSv/year. The cumulative annual doses measured at the mine site exceed the global average.

For the camp and mine site, external exposure doses to natural ionizing radiation (gamma and beta components) are higher than those due to internal exposure through inhalation of radionuclides contained in atmospheric air (radon gas and its short-lived alpha emitting progeny, uranium and its long-lived alpha emitting solid progeny). Opposite results were received for Agatara and Tagaza villages. The contribution of the uranium / long-lived alpha component is practically zero, and that of Rn220 is also negligible.

After the additional measurements carried out in 2023 following the recommendations of the Kando Report (2022), the highest values recorded for the different exposure scenarios including the water carrier are considered to represent the basic condition, which equates to an annual dose of (1.70 + 0.25 + 0.80 + 0.014+0.15) = 2.91 mSv. This figure should be considered the highest level of natural exposure to ionizing radiation encountered in the area of the DASA mine project.

Kando's recommendation for additional sampling for the assessment of exposure to external gamma and beta radiation at stations 2 (mine site) and 4 (Agatara) was taken into account by repeating measurements at all stations.

For water, following the recommendation of the Kando Report (2022) to carry out detailed radiological analyses to determine radionuclide concentrations; complete radiological analysis, including ITD determinations was carried out.

For two boreholes (Base Vie Global Borehole and Agatara Village Borehole) the results of the analyses gave Total Indicative Doses (DTI) above 0.1 mSv/year, which is the quality reference for water intended for human consumption. Although these DTIs of 0.102 mSv/year (Agatara Village Drilling) and 0.15 mSv/year (Global Base Borehole) are well below 0.3 mSv/year which represents the practical limit for corrective actions, it would be necessary, during uranium mining, to periodically check the quality of these waters for human consumption or any other use. The Agatara borehole also has a chemical concentration of uranium (92 µg/l) above the WHO standard (30 µg/l).

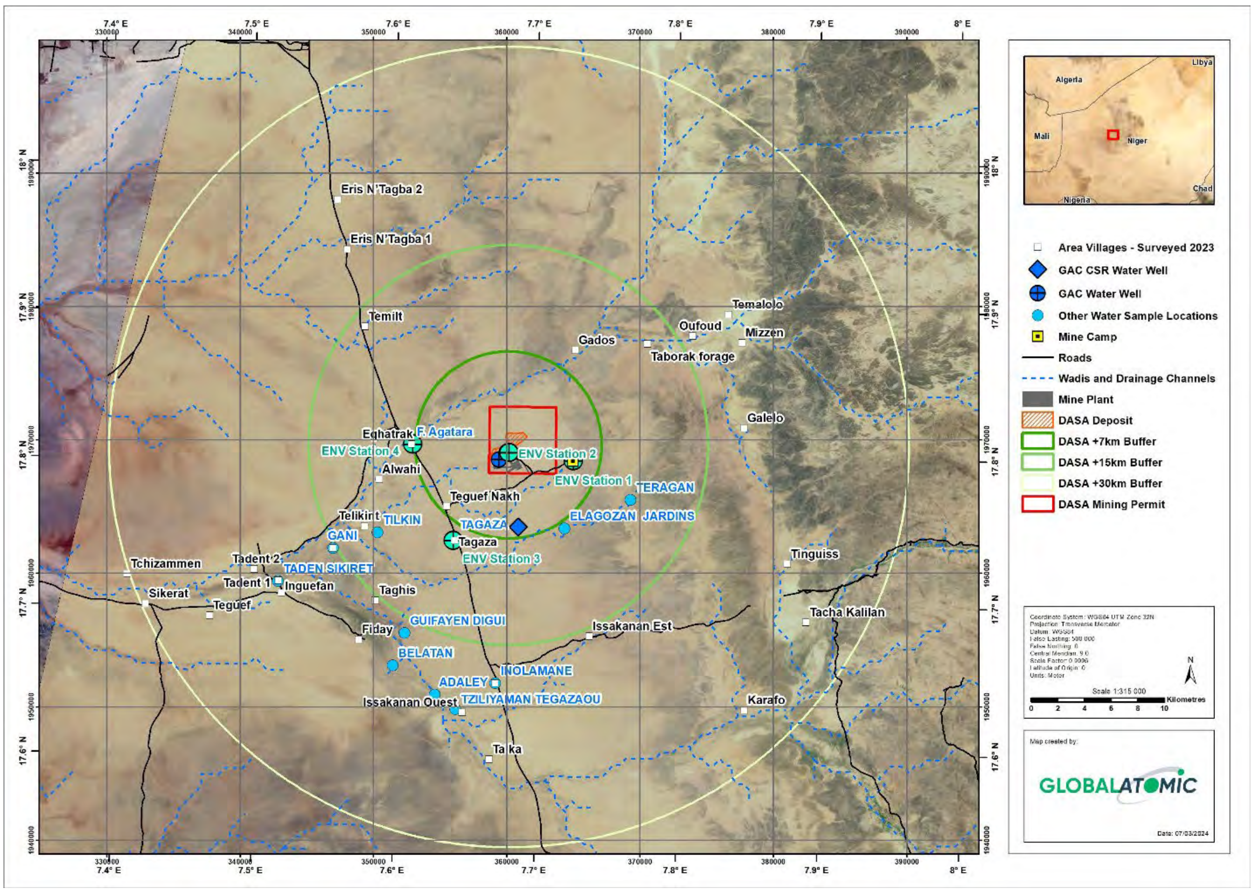

Figure 20-3: Dosimeter and Water Well Sampling Locations.

Socioeconomics

The Tchirozérine department, in which the Dasa area is located, extends from the Mali border in the west, to the Aïr Mountains in the east, from Arlit in the north and south to Ingall in the Agadez region – an area of 154,746 km^2. The population density in the Project area is less than one inhabitant/square kilometre, with over 70% of the population living in the administrative centres of the communes, department, and regions.

The total population of the two communes of Tchirozérine and Dannet, which are considered to be outside the area of direct influence but within regional influence of the Project, is estimated at 116,630 inhabitants, with, respectively, 80,000 inhabitants in the urban commune of Tchirozérine and 36,630 in that of Dannet

(CSA PEA 2020). This population is characterized by its large number of young people. For example, according to the 2017 PDC for Tchirozérine, a quarter of the population is between 0 and 14-years of age and 36% is between 14 and 40-years of age. This population is of Kel-Tamashek (Tuareg) origin and consists of several tribes belonging to the Kel Ewey Confederation. These tribes belong to three chiefdoms: Sultan, Anastafidat and Imakitan, which all live outside of these communes (Agadez and Timia).

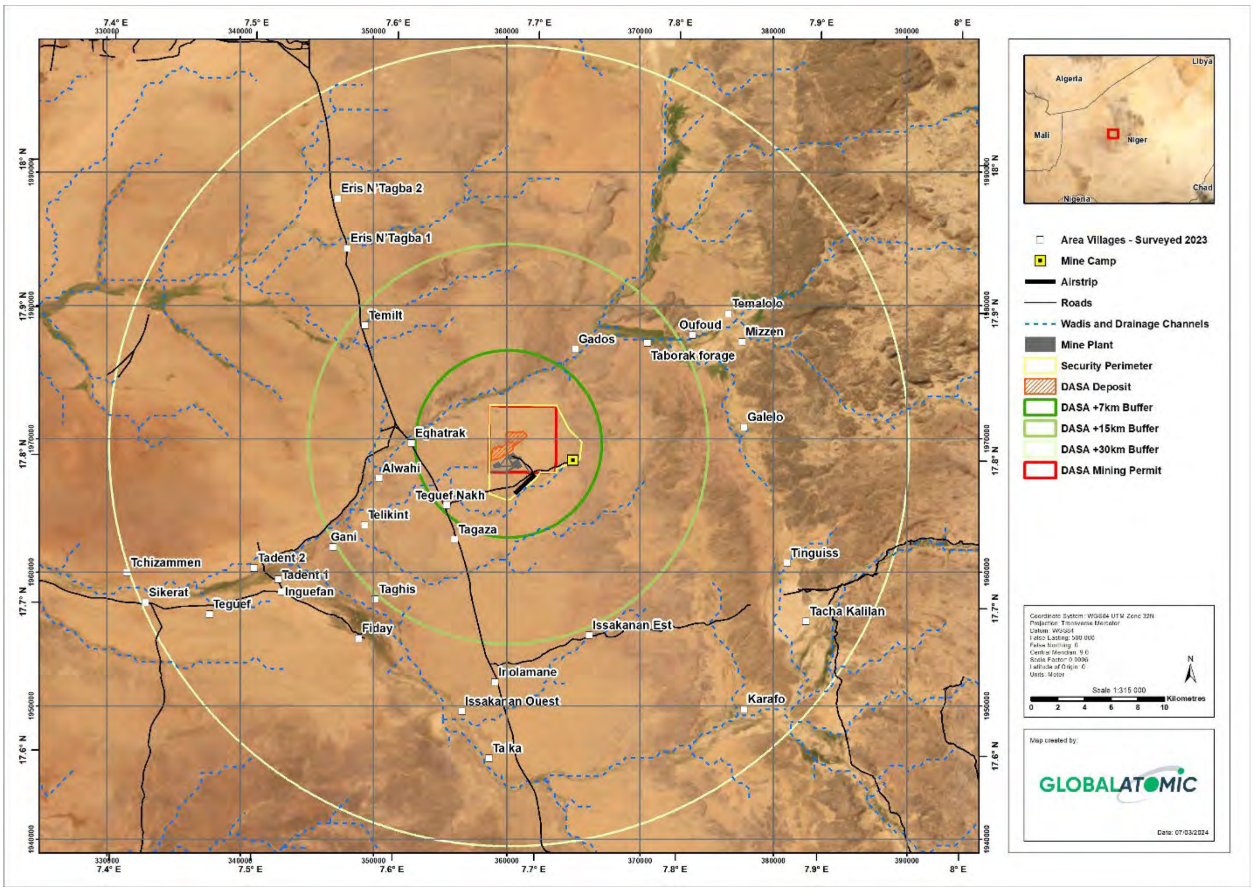

Table 20-4:Project Location Showing Nearby Communities.

The area immediately surrounding the Project site is sparsely populated, with most people living in villages, the nearest of which are Agatara, Teguef Nakh and Tagaza, more than 5 km to the west. Within a radius of 15 km the total population is 14,830 people, according to surveys undertaken in 2021. Of these, approximately 10,000 are permanent residents, the remainder being present only during the dry season.

Small clusters of huts occupy land along the koris. Settlement within the koris is limited to families with small groups of animals on an approximate 200-300 m spacing. During the wet season from July to September and winter season from December to March, the inhabitants move to the edges of the koris where it is dryer and warmer respectively.

The nearest permanent settlement to the Project site is a collection of three huts approximately 1.5 km to the east-southeast. The huts are occupied by a single family. The head of the family has been an employee of GAC (and now SOMIDA) since the exploration phase of the Project is currently employed by the Nigerien Catering Company that provides food services at the camp. There are not believed to be any permanent settlements within the 36 km^2 buffer area around the Project site.

The local Kel Tamashek (Tuareg) population is considered to be an 'indigenous people' in the context of IFC PS7. This standard recognizes that indigenous peoples, as social groups with identities that are distinct from mainstream groups in national societies, are often among the most marginalized and vulnerable segments of the population and may have limited capacity to defend their rights and interests, or benefit from development projects. Since 2008, GAC has been consulting and seeking the informed participation of the local community. More recently, broad community support for the Project has been demonstrated at both the local village and regional administrative levels, through receipt of signed letters of support.

The Kel Tamashek (Tuareg) has historical connections with the natural environment and traditionally had migrated between regions and between departments in search of pasture and seasonal jobs. It is mainly the men who travel while the women stay in their village. With the decline in traditional livelihoods, young people migrate to urban areas in search of alternative lifestyles. Migration from other parts of Niger, as well as from neighbouring countries, is mainly associated with job opportunities in the mining industry. For several years now, Arlit, Akokan, Tchibarakaten, Djado and Tchirozérine have been multi-ethnic centres serving the mining industry.

Livestock breeding is the main activity of the people of the Agadez region. The region has a large livestock population consisting mainly of camels, goats, sheep, donkeys, and cattle. There are estimated to be about 15,000 head of livestock within a 15 km radius of the Project. This concentration of livestock has resulted in degradation and overgrazing.

The infrastructure that constitutes the watering points for livestock in the area include wells installed by GAC, pastoral wells, traditional wells, boreholes, and temporary ponds. In the Project area, it is understood that the east-northeast to west-southwest trending koris are used by local pastoralists. Use of the koris as a transit or nomadic herding corridor is limited in scope and primarily undertaken in connection with the "Cure Salee", a meeting of Kel Tamashek (Tuareg) from around the region which celebrates the end of the rainy season and is held at the Town of In-gal, located approximately 150 km south-west of Dasa.

A small amount of market gardening is practised by the population at Elagozan, approximately 5 km south of the mine site. The estimated area under cultivation within the 15 km radius study area is 7.29 ha. The main crops are vegetables including lettuce, peppers, cabbage, carrots, and watermelons. Some of the produce is consumed locally and the rest is sold at markets in Arlit, Tchirozérine and Agadez.

Trade in the Agadez region is characterised by the predominance of the informal sector, in which a multitude of retailers and a few semi-wholesalers operate. The towns of Agadez and Arlit are the main commercial centres of the region, in addition to which there are approximately ten rural markets.

In the Dasa Project area, commercial activities are mainly based on small-scale trade, in particular the sale of livestock products, market gardening, woodcutting, and charcoal making.

The Project area of influence has elementary schools and middle schools, with a secondary school in Tchirozérine. Schools are also located in Galelo, Inabizguin Solomi and Tagaza (2011 ESIA). Koranic schools are also present which tend to be in less permanent structures with fewer facilities. Many of the schools operate with international NGO support. Vocational training is also available in the region.

The healthcare facilities identified in the two communes include a hospital run by the Niger government power company, SONICHAR, health clinics and pharmacies. The healthcare services are considered limited by the local authorities, with a lack of medicines as well as poor quality facilities.

Archaeology and Cultural Heritage

FEED Consult carried out a comprehensive review of archaeology and cultural heritage in the Project area for the 2022 ESIA, including consultation with local authorities. Of the cultural and archaeological sites identified, the Dabous giraffe carvings site, located approximately 9 km north-west of Dasa, is known worldwide. None of the identified sites are located within the Project license area. However, there are two small graveyards located along the access track to the west of the exploration camp.

20.3. Impact Assessment, Mitigation and Management

Impact Assessment

The assessment of potential impacts was carried out for each of the construction, operation, and closure phases of the Project. For each phase, the activities which might cause impacts were identified and then compared with a list of elements of biophysical and human environments (i.e. ‘receptors’) which might be affected, to identify likely impact scenarios.

Then, for each scenario, a characterisation exercise of the potential impact was undertaken, taking into consideration the likely intensity of the impact, the perceived value of the receptor, the degree of disturbance, its spatial extent, and the duration. Based on this characterisation, the significance of each potential impact was evaluated as either Minor, Medium, or Major, and either positive or negative.

The following summarises the potential Major and Medium impacts identified for the three Project phases.

For the construction phase, one potential impact was assigned a Major positive significance: the effects of the Project on the economy, including local employment. No potential impacts of Major negative significance were identified.

Potential impacts of medium significance associated with the construction phase included:

- Contamination of soil by fuel, oil, and solid and liquid wastes.
- Degradation of air quality by exhaust gas emissions and dust.
- Depletion of groundwater resources by extraction for Project use.
- Modification of surface drainage patterns.
- Contamination of water resources.
- Disturbance to fauna by habitat destruction, noise, vehicle movements, and poaching by Project staff.
- Loss of vegetation due to site clearance and smothering of nearby vegetation by dust.
- Health and safety risks to workers and local communities, including accidents, disease transmission, contamination, and risks of conflict.
- Noise nuisance; and,
- Reduction of access to land for pastoral activities.

For the operational phase, the effects of the Project on the economy were again assigned a Major positive significance. No impacts were assigned a Major negative significance.

Medium-significance impacts associated with the operational phase included:

- Increased soil erosion and contamination of soil by fuels, oils, process chemicals, solid and liquid wastes, and radioactive dust.
- Degradation of air quality by exhaust gas emissions from mobile plant and vehicles, and fugitive emissions from processing.
- Depletion of groundwater resources from extraction for Project use and mine de-watering.
- Contamination of water resources by process chemicals, or discharge of untreated wastewater.
- Disturbance to fauna by habitat destruction, noise, vehicle movements, poaching by Project staff, and risk of mortality from falling into ponds.
- Degradation of landscape quality (i.e. visual impact).
- Health and safety risks to workers and local communities, including accidents, disease transmission, contamination (including radiological), and risks of conflict.
- Noise nuisance.
- Reduction of access to land for pastoral activities, and potential injury to livestock from contamination, vehicle collisions, and falling into ponds.
- Decline of local traditions and customs due to in-migration of people; and,
- Degradation or destruction of archaeological or cultural heritage.

For the closure phase, the effect on the economy and local employment was considered a Major negative impact, due to the loss of direct and associated jobs and revenue after the mine closes. On the other hand, removal of mine infrastructure and restoration of the affected area was considered a Major positive impact on flora, and a Medium positive impact on fauna and landscape character.

Medium-significant negative impacts associated with the closure phase included:

- Potential contamination of soil by fuel, oil, solid and liquid wastes, and radionuclides during dismantling activities.
- Degradation of air quality by exhaust gas emissions and dust.
- Potential contamination of water resources by fuel, oil, solid and liquid wastes, and radionuclides during dismantling activities, Health and safety risks to workers and local communities, including accidents, disease transmission, and contamination.

In considering the potential for cumulative impacts, it is recognised that the Dasa Project lies within an established uranium mining region. Although the Cominak mine near Arlit (about 110 km north of Dasa) closed in March 2021, the Somair mine, also near Arlit, is currently operating and expected to do so until at least 2035. Also, near Arlit is the Madouela project, for which a mining feasibility study was published in

2022. Approximately 50 km west of Dasa is the Imouaren deposit; designed as a large open pit operation, development has been on hold since 2015, pending improved market conditions.

Other than the Somair mine there is little other industrial development in the area. However, should both the Imouaren and Madouela projects come on stream during the Dasa project's life, social impacts may become significant. These may include common pressures associated with influx of workers, including inflation of the local economy, pressure on local infrastructure and services, over-depletion of natural resources, and loss of traditional cultural heritage and ways of life.

Mitigation

The following briefly summarizes the mitigation measures deemed necessary to reduce the significance of negative impacts, or enhance the significance of positive impacts, identified above. They include measures to align the Project with good international industry practice, including EP4, IFC PS, IFC EHS Guidelines, and IAEA guidance.

Greenhouse Gases ("GHG") and Climate Change

SOMIDA has estimated its base-case operations-phase GHG emissions as 65,395 tonnes per annum ("tpa"), which assumes that most of the Project's electricity will be provided by coal-fired power via the Nigerien national grid, and that vehicles will be fuelled by diesel. There is an optimized plan to install solar photovoltaic ("PV") panels linked to battery storage and back-up diesel, with the intent of providing approximately 20% of the Project's total requirement in the form of renewable energy. This would reduce the total estimated GHG emissions to 52,871 tpa. Furthermore, there is a conceptual plan to reduce the mine site power demand from 12 megawatts ("MW") to 9 MW which, coupled with solar PV and battery storage, and back-up diesel, would target a reduction in GHG emissions to 43,000 tpa; a 34% reduction from the base case scenario.

SOMIDA also plans to introduce battery electric vehicles to the underground and surface fleets over time to the extent practical.

It should be noted that, according to the European Nuclear Society, one kilogram of natural uranium, following enrichment and used for power generation in light water reactors, generates 45,000 kWh of electricity, which is equivalent to the electricity generated by the combustion of 14,000 kg of coal.

By 2050 in Niger, the African Development Bank predicts an increase of between 2.0 °C and 2.5 °C in average, with rainfall either unchanged or increasing by up to 50%, and the number of heavy rainfall days, and number of rainy days per year, also either unchanged or increasing by up to 50%. These predictions suggest three main actions to be considered by SOMIDA in implementing the Dasa Project:

- Ensure potential heat-related effects (thermal stresses) are addressed in occupational health and safety planning.
- Ensure Project infrastructure is protected from potential surface water flooding; and,
- Support local initiatives for agricultural efficiency and food security for local people.

Air Quality

A dust management plan has been implemented (as part of the Project's Air Quality and Greenhouse Gas Emissions Management Plan) and is in line with good international industry practice. The plan aims to minimize dust emissions by controlling vehicle speeds, transferring dust-generating materials with a minimum height of fall, clearing undisturbed areas only when absolutely, necessary, and immediately prior to construction works, and covering and re-vegetating exposed soils as soon as possible. Roads and working areas susceptible to dust generation will be watered, and dust suppression spray systems and covers will be deployed where necessary.

The Project will employ standard good international industry practices ("GIIP") for the minimization of air emissions, including procurement and maintenance of fuel-efficient vehicles and equipment, provision of training in fuel-efficient practices for drivers and operators, provision of sulfur dioxide and nitrogen oxides control systems at point sources, and establishment of leak detection and repair programs for fugitive emissions.

Ambient air quality will be monitored at points around the Project site boundary. An Environmental Monitoring Plan will be established to define the number of monitoring stations required, and the methodologies to be employed. The residents of the huts 1.5 km east-southeast of the site will be considered the primary potential receptors for monitoring.

Noise and Vibration

The Project will employ GIIP for the minimization of noise, including designing enclosures or sound barriers for source equipment, procuring, and properly maintaining equipment with lower sound power levels, optimizing traffic routing, and reducing working at night when possible. A mechanism has been developed to record and respond to complaints, and the Environmental Monitoring Plan includes a program of periodic noise and vibration monitoring (the latter for at least the first few underground blasts and confirms that resulting vibration is not of concern).

Soils

Soil (and water) resources will be protected by the implementation of GIIP for materials and waste handling. A Hazardous Materials Management Plan is in place to address both occupational health & safety and environmental risks, and includes the maintenance of a hazardous materials inventory, job safety analysis, hazard communication and training programs, provision of personal protective equipment ("PPE"), emergency eyewash and shower stations, ventilation system, sanitary facilities, and monitoring and record-keeping.

Hazardous materials control measures include secondary containment for liquids, impervious surfacing of areas used for the transfer of hazardous materials between vehicles and storage, use of dedicated fittings, pipes, and hoses specific to hazardous materials in tanks, and regular inspection and maintenance thereof.

A spill response and management plan has been formulated as part of the overall Emergency Preparedness and Response Plan. For each hazardous material in the site inventory, analysis will be undertaken of potential spill and release scenarios, the potential for uncontrolled reactions such as fire and explosion, and the potential consequences in terms of effects on Project workers and the surrounding environment. Project staff are trained in release prevention, and inspection programs implemented to maintain the mechanical integrity and operability of pressure vessels, tanks, piping systems, etc. Standard Operating Procedures govern the filling of storage tanks and other containers or equipment, and for transfer operations, by trained personnel. Specific PPE, spill response equipment and training needed to respond to an emergency are available. Response activities in the event of a spill, release, or other chemical emergency will be documented.

A Waste Management Plan has been prepared, to include procurement measures that recognize opportunities to return usable materials; minimizing hazardous waste generation by implementing stringent waste segregation; establishing recycling objectives and formal tracking of waste generation and recycling rates; biological, chemical, or physical treatment of waste material to render it non-hazardous prior to final disposal; and ensuring that contractors handling, treating, and disposing of hazardous waste, and the receiving facilities, are reputable, legitimate enterprises, licensed as applicable, and following GIIP for the waste being handled.

Hazardous waste is stored in closed containers away from direct sunlight, wind, and rain, and with secondary containment where appropriate, in a manner that prevents contact between incompatible wastes.

Materials to be transported to or from the mine and which are both hazardous and present risks in terms of security are subject to special arrangements. These materials include explosives, detonators, uranium concentrate product, and radioactive wastes. These materials will be transported by formal convoy, including escort and security vehicles.

Surface Water

Mine site construction will avoid the main koris which traverse the area. During clearing and construction works, topography will be respected and disturbed areas will be restored as quickly as possible to avoid the risk of altering the drainage system. Diversionary or collection channels will be installed both to manage the probable maximum flood level, and to allow sufficient retention time to allow suspended solids to settle out. Site water management systems will be designed to separate clean water from contact water. No contact water will be discharged to the environment; it will be routed to storage ponds for use in processing or evaporation.

A flood risk area has been identified north of the mine infrastructure. While maximum flood levels are not predicted to impact the mine facilities, flood protection measures in these northern areas will be considered in order to reduce the potential flood risk to mine surface infrastructure.

Groundwater

Measures for protecting groundwater against contamination by hazardous materials and wastes are the same as those listed above for soil protection. In addition, HDPE liners will be installed at the bottom of the DSTSF and storage ponds to avoid the risks of infiltration, and such structures will be monitored to detect possible leaks.

Groundwater modelling has predicted that inflow of groundwater to the mine will be greater than the total Project water demand, including processing and domestic needs. This groundwater must be removed to allow mining to progress safely and results in a risk of natural groundwater levels in the surrounding area being drawn down. Modelling suggests water levels in wells in Tagaza and Agatara may decline by around 2 m as a result of mine dewatering, whereas at the market gardens of Elagozan (5 km south of the mine) the decline may be 10 m or more at end of mine life (23-years).

Any impact to local wells would be gradual, and detectable by appropriate monitoring, thus enabling the early planning of appropriate mitigation measures (e.g. provision of an alternative water supply, lowering the pump in the existing well, deepening the existing well, or installing a replacement deeper well).

As per IFC PS3, the Project has an obligation to work towards using natural resources including water, in a sustainable manner. SOMIDA is investigating strategies to reduce the inflow of groundwater to the mine, in order to reduce the requirement for dewatering and thus minimize the need for water handling, storage and disposal. Such strategies should also lower the risk of significant drawdowns in local community wells. Strategies under consideration include a combination of targeted grouting to block water inflows and the installation of dewatering boreholes. Also under consideration is the extraction water up-gradient of the mine and re-injection into the aquifers down-gradient of the mine.

Biodiversity

The measures for air quality, noise, and hazardous material and waste management listed above are also applicable to biodiversity. The land areas disturbed and used for the Project will be reduced to the minimum necessary. These areas will be clearly delineated (by fencing or otherwise) and there will be no encroachment outside them (this applies particularly to off-road driving). Trees or areas of dense vegetation will be retained when possible (species protected in Niger will be respected).

Soils removed from the Project footprint will be stockpiled for future use, and disturbed areas will be restored as soon as possible and progressively whenever possible. Placed soils will be revegetated promptly, to reduce erosion and dust generation. Areas presenting potential hazards to fauna (e.g. deep excavations, ponds, chemical storage areas) will be made secure (e.g., by fencing).

Poaching is prohibited, workers will be made aware of the importance of protecting wildlife, and there will be monitoring for the presence of invasive species.

Occupational Health and Safety

Standard GIIP in health and safety management will be employed at Dasa. Given that the Project is a uranium mine, occupational health, and safety with respect to the ionizing radiation hazard is a primary concern. The Dasa mine benefits from the employment of experienced senior staff formerly employed at the Cominak underground uranium mine located approximately 110 kms north of the Project. A Radiation Management Plan has been implemented, and includes an organizational structure which addresses accountability, responsibilities, and roles; arrangements for the measurement of radiation levels at the site and potential exposures of workers and the public; the designation of areas where radiation control is required; safe operating procedures and rules, including supervision; maintenance of a data recording and reporting system related to the control of radiation, exposure of workers and decisions on measurements for occupational radiation protection; a training program on radiation hazards and requirements for protection; an emergency response plan; and a health surveillance program.

SOMIDA has also developed a procedure designed to protect workers from silicosis (a lung disease caused by the inhalation of silica dust). The procedure, which reflects the methodology employed at the Cominak mine, involves establishing a reference dust level (based on the flow rate of air through the mine), and classification of each zone of the mine (and process areas as applicable) according to its dust content. Mitigation measures are applied according to the classification level, including for example, water sprays and dust extraction, as well as appropriate levels of PPE. Personnel who have worked in high-dust areas are subject to additional medical surveillance.

There is an infirmary at the camp, staffed by two nurses working on complementary rotation to ensure a permanent presence. The infirmary is equipped to treat medical emergencies, minor illnesses, and injuries for employees, contractors, and area villagers. SOMIDA is currently working towards finalizing an agreement between the Arlit Health District Hospital (formerly COMINAK Hospital) and SOMIDA. SOMIDA is also in discussions with ORANO to arrange access to its medical centre at the SOMAIR mine site in Arlit for more serious injuries. There is also a regional hospital in Agadez.

Community Health and Safety

Although the Project site is relatively remote, there are health and safety risks to the local population when the Project and community interact. SOMIDA has established a Community Health, Safety and Security Plan guided by IFC EHS General Guidelines, the United Nations Environment Program's Awareness and Preparedness for Emergencies at Local Level ("APELL") standard, and the Voluntary Principles on Security and Human Rights ("VPSHR"). The APELL process aims to improve community-level emergency preparedness efforts and supports government and community initiatives to minimize the occurrence and harmful effects of technological hazards and environmental emergencies.

Although the Project Community Health, Safety and Security Plan is intended to be comprehensive (i.e. to be in line with IFC EHS Guidelines), the ESIA results require that the following be given particular attention:

- Safe transport of hazardous materials on public roads (explosives, chemicals, uranium product, and wastes).
- Radiological hazards, mitigation, and monitoring.
- Awareness raising of the risks associated with respiratory diseases.
- Awareness raising of the risks associated with sexually transmitted infections, including HIV/AIDS. This can be a particular risk in cases where there is a large influx of migrant workers to an area. In addition, long-haul transport activities may serve as disease conduits.

Pastoral Activities

A fenced area of approximately 2 km^2 will be established around the mine site and a fenced area of approximately 1 km^2 will be established around the camp. Both areas are outside the koris, which host most of the area's natural vegetation and within the 36 km^2 buffer zone, access to which will not be restricted. However, local people will be discouraged from prolonged stays in this area (e.g. setting up camps), via a program of stakeholder consultation and awareness raising. The buffer zone avoids the koris, which tend to be used by local pastoralists and is not considered to represent a significant adverse impact to the Kel Tamashek (Tuareg) and will serve to limit their exposure to certain risks associated with proximal habitation to a mining operation such as noise, dust, gases, and vibration.

There will be no relocation of permanent communities, and traditional herding routes along the koris will not be affected. SOMIDA has consulted with area villages regarding the fenced and buffer zones and secured their acknowledgement of and approval of these zones. SOMIDA will endeavour to support pastoralists through initiatives such as the provision of livestock feed banks, the provision of training in agricultural techniques to maximize fodder crop yields and ways of harvesting and storing fodder, refurbishing, and maintaining watering points, and setting up a system to monitor impacts on pastoralists.

Landscape Quality

The Project site is not visible from the main settlements, but lighting may impact on the character of the area at night. SOMIDA will implement good practice including directing lights downwards or otherwise shielding them to prevent excessive illumination outside the working area; maintaining a tidy site to reduce disturbance of the visual quality of the landscape; planting tree screens around some facilities to reduce visibility; and selecting colors for buildings that blend with the landscape.

Local Traditions and Customs

Although the Project is expected to bring significant benefits to the local area in terms of direct and indirect employment opportunities and incomes in general, through stakeholder engagement SOMIDA is aware of concerns over local traditions and customs potentially being lost as a result of an incoming workforce and a potential switch to mining-related livelihoods.

This risk will be lowered by the Project having its own, self-contained accommodation camp located at distance from the local villages. SOMIDA has formulated a plan to raise awareness among staff and subcontractors about respecting traditional practices and customs of the local population. A Code of Conduct has also been drawn up to encourage respectful interaction with the local communities. Camp residents will be discouraged from entering local communities for recreational purposes.

The mine camp will be designed and maintained according to good international practice with the aim of preventing overcrowding and reducing the transmission of communicable respiratory diseases. The IFC/EBRD's document, Workers' Accommodation: Processes and Standards, is being used to guide camp development.

Economy

SOMIDA intends that the Dasa Project will bring significant benefits to the local economy, via the provision of direct and indirect employment opportunities. SOMIDA will prioritize local labour in recruitment, prioritize local companies in subcontracting, and enhance local procurement opportunities for the providers of local goods and services to the extent practical.

Benefits to local communities will also be affected through education and training, and the enhancement of health care. These initiatives are in addition to the benefits that will accrue to the local and regional population from the payment by SOMIDA of mining royalties and tax revenue, a portion of which will be returned to local and regional authorities.

There is potential for significant impact on the local economy and livelihoods when the mine finally closes, and those jobs and procurement activities cease. Therefore, the mine closure plan, which currently exists in conceptual form and will be developed as the Project progresses, will address social aspects of closure, in terms of direct workers, indirect livelihoods, and associated communities (those with a high proportion of workers or suppliers of goods and services). In particular, SOMIDA will devise a retrenchment program aimed at retraining workers in other occupations.

Throughout the Project lifetime, SOMIDA will provide support to traditional livelihoods and traditions as outlined above, with the aim that such activities are not lost as a consequence of the Project.

Archaeology and Cultural Heritage

There are two small graveyards located adjacent to the access track to the west of the exploration camp. In consultation with local communities, SOMIDA has agreed to fence these areas off for protection during Project operations. Access to the sites by local people will not be restricted.

No archaeological or cultural heritage sites have been identified as being at risk from Project activities, but it is recognized that as-yet undiscovered sites or artefacts may be present within the Project footprint. Therefore, a chance-finds procedure and awareness-raising thereof have been put in place. All staff and contractors will be trained to recognize objects or sites of interest that may be encountered and works proximal to any such suspected object or site being found will stop so that evaluation by the appropriate authorities may take place.

Environmental and Social Project Management

The mitigation measures outlined above will be incorporated into the ESMP for the Project. The ESMP is based upon the Environmental Permit (Cahier des Charges or CCES) issued by the Government of Niger as part of its approval of the 2020 ESIA.

In practice, the ESMP includes a suite of topic-specific documents and includes management plans deemed critical for implementing the mitigation measures outlined in the ESIA, and additional management plans required to align the Project with EP4, IFC PS, and GIIP. The management plans listed below have been developed for the construction phase of the Project and will be amended and additional management plans, policies and procedures added as necessary to carry the Project into the operational phase and through the closure phase:

- Occupational Health and Safety Plan.
- Radiation Management Plan (including worker protection).
- Community Health, Safety and Security Plan (including human rights, population influx, security, indigenous peoples).
- Human Resources Management Plan (including gender-based violence in the workplace, forced labour, child labour, etc.).
- Contractor Environmental Management Plan.
- Stakeholder Engagement Plan (including grievance mechanism).
- Progressive Restoration Plan.
- Water Management Plan.
- Air Quality and Greenhouse Gas Emissions Management Plan (including dust management).
- Noise and Vibration Management Plan.
- Biodiversity Management Plan (including invasive species management).
- Hazardous Materials Management Plan (including naturally occurring radioactive materials).
- Waste Management Plan.
- Tailings Management Plan (including geochemical considerations for waste rock and tailings).
- Emergency Preparedness and Response Plan (including spill prevention and management).
- Chance Finds Procedure (archaeology and cultural heritage).
- Environmental Monitoring Plan; and,
- Mine Closure Plan.

While the component management plans of the Project ESMP address technical topics, the role of the Environmental and Social Management System ("ESMS") is to provide the organizational framework necessary to ensure the successful implementation of the ESMP. The ESMS defines, amongst others, the company organizational structure, staff training provisions, communication networks, document control procedures, and systems for checking progress that are required for demonstrable achievement of the Project's aims for sustainable development.

The ESMS will aim to fulfil the IFC PS1 requirement to have “a dynamic and continuous process initiated and supported by management”, which “involves engagement between [SOMIDA], its workers, [and] local communities directly affected by the project”.

The ESMS will be modelled on the ISO 14001 Environmental Management Systems standard and its “Plan-Do-Check-Act” methodology that strives for continuous improvement. As required by IFC PS1, the ESMS will incorporate social and labour elements.

20.4. Community Engagement and Support

GAC and now SOMIDA has been engaging with local communities since its arrival in the area in 2008. In 2020, as part of the ESIA undertaken for the national permitting process, formal consultations took place in the communities around the Project area, including Tagaza, Agatara, Issakanan, Sikiret/Tadant, Oufound, Mizeine, Ghalab, the Kelezeret Tribe and Inolamane.

FEED Consult carried out additional engagement in the local villages as part of the ESIA conducted in 2022. The 2022 engagement also included the Governorate, the Regional Council, the Regional Director of Mines, the Regional Directorate for the Environment and the Fight against Desertification, the Regional Directorate for the Advancement of Women and Child Protection, the Regional Directorate of Hydraulics and Sanitation, the Regional Labour Inspectorate, and the Regional Directorate of Livestock. At Departmental level, the Town Hall, and the Prefecture as well as the villages listed above were consulted.

During the 2022 and earlier consultations, participants raised various environmental and social concerns regarding the Project, which the ESIAs have aimed to address.

Over the course of 2023, SOMIDA continued consultations with the parties listed above and expanded the geographical scope of consultations to include communities within a 30 km circle of the Dasa Project. SOMIDA also shares the results of its local consultation program and its wider social programs with government authorities in the urban centers of Agadez, Tchirozérine, Danet and Arlit and, regional and national Government Ministries.

The principal aim of SOMIDA’s consultation program is to keep local people informed as to the progress of the Project, encourage community involvement in the Project through local employment and subcontracting and provide a forum for concerns to be expressed.

GAC and now SOMIDA has been supporting local communities through various Community Social Relations ("CSR") programs since 2008, as summarized in the table below which also shows anticipated increased levels of support and new programs through the construction phase and mining operations. Support programs will be evaluated on an on-going basis through the operations and closure phases of the Project.

Table 20-5: Summary of Community Support Initiatives Undertaken and Planned

Global Atomic Corp - CSR / ESG	Exploration														Construction			Ops
	2008	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025
Food																		
millet			x	x	x	x	x	x	x	x	x	x	x	x	x		x	x
sugar			x	x	x	x	x	x	x	x	x	x	x	x	x	x	x	x
rice			x	x	x	x	x	x	x	x	x	x	x	x	x		x	x
Medical																		
ambulance										x								
supplies										x				x	x	x	x	x
food										x				x	x		x	x
Covid													x					
Infrastructure																		
roads								x	x	x	x	x	x	x	x	x	x	x
water well - local / herding								x		x		x		x		x	x	x
water well - Camp / community use												x			x	x	x	x
water well - agricultural supply												x			x		x	x
Environment																		
EISA and baseline studies / inventory		x	x										x	x	x	x	x	x
project area inventory													x	x	x	x	x	x
re-vegetation initiatives														x	x	x	x	x
mitigation programs																x	x	x
Education / Training																		
education - exploration			x	x	x	x	x	x	x	x	x	x	x	x	x	x	x	x
training - exploration			x	x	x	x	x	x	x	x	x	x	x	x	x	x	x	x
training - mining apprenticeship															x	x	x	x
traininig - environment														x	x	x	x	x
agriculture - training / support																x	x	x
Local Business Support / Procurement																		
agriculture														x	x	x	x	x
food services														x	x	x	x	x
micro business - community			x	x	x	x	x	x	x	x	x	x	x	x	x	x	x	x
camp supply				x	x	x	x	x	x	x	x	x	x	x	x	x	x	x
Regional / National procurement																		
exploration drilling	x	x	x	x	x	x	x	x			x	x	x	x	x		x	x
road work					x									x	x	x	x	x
camp site development / maintenance					x	x	x	x	x	x	x	x	x	x	x	x	x	x
food services			x	x	x	x	x	x	x	x	x	x	x	x	x	x	x	x
water wells install / maintain					x	x	x	x	x	x	x	x	x	x	x	x	x	x
camp security - regional / federal					x	x	x	x	x	x	x	x	x	x	x	x	x	x

Future development support will likely be delivered in partnership with non-governmental organisations currently active in-country and provide targeted benefits to women including enhanced irrigation, training and support of existing market gardening initiatives, support for development of goods and services related to workers apparel and PPE, and associated education, training, and mentoring programs.

One of the key achievements of 2023 is the success of training programs aimed at untrained youth and collaboration with Universities and Technical Colleges in Agadez and Niamey, the national capital. This initiative resulted in the hiring of 19 individuals in 2023 and will be expanded in 2024 to include collaboration with the Mining School of Agadez University.

Employment opportunities range from camp services, facilities management to equipment operator positions. The Project currently employs approximately 275 people. SOMIDA prioritizes local and regional hiring whenever practical. The workforce is expected to peak at 700 during construction and level out at 625 during mining operations plus about 200 contractors and therefore represents a significant opportunity for local and regional economic development. The SOMIDA workforce is currently 98% Nigerien and expected to remain so during the full life of the Project.

As the Project ramps up into commercial operations, corporate social responsibility contributions will be reviewed with reference to the success of projects to date, and priorities will be identified in consultation with communities via implementation of the stakeholder engagement plan.

20.5. Conclusion

The Dasa Project aligns with the strategic development aims of the Government of Niger, including the National Policy on the Environment and Sustainable Development, the National Environment Plan for Sustainable Development ("PNEDD"), the Sustainable Development and Inclusive Growth Strategy ("SDDCI Niger 2035"), the Economic and Social Development Plan ("PDES 2022-2026"), and the National Mining Policy adopted in 2020 and covering the period 2020-2029.

The Project has the potential to negatively impact elements of the biophysical and human environment of the area, as summarised above. However, a suite of mitigation measures has been defined to avoid or reduce these impacts and all are considered manageable. An ESMP has been implemented to ensure these measures are carried out through the full Project lifecycle. SOMIDA is committed to operating the DASA project according EP4 and IFC Performance Standards.

The Project will have particularly important positive impacts, including the creation of a significant number of direct and indirect jobs; the improvement of incomes; additional business opportunities for regional and local companies and subcontractors, and increased government revenues at the local, regional, and national levels through the payment of taxes and royalties.

The restoration of skilled, good-paying mining industry jobs lost due to the shut-down of the Cominak Mine in Arlit in March 2021 after nearly 40-years of operation, together with training programs focused on area youth, will enhance socio-economic stability, and provide the foundation for long term economic opportunity. The Project will also facilitate the delivery of a wider scope of community enhancement programs and increased investment in the area, which will result in improved infrastructure such as health clinics, access to water, schools, and transportation.

21. Capital and Operating Costs

21.1. Mining Capital Costs

Basis of Estimate

The capital mining cost estimate was determined by Bara Consulting. The capital cost estimates have been determined through the application of enquiry quotations, budget quotations, database costs and estimated costs to bills of quantities, material take offs and estimate quantities. Most of the capital cost is related to the mine design and mine plan, the quantities of which were computationally modelled and scheduled in three-dimensional space. Other costs relate to specific engineering designs, for which drawings have been produced and quantities have been generated from these drawings.

Definition of Capital Cost

Capital costs have been defined in terms of project capital cost and sustaining capital cost. Project capital cost include all capital costs from the initiation of the project to January 2026 this includes:

- The cost of all site preparation and surface infrastructure related to the mining infrastructure complex, including but not limited to the offices, change houses, workshops, and other surface facilities.
- The cost of all second egress facilities.
- The cost of mine development, power, mining equipment, underground facilities, and services up to January 2026.
- Contingency costs related to the above.

Sustaining capital includes:

- The cost of mine development, mining equipment, underground facilities, and services after the point at which uranium is first produced.
- Indirect and contingency costs related to the above.

Summary of Mine Capital Cost

A summary of the Capital Cost is presented in Table 21-20. The table presents the Project Capital, Sustaining Capital and Total Capital cost against the mining related areas of the project WBS.

Table 21-1: Summary of Mine Capital Cost.

Area	Project Capital Cost ($)	Sustaining Capital Cost ($)	Total ($)
Site Preparation	319 679	-	319 679
Surface Facilities	13 342 057	-	13 342 057
Emergency Exit & Ventilation holes	1 993 201	-	1 993 201
Surface Utilities and Reticulation	4 429 964	-	4 429 964
U/G Facilities and Services	3 736 232	3 926 233	7 652 666
U/G Utilities and Reticulation	1 900 765	12 595 270	14 496 035
Mining Equipment	13 114 714	78 226 996	91 341 710
Mine Development	14 115 502	76 296 692	90 412 195
Indirect Costs	2 056 968	1 321 720	3 378 688
Contingency	8 249 892	25 855 037	34 104 929
Capitalized Working Costs	14 328 991		14 328 991
Total	**77 578 165**	**198 221 949**	**275 800 114**

Site Preparation Capital

For capital costs related to site preparation, Bara predominantly used costs obtained by METC. METC has approached the market on enquiries for the scope of work which includes earth works, civil works, structural steel work as well as piping works relating to the processing plant. METC then selected the preferred suppliers / contractors, and the costs supplied by these selected suppliers / contractors were used to compile the costs for similar scope of work forming part of the site preparation. The cost of the equipment that will be installed at the general laydown area, were compiled from database costs.

Table 21-2: Site Preparation Capital Cost.

Site Preparation	Project Capital Cost ($)	Sustaining Capital Cost ($)	Total ($)
Access Roads	77 412	-	77 412
Bus Stop (Drop Off Zone)	5 175	-	5 175
Earthworks and Terracing	75 346	-	75 346
General Laydown Area	34 263	-	34 263
Parking Areas	6 530	-	6 530
Site Access Control and Fencing	120 953	-	120 953
Total	**319 679**	**-**	**319 679**

Surface Facilities Capital Cost

For capital costs related to surface facilities, Bara predominantly used costs obtained by METC who has approached the market on enquiries for the scope of work which includes earth works, civil works, structural steel work as well as piping works relating to the processing plant. METC then selected the preferred suppliers / contractors, and the costs supplied by these selected suppliers / contractors were used to compile the costs for similar scope of work forming part of the surface facilities. Budget quotes were received for prefabricated type buildings such as the change house and laundry building and the costs of the other steel structured buildings, such as the workshops, where factorised from costs received by METC on similar type buildings at the processing plant. The cost of the equipment that will be installed inside the different buildings were compiled from a mixture of budget quotes and database costs.

Table 21-3: Surface Facilities Capital Cost.

Surface Facilities	Project Capital Cost ($)	Sustaining Capital Cost ($)	Total ($)
Brake Test Ramp	23 874	-	23 874
Canteen and Restroom	55 891	-	55 891
Change house and Laundry	336 304	-	336 304
Crush	-	-	-
Emulsion Silos	220 534	-	220 534
Explosive Magazines	32 641	-	32 641
Helipad	-	-	-
Lamp Room	5 586 276	-	5 586 276
Lubrication and Diesel Refuelling Stations	453 802	-	453 802
Offices	947 742	-	947 742
Prayer Room	49 126	-	49 126
Stores	72 819	-	72 819
Workshops	5 563 047	-	5 563 047
Surface Facilities Total	**13 342 057**	**-**	**13 342 057**

Emergency Outlet Capital Cost

The capital cost included the cost of emergency exit and ventilation holes and is an estimated cost and includes a provision for lifting equipment required to install services and ladderways for the emergency exit only.

Table 21-4: Emergency Exit and Ventilation Holes Capital Cost.

Emergency Exit & Ventilation holes	Project Capital Cost ($)	Sustaining Capital Cost ($)	Total ($)
Headframe and Shaft Equipping	1 975 484	-	1 975 484
Site Preparation	17 716	-	17 716
Total	**1 993 201**	**-**	**1 993 201**

Underground Infrastructure and Reticulation Capital Cost

Underground infrastructure costs comprise all costs related to facilities, utilities and services required to support the mining operation. This includes facilities, such as pump stations, fan installations and dams. In addition, all compressed air, water handling and electrical infrastructure is included as utilities and reticulation. A detailed summary of underground costs relating infrastructure is presented in Table 21-5 and Table 21-6. Most of these costs was developed through cost estimation of engineering designs based on drawings to which a BoQ's and material take-offs were developed. The material take-offs were also developed by detailed evaluation of the mine development layout; with respect to lengths of cabling and piping required. Budget quotations, database costs and estimates were applied to these quantity take-offs and scheduled according to the mine plan.

Table 21-5: U/G Facilities and Services Capital Cost.

U/G Facilities and Services	Project Capital Cost ($)	Sustaining Capital Cost ($)	Total ($)
Cascade Dams	24 267	91 528	115 795
Charging Stations	-	-	-
Dewatering	913 268	3 430 026	4 343 294
Potable Water Tanks	-	-	-
Refuge Chambers	-	-	-
Service Water Booster Pump Station	-	185 768	185 768
Substations	-	-	-
Underground Satellite Workshop	107 485	-	107 485
Ventilation	2 620 689	-	2 620 689
Ventilation Raise Equipping	70 523	209 112	279 634
U/G Facilities and Services Total	**3 736 232**	**3 916 434**	**7 652 666**

Table 21-6: U/G Utilities and Reticulation Capital Cost.

U/G Utilities and Reticulation	Project Capital Cost ($)	Sustaining Capital Cost ($)	Total ($)
Backfill Reticulation	618 680	7 243 214	7 861 894
Compressed Air	47 050	177 461	224 512
Control and Instrumentation	253 409	955 790	1 209 199
Dewatering	353 330	1 332 664	1 685 994
Electrical Reticulation	589 859	2 224 788	2 814 647
Pipe Support	87 825	338 408	426 233
Potable Water	-	-	-
Service Water	57 329	216 229	273 558
U/G Utilities and Reticulation Total	**2 007 482**	**12 488 553**	**14 496 035**

Mining Equipment Capital Cost

Mining equipment capital cost comprises all costs related to purchase of equipment required to support the mining production and development operations. Ancillary mining equipment includes equipment such as small pumps, fans, temporary piping and cabling, mobile and general mining equipment. Most of this cost was developed through cost estimation of engineering designs based on forming a BoQ to which, budget quotations, database costs and estimates were applied.

Trackless mechanised equipment includes the cost of purchasing the mechanised equipment required for decline development, ore and waste trucking and men and material movement over the life of mine. The numbers presented in Table 21-7 includes initial equipment required for the operation and the replacement of that equipment once required.

Table 21-7: Mining Equipment Capital Cost.

Mining Equipment	Project Capital Cost ($)	Sustaining Capital Cost ($)	Total ($)
Ancillary Mining Equipment	3 300 262	472 719	3 772 981
Trackless Mining Equipment	10 230 939	77 337 791	87 568 730
Total	**13 531 201**	**77 810 510**	**91 341 710**

A summary of the capital cost of the mining fleet is shown in Table 21-8 below.

Table 21-8: Summary of Mining Fleet Capital Cost.

Item	Qty	Capital Cost ($)	Total ($)
Development drill rigs Twin boom	3	1 748 123	5 244 369
Bolter	2	1 483 200	2 966 400
Long Hole Drill Rig	2	1 600 000	3 200 000
LHDs (14 t)	3	1 391 040	4 173 120
ADTs (42 t)	5	1 260 616	6 303 080
Grader	1	390 000	390 000
Utility vehicles	3	240 000	720 000
Transmixer	1	270 000	270 000
Shotcreter	1	300 000	300 000
Telehandler	2	240 000	480 000
Pick-ups	6	80 000	480 000
Troop carrier	2	80 000	160 000
Grade control Rig	1	362 000	362 000
Service Holes Rig	1	2 394 630	2 394 630
Total Cost			**27 443 599**

Mine Development Capital Cost

Mine development capital cost comprises all the cost of the development of excavations such as the declines, ore drives, station connections, return air ways and crosscuts. The development costs comprise consumable (stores) costs, power costs and labour costs.

Table 21-9: Mine Development Capital Cost.

Mine Development	Project Capital Cost ($)	Sustaining Capital Cost ($)	Total ($)
Consumables	7 588 888	39 120 669	46 709 558
Power	4 416 477	42 840 351	47 256 828
Labour	3 809 342	10 725 966	14 535 309
TMM	5 225 500	30 256 507	35 482 008
Total	**21 040 208**	**122 943 494**	**143 983 702**

Other Capital Costs

Contingency has been included at 15% of the total mining capital cost and has been determined through consideration of the estimate accuracy. Total contingency equates to 33.6 million over the life of the Phase 1 mine.

Exclusions

The mining capital cost estimate does not make provision for any environmental or closure costs related to the infrastructure or mine plan presented in this report. No provisions have been allowed for escalation of any costs.

21.3 Mining Operating Costs

Definition of Operating Cost

Operating costs have been defined as the cost of all activities related to ore mining and production, and these include:

- Direct variable mining costs, including the cost of consumables (explosives, drilling consumables, fuel, etc.) for ore development and stoping activities.
- Direct fixed mining costs, including the cost for labour required for mine production, technical services, surface, and underground engineering.
- Power cost for all mining activities
- General administration costs including mine management labour.

Summary of Operating Cost

The operating cost estimate is presented in Table 21-10, below. The table presents the life of mine total and the unit operating cost per tonne milled and per pound of uranium before recovery, by activity or area.

Table 21-10: Summary of Mining Operating Cost

Operating Cost	Phase 1 Total $	Cost / t [$ / t milled]	Cost / lb [$ / lb]
Consumables	196 752 883	24	2.9
Power	226 718 207	28	3.3
Labour	68 259 916	8	1.0
TMM	124 236 870	15	1.8
Other / Contractors	4 260 000	1	0.1
Total	**620 227 876**	**77**	**9.1**

Mining Consumables

Mining consumable cost comprises all costs relating to drilling, blasting, support, backfill and general mining consumables for the mining of ore drives, slot raises and stope blocks. These costs were determined from first principles. Prices were obtained for the major consumables such as explosives, ground support products, drilling consumables, ventilation consumables, and cement from suppliers, either local in Niger or in South Africa. For the items with a South African cost base the cost of shipping the items to site in Niger was added.

Trackless Mechanised Machine (TMM) Cost

TMM consumable cost comprises all costs relating to fuel, oil, tyres, and parts required to service the mechanised mining equipment; specifically, to mine ore drives, slot raises and stope blocks. These costs were determined from first principles. The cost of operating the underground mining fleet was calculated based on predicted operating costs provided by the OEMs or from database sources. For the primary equipment the hours required to support the mining schedule were calculated and multiplied by the cost per hour to operate the machine. For the secondary fleet the hours per month were estimated and multiplied by the hourly operating cost.

Power Cost

Power costs were determined through application of a cost for power to the life of mine power consumption which was determined from first principles. Power costs were assumed to be $0.2595 per kWh as per the tariff provided by METC.

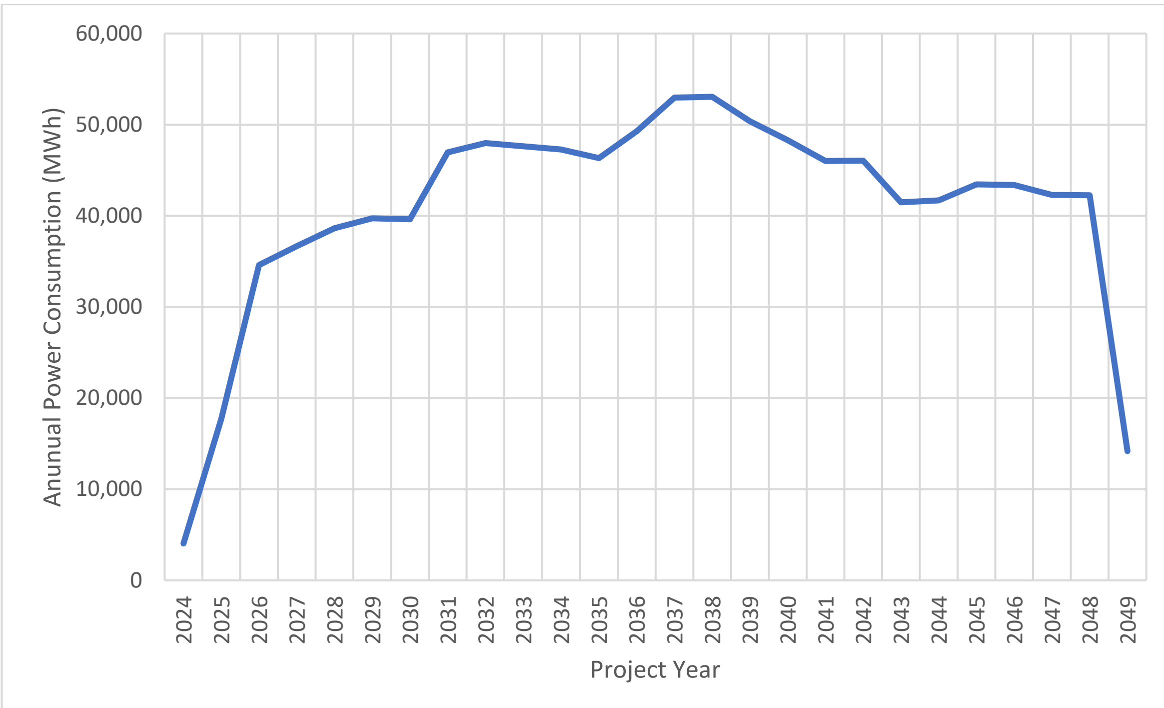

Figure 21-1: Dasa Phase 1 Mine Power Consumption.

Labour Cost

Labour costs, including management (G&A), direct mining labour, technical services and engineering was determined through application of a cost to company labour rate to a manpower/labour schedule which was determined for the life of mine from first principles.

Two separate cost structures have been used, one for expatriates and the other for Nationals, which are detailed in

The approach followed, is to limit the number of expatriates to the minimum, with the Mining Manager, Resident Engineer, and Mining Engineer / TSM posts being filled by expatriates and the remainder of the positions by Nationals as per Table 21-11 and Table 21-12.

Table 21-11: Cost Structure for Nationals.

Designation	Category	Monthly Salary ($)	Annual Salary ($)
Management	1	3 043	36 518
Middle Management	2	1 861	22 328
Foreman / Supervisor / Technician	3	1 861	22 328
Training / Safety	4	1 861	22 328
Artisans	5	1 408	16 899
Control Room Controller	6	1 005	12 059
Operator	7	640	7 675
Team Leader / Clerk	8	595	7 140
Attendant / Eng. Assistant	9	595	7 140

Table 21-12: Cost Structure for Expatriates.

Designation	Category	Monthly Salary ($)	Annual Salary ($)
Mining Manager	1	20 000	240 000
Mining Engineer / TSM	2	15 000	180 000
Mine Overseer	3	15 000	180 000
Production Engineer	4	15 000	180 000
Snr Training Officer	5	12 000	144 000
Planner	6	12 000	144 000
Foreman	7	12 000	144 000
Artisans	8	9 000	108 000

Other Operating Costs

Estimated allowances have been made with respect to other costs related to consumables for general and administrative purposes and outside contractors for mining and administrative activities.

21.4. Plant Capital Cost Estimate

The process plant capital cost estimate is based on the metallurgical test work and development of the PFD's. After the process plant layout was developed, the engineering designs were used to obtain equipment and installation prices from the market. The feasibility study mill, capital cost developed in 2021 has been updated by Professional Costs Consultants (PCC) based on their extensive project database developed over several decades within the industry, to base date of January 2024 rates. Infrastructure capital costs to support the process plant are based on current design and cost estimates, which differ somewhat from the 2021 study.

These steps are detailed below:

Test Work and Process Development

Samples of ore were sent to laboratories in Canada where typical physical and metallurgical tests were undertaken, including 3 pilot plant campaigns. The processes followed and the results obtained are detailed in sections 13 and 17 of this report.

3 trade-off studies were undertaken to determine the following:

- The most effective method of crushing and grinding of the ore using known industry methods to effectively liberate the uranium.
- Which one of three precipitation agents used in the industry could best satisfy the requirements to deliver the most appropriate finished product in terms of density, colour, and best recovery to meet market demands.
- The most cost effective and practical application to deliver tailing material from the plant to the tailing storage facility (TSF)

From the pilot plant campaigns and the test work, a Block Flow Diagram (BFD) was developed to determine an appropriate processing regime. The process was further refined by the development of Process Flow Diagrams (PFD's) and a Mechanical Equipment List (MEL) was developed, which provided typical equipment types, sizes, and performance requirements. The MEL also provided guidance on the electrical loads of the equipment to allow the engineering of a detailed electrical load list.

Plant Layout and Basic Engineering

The PFDs were used to develop a process plant layout to show the arrangement of the equipment to allow for a logical flow of materials (feed ore, waste, product, reagents, and utilities). From the layout a preliminary 3D model was developed to enable enhanced designs to be progressed which included equipment sizes (dimensions and loads), and this in turn lead to further development of the 3D model.

In the equipment cost determination process, a supply vendor list was developed between the procurement and engineering departments (with input from the client) and then issued for use. As the project developed this vendor list was added to, to widen the vendor participation in the enquiry process.

Mechanical Engineering

Suitable and appropriate mechanical equipment was selected to match the duties and requirements detailed in the mechanical equipment list, and for each piece of mechanical equipment, data sheets and specification documents were developed. These data sheets and specification documents were issued through METC Engineering's procurement department to the prospective vendors and suppliers in a process called request for quotations (RFQ's). RFQ's specified both the technical and commercial requirements, with the commercial section specifying a firm closing date, INCOTERMS, payment terms and schedules for completion by vendors. Bids received from the market were adjudicated both technically and commercially and after clarification was resolved on any issues, a final recommendation was made. These adjudications were sent to the client for review and ratification on the selection of an appropriate vendor and pricing for inclusion in the plant capital cost estimate.

Concurrent with the RFQ process and the gathering of equipment sizing and masses, the iterative process of development and refinement of the 3-D model continued.

Electrical Engineering

From the mechanical equipment list an electrical load list was compiled and this was further refined and populated as the mechanical equipment RFQ's were received back from the market. The electrical load list together with the developing plant layout was used to logically size and place motor control centres (MCC's) so that a medium voltage power distribution routing and network could be developed. This medium voltage distribution network was used to develop the Single Line Diagram (SLD) indicating distribution transformers and medium voltage switchgear. Load allocations were made for remote consumers e.g., the permanent camp, mining surface infrastructure, borehole well field, etc.

Once the electrical equipment was appropriately sized, data sheets and specifications documents were developed and, in a process, like that described above in the mechanical engineering section, RFQ's were issued to the market, adjudicated and recommendations made for vendor selection and pricing.

From the layout (including positioning of the MCC's), cable routing was plotted from the furthest corner of the MCC to the furthest corner of the process area fed from the MCC to determine cable lengths. The number, size, and length of each cable was consolidated into a detailed Bill of Quantities (BOQ), together with earthing, small power and lighting, light fittings, field stations and all ancillary electrical equipment for incorporation into the Electrical, Control, and Instrumentation (E, C&I) installation RFQ.

Control and Instrumentation

Valves (manual and actuated) together with all instrumentation requirements (quantity, size, and type) were derived from the intelligent Piping and Instrumentation design software used for the project. Pricing for the valves and instrumentation was obtained from a database recently updated through the formal RFQ process and the instrumentation cost estimate from a supply perspective was developed. Typical hook up, loop diagrams and Input/Output schematics were developed and populated to match the control philosophy and logic developed by the process department.

In a similar process to that followed for the electrical discipline, typical routing of instrumentation and control cabling and fibre optic cable connections were determined and used to populate the installation RFQ's.

Structural Steel and Structural Concrete Designs

As the mechanical RFQ process advanced and selected vendors provided indicative dimensions and loads for their equipment, the 3D model was continuously being advanced to incorporate this information. As individual structures and or areas of the plant were advanced to a point where further changes were unlikely, extracts of the model were issued for preliminary structural steel and concrete designs.

Concurrent with the early process plant layout and selection of a suitable tailing storage facility (TSF), a geotechnical investigation was undertaken by an approved local company. The investigation included excavation pits and bore hole drilling according to a grid determined by the engineering teams that had identified specific structures and areas of the plant for geotechnical investigation.

Using the model and the Geotechnical report, the structural teams produced preliminary designs and developed detailed Bills of Quantities for all concrete works and all structural steel designs.

Earthworks Designs

Once the process equipment layout was nearing finalisation, the secondary services (infrastructure to support the process plant) were developed. This included administration buildings, workshops and stores, water ponds, access roads, etc. Following review and approval of the "complete" layout, it was issued to the infrastructure designer and using 3D modelling software, terrace sizing and volumes were modelled, and quantities determined based on the geotechnical requirements.

The earthworks and concrete works were consolidated into a single RFQ package and issued to the market with the supporting specifications, bills of quantities and commercial terms.

Buildings and Camp

It was determined that once the process plant becomes operational, all staff would be accommodated and fed by the Company at an on-site facility. All staff will be transported between the mine and their hometown on a varying rotation cycle, dependent on their roles and responsibilities at the Dasa Project site.

Architectural drawings for accommodation units and related facilities such as a recreation centre and canteen were developed, together with all other infrastructure buildings such as administration building, canteen and training facility, laboratory, change house and ablution facilities, guard houses, security and access control rooms, weighbridge control room, etc. - The building construction methodology is based on a combination of pre-engineered, concrete, and brick-and-mortar structures as most appropriate for each building. Costs are based on a combination of contracted costs and costs built up from first principles, dependent on the present stage of procurement.

Process plant and mining operational manpower requirements were estimated with an expected mix of both expatriate and local labour. Process plant construction and mining development teams were determined following the development of a construction execution schedule. Using known market related metrics for construction and mine development, construction labour histograms were developed and used to quantify the expected numbers of construction and development staff on site. The construction histograms, operational readiness histograms, and steady state operational histograms provided guidance for the required accommodation and infrastructure buildings.

Construction and Erection Contracts

The market was investigated for suitable and appropriate construction and installation contractors to execute the construction work in Niger. Contractors were identified from Europe, West Africa, and South Africa. Installation and construction RFQ’s for the various disciplines was compiled and distributed for internal (and client) review, with any additions and alterations made before the RFQ’s were issued to the market.

The construction packages were managed in a similar process of issuing, adjudication and recommendation as detailed above for the equipment purchases.

Earthworks and Concrete Works

The quantities related to earthworks (terracing, roads, ponds, and dams) were developed through the 3D modelling software. These quantities together with the appropriate designs and specifications were issued to the vendors and contractors on the vendor/contractor list via the formal RFQ process. The bids received were reviewed, clarified, adjudicated and recommendations made for the selection of a suitable contractor.

Structural Steel, Mechanical installation, Platework and Piping (SMPP)

The preliminary structural steel designs together with platework and tanks designs were consolidated into a detailed bill of quantities. The piping and instrumentation diagrams (P&ID's), together with the layout drawing were used to determine a routing and pipe length for each pipe represented on the P&ID's. Factors were applied to each pipeline to determine the quantity of fittings (elbows, Tee's, reducers, flanges, gaskets, etc.). A detailed piping bill of quantities, indicating pipe specification, lengths, fittings, etc. was compiled for incorporation into the relevant RFQ. An SMPP RFQ package was compiled and issued to the market. This RFQ package included the supply of structural steelwork, platework and piping with the mechanical equipment to be issued to the selected contractor for installation.

The SMPP bids received were reviewed, clarified, adjudicated and recommendations made for the selection of a suitable and appropriate installation contractor with the necessary capabilities to fabricate structural steel, platework and piping as defined in the RFQ.

Electrical, Control and Instrumentation (E, C&I)

Concurrent with the SMPP package an RFQ for the onsite E, C&I installation and supply of selected materials was developed and issued to the market.

The E, C&I bids received were reviewed, clarified, adjudicated and recommendations made for a suitable and appropriate installation contractor.

Logistics

The total logistics costs were derived from several sources:

The SMPP RFQ included the delivery to site of all the materials to be supplied by the contractor as detailed in the RFQ (this excluded the free issue mechanical equipment) and hence these materials were excluded from the masses and volumes provided to the logistics team to quantify the logistic costs associated with the project. The RFQ's issued to the market for equipment supply pricing, included schedules for the suppliers to complete, indicating the equipment masses and volumes. These RFQ's requested a cost ex-works and a supplier determined transport cost to the port of Cotonou in Benin – Cotonou was selected as it is the most commonly used seaport for the delivery of all goods into Niger.

The equipment supply mass and volume schedules were consolidated into totals for the movement of break bulk, abnormal loads, general purpose 20-foot and 40-foot containers, open top containers, etc. to form a typical categorisation matrix of goods and their associated masses and volumes. Logistics service providers based in Niger and neighbouring countries were identified and requested to express an interest to provide the logistical services for the project.

The logistical categorisation matrix was included in a formal RFQ process, and bids received were adjudicated and recommendations made.

Tailing Storage Facility (TSF)

The total CAPEX associated with the development of the four Dry Stack Tailing Storage Facilities (DSTSF's) required over the 24-years of mining has been determined to $28.94 million including contingency with an accuracy of ±35% based on quantities measured by Epoch and rates obtained from De Simone Ltd in 2021, as provided by METC, and escalated by 26%.

The selection of a suitable site for the first TSF (TSF 1) was done, taking into consideration:

- The method of deposition.
- The route to deliver the tailing to the DSTSF.
- The prevailing wind direction, and the potential impact of dust from the DSTSF impacting on the process plant and surface infrastructure (including the mine surface infrastructure).
- Topographical anomalies and
- Proximity to flood lines.

From the site selection process, a geotechnical investigation was undertaken for DSTSF 1, and a report generated. This geotechnical report together with a geotechnical and chemical analysis of the tailing product enabled a design of the starter walls and selection of a suitable lining material to be done. These designs included the drains and other relevant design considerations to meet industry best practices.

DSTSF 1, providing a 6-year tailings storage solution will be constructed progressively over the life of facility. In total, three compartments and two tiers will be constructed with the capital cost reflecting the work done to build starter walls, drains, and install liner material. Compartment one's starter walls, drains and liner will need to be completed prior to the process plant becoming operational. The costs associated with this work ($3,192 million including contingency) has been considered in the initial capital costs of the project. The tailing product from the process plant will be deposited and compacted within the starter walls until the design height of the first tier is achieved. Before compartment one is full, the starter walls, drains and liner for the second compartment will be constructed and sections of compartment one will be capped and the process of closure of DSTSF 1 will begin. Compartment three will follow. Once all three compartments are constructed to the elevation of the first tier, the entire facility will be lifted by a second tier. Once final elevation of the second tier is achieved, the entire DSTSF 1 will be closed and capped in terms of the environmental requirements.

The development of the second DSTSF (DSTSF 2) will commence prior to the completion of the second tier of DSTSF 1. The phased development of DSTSF 2 will follow the same methodology as that of TSF 1, followed by a third DSTSF (DSTSF 3) and finally the fourth DSTSF (DSTSF 4). The construction of compartments two and three of DSTSF 1, along with the construction of DSTSF 2, DSTSF 3 and DSTSF 2, together with the capping and closure of all 4 facilities are included in the sustaining capital cost as reflected in the financial analysis.

It must be noted that the Life of Mine CAPEX estimation of the dray stack tailing's storage solution for the mine, assumed four identical DSTSF's would be constructed. No site selection was undertaken for the additional 3 facilities and as such no site-specific requirements were considered for DSTSF 2, DSTSF 3 and DSTSF 4 in the CAPEX estimations.

Grid Power and Electrical Supply

From the plant mechanical equipment list and electrical load list the total plant demand was determined (this included the accommodation camp and other surface infrastructure). The electrical demand for the underground mine and the mining surface infrastructure was determined from the mine design and associated requirements for pumping, ventilation, and general services. The combined power requirements and load profile for the Dasa Project and load profile was determined to be 12.2 MW.

Safety critical and operationally critical processes and equipment were identified to determine the emergency standby diesel generator requirements – these loads are included in the total plant demand.

A preliminary grid power connection and design, incorporating the relevant transformers, switchgear, and power factor correction together with the specified voltage and design was issued to the market to obtain a capital cost estimate for a grid power connection for the switch yard and the 132 kV overhead powerline.

Owners Costs

The client provided a listing of the expected costs to be incurred prior to commercial product. These costs included:

- Client project team.
- Camp and catering costs for the full construction period for 700 people.
- Project insurance.
- Operational readiness.
- Fuel for vehicles for the owners and EPCM team on site.
- Commissioning services, and First Fills.
- Indirect site and Niamey costs as well as mine operating costs until mill commissioning at the end of 2025.

EPCM Cost

METC completed a detailed manhour estimate from first principles for the head office tasks of engineering and procurement. An execution schedule was also developed and issued with the installation RFQ’s. The recommended contractor’s manpower and plant histograms confirmed the execution schedule durations and subsequently a construction management histogram was developed. A construction team rotation roster was developed and used to determine flights, visas and other EPCM incidental costs.

Preliminary and General Costs (P&G’s)

For the installation contracts and bids received from the market for the various disciplines, the contractors provided various schedules related to plant and equipment, manpower and associated costs, the cost of offices and workshop facilities on site, site establishment and dis establishment costs, etc. These costs were consolidated and used in the adjudication process to select a preferred vendor.

Using the P&G's costs from the various disciplines for the onsite construction / installation work a consolidated P&G's cost was established and was shown as a unique line in the capital cost estimate.

Spares

All equipment supply, RFQ's incorporated a schedule to be completed by the vendors / suppliers related to spares. The spares were separated into capital / strategic spares, commissioning spares and 2-year operational spares. The adjudicated and recommend vendor / suppliers recommended spares list was reviewed and assessed by the engineering team. The engineering team took cognisance of the criticality of equipment, the remoteness of site from supply centres and uniqueness of the selected equipment to determine (on a line-by-line basis), which spares to include in the consolidated list for inclusion in the capital cost estimate.

Mobile Equipment

A mobile equipment list was developed, based on knowledge of the process, the plant layout and input from team members with operational experience in minerals processing plants. Consideration was made of the materials handling requirements, tramming distances, installation of permanent lifting facilities over critical equipment and processes, etc. to determine sizes, reach and capacity of the required equipment. The list was reviewed extensively within the engineering team and subsequently comment, and input was obtained from the client's team. A mobile equipment vendor list was generated, and vendors were selected on their ability to support and service the equipment at what is considered a relatively remote site. The selected equipment ratings and sizes were issued to suppliers / vendors for pricing and the adjudication process followed. A policy of one vendor providing all equipment was considered, however the varying nature of the equipment necessitated that equipment could not be competitively sourced from a single vendor and consequently 6 vendors were ultimately selected.

Contingency

An internal risk assessment was undertaken for each procurement package in the capital cost estimate and an assessment was made in terms of:

- Uniqueness of the process.
- The complexity of the specific equipment.
- The uniqueness of the process requirements.
- The scarcity/rareness of experience of the equipment selected.
- The number of vendors that bid on the equipment.
- The number of compliant bids received.
- The price variance for the compliant bids
- The perceived experience and knowledge of the vendor selected.
- Thoroughness of the bid received.

These were weighted and scored in terms of a variance to the upside of the prices obtained, where a top score received a 5%, a good score 7.5%, an acceptable score 10%, a weak/low score of 15% and a price perceived to have a low level of confidence, a score of 20%.

The likelihood of rate of exchange variances and / or effects of global supply and inflation related increases or decreases was excluded from the ratings applied in the process of determining a contingency.

For the construction / installation contracts, in addition to the criteria used for supply packages, the following criteria were considered:

- Knowledge of, and previous experience in operating a remote site in a potentially hostile environment as demonstrated from the contractor's project reference list.
- The contractors home base relative to the project.
- Competence of the contractor as demonstrated through their completion of schedules, plant and labour histograms and obvious knowledge (or lack thereof) of executing the work as detailed in the BoQ's.
- The ratio of P&G costs to the total installation costs.

Each procurement package and each installation cost in the capital cost estimate was rated and a contingency applied.

Capital Cost Consolidation

The processes described above (items 21.3.1 to 21.3.17) enabled the team to develop detailed capital cost estimates for each discipline and activity / function associated with building a capable and efficient process plant. Costs were separated for Supply and Installation costs per discipline and costs associated with other services required to operate a process plant were grouped under Other Costs. As described in 21.3.18 Contingency, a contingency was applied to the various capital cost line items. The capital costs and contingencies are shown in Table 21-13 below. Table 21-14 below shows the costs incurred to date (Sunk Costs) and Table 21-13 below shows the remaining Feasibility Study Costs to Spend based dated January 2024.

Table 21-13: Process Plant Capital Cost Summary.

Description	Cost ($)	Contingency (%)	Contingency Amount ($)	Total Value (Carried through) ($)
Mechanical	31,275,690	6.2%	1,938,409	33,214,099
Structural Steel	5,360,021	7.5%	402,002	5,762,022
Plate Work	2,417,835	7.5%	181,338	2,599,173
Piping	4,782,210	7.5%	358,666	5,140,876
Electrical	6,374,639	8.6%	550,807	6,925,446
Instrumentation	1,644,089	16.8%	276,341	1,920,430
Valves Logistics	7,652,814	7.5%	573,961	8,226,775
Sub-total Supply	**60,007,049**	**7.2%**	**4,349,989**	**64,357,038**
Erection/Installation				
SMPP	4,025,276	9.3%	374,351	4,399,627
Electrical	775,941	10.0%	77,594	853,535
Instrumentation	388,676	10.0%	38,868	427,544
Earthworks (Supply & Install)	3,546,755	7.5%	266,007	3,812,761
Civils (Supply & Install)	8,309,609	7.5%	623,221	8,932,829
P&G's	5,928,426	25.0%	1,482,106	7,410,532
Additional General Contingency			6,000,000	6,000,000
Sub Total - Install	**22,974,683**	**38.6%**	**8,862,146**	**31,836,829**
Plant Total	82,981,732	15.9%	13,212,135	96,193,867
Owners Costs	64,559,165	3.6%	2,335,505	66,894,670
EPCM	16,173,991	7.5%	1,214,697	17,388,688
Mining	90,366,188	8.8%	7,941,347	98,307,535
Spares	2,063,405	6.2%	127,931	2,191,336
DSTSF (Incl. P&G's)	2,970,000	7.5%	222,750	3,192,750

Surface Mobile Plant	5,565,150	2.7%	150,000	5,715,150
Infrastructure	73,721,378	7.2%	6,687,183	80,408,561
Additional General Contingency			5,304,766	5,304,766
Sub Total	**255,419,277**	**9.4%**	**23,984,180**	**279,403,457**
Total	**338,401,009**	**11.0%**	**37,196,314**	**375,597,323**
Other Costs				
Working Capital	27,800,000			27,800,000
FS Project Cost Total	**366,201,009**	**10.2%**	**37,196,314**	**403,397,323**

Table 21-14: Project Costs Incurred to Date (Sunk Costs).

Description	Amount ($)
Mechanical equipment	2,287,890
Earthworks	446,125
Owners' costs	17,849,059
EPCM	4,027,018
Mining	31,529,816
Surface Mobile	4,240,405
Infrastructure	6,849,548
Total	**67,229,861**

The remaining feasibility study expenditure available to execute the project is presented in Table 21-15 below:

Table 21-15: Feasibility Study Remaining Costs to Spend (January 2024).

Description	Cost ($)	Contingency (%)	Contingency Amount ($)	Total Value (Carried through) ($)
Mechanical	28,987,800	6.7%	1,938,409	30,926,209
Structural Steel	5,360,021	7.5%	402,002	5,762,022
Plate Work	2,417,835	7.5%	181,338	2,599,173
Piping	4,782,210	7.5%	358,666	5,140,876
Electrical	6,374,639	8.6%	550,807	6,925,446
Instrumentation	1,644,089	16.8%	276,341	1,920,430
Valves	499,751	13.7%	68,466	568,217
Logistics	7,652,814	7.5%	573,961	8,226,775
Sub-total Supply	57,719,159	7.5%	4,349,989	62,069,148
Erection/Installation				
SMPP	4,025,276	9.3%	374,351	4,399,627
Electrical	775,941	10.0%	77,594	853,535
Instrumentation	388,676	10.0%	38,868	427,544
Earthworks (Supply & Install)	3,100,630	8.6%	266,007	3,366,636
Civils (Supply & Install)	8,309,609	7.5%	623,221	8,932,829
P&G's	5,928,426	25.0%	1,482,106	7,410,532
Additional General Contingency			6,000,000	6,000,000
Sub Total - Install	**22,528,558**	**39.3%**	**8,862,146**	**31,390,704**
Plant Total	**80,247,717**	**16.5%**	**13,212,135**	**93,459,852**
Owners Costs	46,710,106	5.0%	2,335,505	49,045,611
EPCM	12,146,973	10.0%	1,214,697	13,361,670
Mining	58,836,372	13.5%	7,941,347	66,777,719
Spares	2,063,405	6.2%	127,931	2,191,336

DSTSF (Incl. P&G's)	2,970,000	7.5%	222,750	3,192,750
Surface Mobile Plant	1,324,745	11.3%	150,000	1,474,745
Infrastructure	66,871,830	10.0%	6,687,183	73,559,013
Additional General Contingency	-	-	5,304,766	5,304,766
Sub Total	**190,923,431**	**12.6%**	**23,984,180**	**214,907,611**
Total	**271,171,148**	**13.7%**	**37,196,314**	**308,367,462**
Other Costs				
Working Capital	27,800,000			27,800,000
FS Project Cost Total	**298,971,148**	**12.4%**	**37,196,314**	**336,167,462**

Currency Exchange Rates

This capital cost estimate is provided in US$. The capital cost estimate has exposure to various currencies. The project accepted a table of fixed rates of exchange on 31 December 2023, with the principal currencies and exchange rates shown in Table 21-16 Exchange Rates, below.

Any variations in these exchange rates will require adjustment to the final (US$) estimate total.

Table 21-16: Exchange Rates.

Exchange	Rate
ZAR to USD	ZAR 18.34 = USD 1.00
EUR to USD	EUR 0.91 = USD 1.00
AUD to USD	AUD 1.51 = USD 1.00
CAD to USD	CAD 1.35 = USD 1.00
CFA (West African Franc) to USD	CFA 596.96 = USD 1.00
INR (Indian Rupees) to USD	INR 82.76 = USD 1.00

21.5. Mining Operating Costs

Definition of Operating Cost

Operating costs have been defined as the cost of all activities related to ore mining and production, and these include:

- Direct variable mining costs, including the cost of consumables (explosives, drilling consumables, fuel, etc.) for ore development and stoping activities.
- Direct fixed mining costs, including the cost for labour required for mine production, technical services, surface, and underground engineering.
- Power cost for all mining activities.
- General administration costs including mine management labour.

Summary of Mining Operating Cost

The operating cost estimate is presented in Table 21-17, below. The table presents the Life of Mine total and the unit operating cost per tonne milled and per pound of uranium before recovery, by activity or area.

Table 21-17: Summary of Mining Operating Cost.

Operating Cost	Total $	Cost / t [$ / t milled]	Cost / lb [$ / lb]
Mining	594,898,402	73.93	8.73
Consumables	188,117,731	23.38	2.76
TMM Consumables	121,892,945	15.15	1.79
Labour	55,284,894	6.87	0.81
Power	228,182,832	28.36	3.35
Other / Contractors	1,420,000	0.18	0.02
General and Administration	25,329,474	3.15	0.37
Consumables	3,968,187	0.49	0.06
Labour	18,521,287	2.30	0.27
Other / Contractors	2,840,000	0.35	0.04
Total	**620,227,876**	**77.08**	**9.10**

Note 1: Power Costs Include Pumping Costs Related to Surface Ponds.

Note 2: Milled Tonnage Excludes Inferred Material.

Mining Consumables

Mining consumable cost comprises all costs relating to drilling, blasting, support, backfill and general mining consumables for the mining of ore drives, slot raises and stope blocks. These costs were determined from first principles. Prices were obtained for the major consumables such as explosives, ground support products, drilling consumables, ventilation consumables, and cement from suppliers, either local in Niger or in South Africa. For the items with a South African cost base the cost of shipping the items to site in Niger was added.

Trackless Mechanised Machine (TMM) Cost

TMM consumable cost comprises all costs relating to fuel, oil, tyres, and parts required to service the mechanised mining equipment; specifically, to mine ore drives, slot raises and stope blocks. These costs were determined from first principles. The cost of operating the underground mining fleet was calculated based on predicted operating costs provided by the OEMs or from database sources. For the primary equipment the hours required to support the mining schedule were calculated and multiplied by the cost per hour to operate the machine. For the secondary fleet the hours per month were estimated and multiplied by the hourly operating cost.

Power Cost

Power costs were determined through application of a cost for power to the life of mine power consumption which was determined from first principles. Power costs were assumed to be $0.2595 per kWh as per the tariff provided by METC.

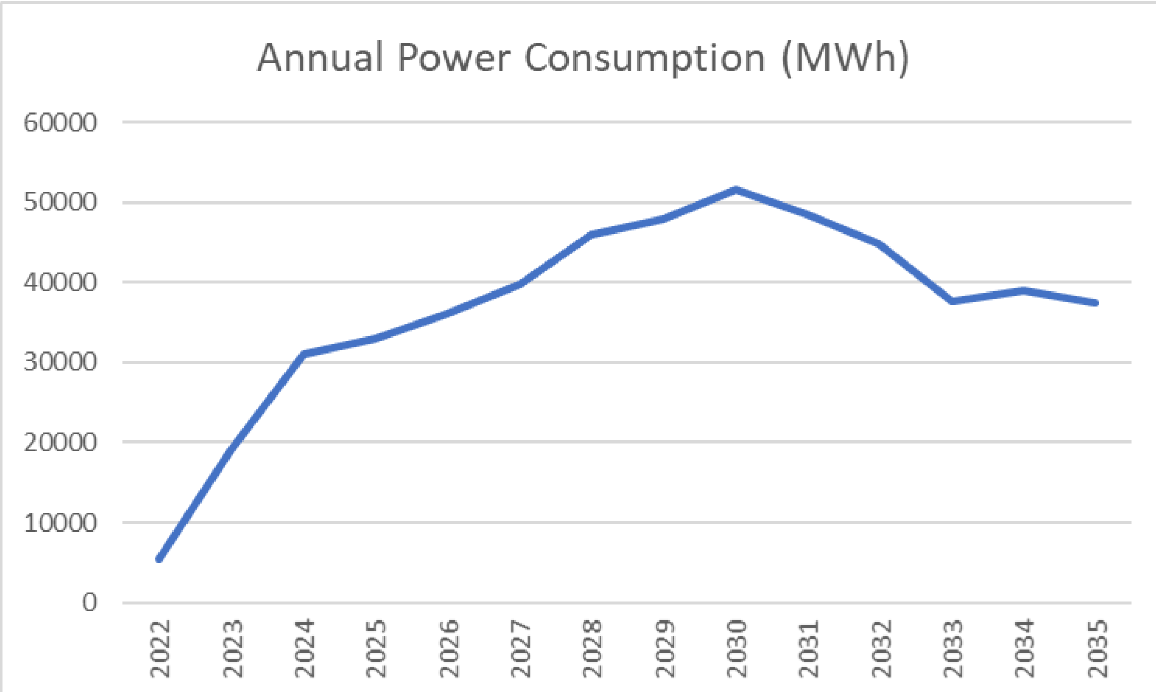

Figure 21-2: Dasa Phase 1 Mine Power Consumption.

Labour Cost

Labour costs, including management (G&A), direct mining labour, technical services and engineering was determined through application of a cost to company labour rate to a manpower/labour schedule which was determined for the life of mine from first principles.

Two separate cost structures have been used, one for expatriates and the other for Nationals, which are detailed in Table 21-18 and Table 21-19. The approach followed, is to limit the number of expatriates to the minimum, with the Mining Manager, Resident Engineer, and Mining Engineer / TSM posts being filled by expatriates and the remainder of the positions by Nationals.

Table 21-18: Cost Structure for Nationals.

Designation	Category	Monthly Salary ($)	Annual Salary ($)
Management	1	2 641.00	31,692.00
Middle Management	2	2 000.00	24 000.00
Foreman / Supervisor / Technician	3	1,750.00	21,000.00
Training / Safety	4	1,500.00	18,000.00

Artisans	5	1,200.00	14,400.00
Control Room Controller	6	900.00	10,800.00
Operator	7	700.00	8,400.00
Team Leader / Clerk	8	605.00	7,260.00
Attendant / Eng. Assistant	9	550.00	6,60000

Table 21-19: Cost Structure for Expatriates.

Designation	Category	Monthly Salary ($)	Annual Salary ($)
Mining Manager	1	15 000.00	180 000.00
Mining Engineer / TSM	2	12 500.00	150 000.00
Mine Overseer	3	9 000.00	108 000.00
Production Engineer	4	15 000.00	180 000.00
Snr Training Officer	5	5 830.00	69 960.00
Planner	6	5 830.00	69 960.00
Foreman	7	5 830.00	69 960.00
Artisans	8	5 830.00	69 960.00

Other Operating Costs

Estimated allowances have been made with respect to other costs related to consumables for general and administrative purposes and outside contractors for mining and administrative activities.

21.6. Processing Plant Operational Costs

The processing plant operating costs (OPEX) were derived from updated quotations received from various vendors. The Cash flow is based on the plant feed tonnage and grade profiles. In addition to the variable unit rates and quantities a fixed cost allowance has been provisioned in the processing plant operational costs.

Common to all operating cost estimates are the following assumptions are as follows:

- The annual power costs were calculated using 70% utilization and a unit price of US$ 0.2595/kWh.
- Labour is assumed to come mostly from within Niger, and salaries are based and benchmarked against similar uranium operations and projects.
- Equipment and materials will be newly purchased.

- Process plant operating costs are calculated based on labour, power consumption, and process and maintenance consumables.
- Grinding media consumption rates have been estimated based on the abrasion index.
- Reagent consumption rates have been estimated based on the metallurgical test work.
- Mobile equipment cost provides for fuel and maintenance.
- Operating costs have been updated to reflect current (2024) pricing. Certain items, specifically Crusher liners, Laboratory costs, and screen panels have not been updated and reflect the 2021 costs.
- Where current pricing was not available, the 2021 rate has been retained.
- Filter cloths have been updated per the 2021 costs +10%.

Exchange rates applied to the OPEX update are as follows:

USD/ZAR	18.34
USD/FCFA	596.96
EUR/USD	1.10

21.7. Basis of Process Plant Operating Cost

The basis of the plant operating costs is based on the following:

Power

The power for the project will be sourced from a neighbouring state-owned coal fired power station as well as solar plant constructed on-site as part of the project infrastructure. The power cost is calculated from the overall plant power draw, which has been determined from the mechanical equipment list developed by METC during the feasibility with the assumption that the power draw is 70% of the installed power, and the power cost has been calculated based on a per kilowatt hour cost of US$ 0.2595/kWh. The delivered power cost was calculated as a combined cost of supplies from the state-owned power station and the solar power plant. The power cost of US$ 0.2595/kWh has increased from US$ 17.25/kWh as reported in the feasibility study in 2021, based on the OPEX calculation power accounts for 31% of the total processing plant OPEX.

Reagents

The reagent profile was developed from the test work as indicated in Section 13 of the report. The test work enabled the addition and consumption rates of reagents to be determined. Where test work was not available, benchmarking against currently operating unit technologies was reviewed and applied to the reagent OPEX calculation. The reagent supply costs were obtained from vendors as delivered costs to the ports of either Tema or Lomé. Freight costs from the Port to the Dasa Plant site were obtained based on estimates from Freight Companies specialising in transport and logistics within the region. The cost of freight within the region is subject to fluctuation due to the disruption of major shipping routes within the region.

The cost of reagents accounts for 31.5% of the overall processing plant total OPEX. Where practicable, updated reagent costs have been obtained from reagent suppliers and utilised in the updating of the processing plant OPEX.

The following summarizes the status of the reagent and transport rates that have been applied to the updating of this phase of the study.

Table 21-20: Project Reagent Rates.

Reagent / Commodity	Units	Applied Rate ($/Unit)	Transport Rate (Lomé to Site) ($/Unit)	Comments / Assumptions
Ball Mill Grinding Media	$/tonne	1 237	515.45	Updated per Magotteaux Quote
Crusher Liners	$/a	35 235	515.45	Not Updated (Metso)
Mill liners ($/t)	$/liner set	256 586	515.45	Updated per Magotteaux Quote
Sodium Nitrate	$/tonne	750	515.45	Updated per Nowata
Hydrogen Peroxide (60% concentrate)	$/tonne	650	515.45	Updated per Nowata
Sodium Hydroxide - Pearls (99%)	$/tonne	750	515.45	Updated per Nowata
Sodium Carbonate (99%)	$/tonne	400	515.45	Updated per Nowata
Hydrated Lime ($Ca(OH)_2$) (92%)	$/tonne	695	515.45	Updated per Nowata
Flocculant - FO4490	$/tonne	4250	515.45	Updated Per Chemquest
Coagulant - DB54SH	$/tonne	5 950	515.45	Updated Per Chemquest
SX Diluent - Exxsol D80 (Shellsol D70)	$/tonne	3100	259.31	Updated per Nowata
SX Extractant - Alamine 336	$/tonne	15 000	515.45	Not Updated. BASF indicated the current rate is still valid
SX Modifier - Exxal 10™	$/tonne	5 989	515.45	Updated Per Chemquest
*Sulphur	$/tonne	162	160.00	Updated with price from Rein
Anthracite	$/tonne	1 481	367.35	Updated Per Chemquest
Garnet	$/tonne	1 681	367.35	Updated Per Chemquest
Bentonite	$/tonne	1 238	367.35	Updated Per Chemquest
Belt Filter Cloths	$/a	155 500	515.45	Updated (Added 10%)
Diesel	$/Litre	1.00	0.00	Updated per Power trade-off study. Trans. Included
Sulphuric Acid (98%)	$/tonne	490	315.50	Updated per Nowata

Reagent / Commodity	Units	Applied Rate ($/Unit)	Transport Rate (Lomé to Site) ($/Unit)	Comments / Assumptions
Binder – Ordinary Portland Cement	$/tonne	0		Not Updated
**Barium Chloride	$/tonne	1 350	515.45	Updated per Nowata
Packaging Drums	$/drum	82	129.65	Not updated
Silica Sand	$/tonne	645	515.45	Not updated

**The original price quoted for Sulphur in the Feasibility Study was based on a US$ 220/t, the Sulphur price has been updated based on the rates as provided by DASA and has been recalculated using a rate of US$ 162/t which includes an estimation of the ocean freight to Lomé as quoted by the Client and supported by the indicative quotation as received from Protea Chemicals.*

METC has included quote from Protea Chemicals for US$ 102/T FoB China. The ocean freight rate applied is based on a rate of US$ 60/t the Fob supply rate of US$ 162/ton CFR Lomé.

*** Barium Chloride consumption has not been specified and although a rate has been obtained, it does not contribute to the reagent cost(s).*

Transport costs used for the 2024 OPEX has been increased by 62% compared to 2021 i.e., US$ 515/t vs. US$ 316/ton as per the 2021 feasibility study. This is primarily attributed to the increase in shipping container rates. The ports of delivery assumed for this exercise has been based on either Lomé or Tema versus the port Cotonou used in the 2021 feasibility study.

Consumables

Consumables were identified as non-reagent requirements/replacements and are related to the crushing, grinding and filtration circuits. Consumables represent 7.7% of the total processing plant OPEX. The following items have been included under consumables, but not updated, thus the costs reflect that of the 2021 study.:

- Primary Crusher liners.
- Screen mesh/panels.
- Filter cloths use 2021 rates +10%.

Consumables costs for these items were obtained from vendors as delivered costs up to the Port of Cotonou. Freight costs from the Port of Cotonou to the Dasa Plant site were obtained from Freight Companies specialising in transport and logistics in that region. Consumable costs have been left unchanged at this stage.

An updated cost was received for Grinding media and there was minimal variance from that reported in the previous study. The mill liner costs have increased significantly.

Maintenance

The process plant annual maintenance costs were derived from the total installed mechanical cost and using a factor of 5% and has been checked against the latest predicted costs which has remained unchanged since 2021 and represent 5.1% of the processing plant OPEX.

Laboratory Services

Laboratory costs have remained unchanged and represents 2.4% of the processing plant OPEX. The OPEX estimate for the laboratory and assay activities were based on the number of assays required per day and per year, as well as the type of assay (atomic adsorption, etc.). The assay costs are important for metallurgical accounting and process control and can be attributed to:

- Assaying of mine samples from blasthole drilling for grade control.
- Monitoring of grade/recovery for unit operations to permit optimization of the process plant.
- Environmental analysis.
- Duplicate assays and assaying of reference standard samples for quality control.

Labour

The labour (plant and plant maintenance) represents 13.6% of the processing plant OPEX. The labour estimate was determined from benchmarking against similar projects with comparable unit processes (crushing, grinding, leaching, and laboratory). Labour was estimated for plant operations personnel based on two 12-hour shifts.

The estimate includes staff for the following areas, as well as an allowance for contract labour:

- Process Plant management staff and administration.
- Health, Safety and Environment.
- Process Plant operations.
- Laboratory.
- Process Plant maintenance.

For the purposes of the 2024 OPEX estimate, the HSE component of the labour rates has not been included, following the initial reporting structure. The staff complement and rates differ from the 2021 study. The Process Plant annual cost is based on the sum of personnel classified under Plant Management and Mill Operators and the Maintenance Portion is comprised of personnel classified under Plant Maintenance.

The above personnel cost has been obtained from the “Site Fixed Variable Costs 21/01/2024) spreadsheet as provided by the Client.

An organizational roster outlining the labour requirement for the process plant was developed for the estimation of head count required for the plant.

Labour requirements are fluid and can influence the overall US$/ton plant operating cost by as much as 6% depending on the requirements at a point in time.

Diesel

Requirements for diesel for air heating for drying ore in the SAG mill was calculated based on heating energy and air volume calculations for moisture evaporation. Mobile equipment and light duty vehicles diesel consumption rates were estimated from Equipment Handbooks. Annual equipment operating hours were estimated from experience. Diesel prices have been updated to reflect an in-country diesel from of US$ 1.00/l, up from US$ 0.78/l initially. The costs of Diesel represent 8.5% of the processing plant OPEX.

Table 21-21: Processing Operating Cost Summary (2024 Estimate).

Category (2024)	US$/t milled	US$/lb	% OPEX
Consumables	6.23	0.68	7.76
Diesel	6.87	0.75	8.56
Reagents	25.32	2.77	31.57
Power	24.78	2.71	30.9
Maintenance	4.08	0.45	5.09
Laboratory	1.93	0.21	2.41
Labour - Process Plant and Maintenance	11.00	1.20	13.7
Total	**80.21**	**8.76**	**100.00**

Table 21-22 below indicates the operating costs as reported in the DASA 2021 OPEX and is included for comparison purposes.

Table 21-22: 2021 OPEX for Reference Purposes.

Category (2021)	US$/t milled	US$/lb U_3O_8 recovered	% of OPEX
Consumable	6.27	0.55	9.9
Diesel	6.05	0.53	9.5
Reagents	25.02	2.21	39.4
Power	16.47	1.45	25.9
Maintenance	4.08	0.36	6.4
Laboratory	1.98	0.17	3.1
Labour – Process Plant and Maintenance	3.69	0.33	5.8
TOTAL	**63.56**	**5.61**	**100**

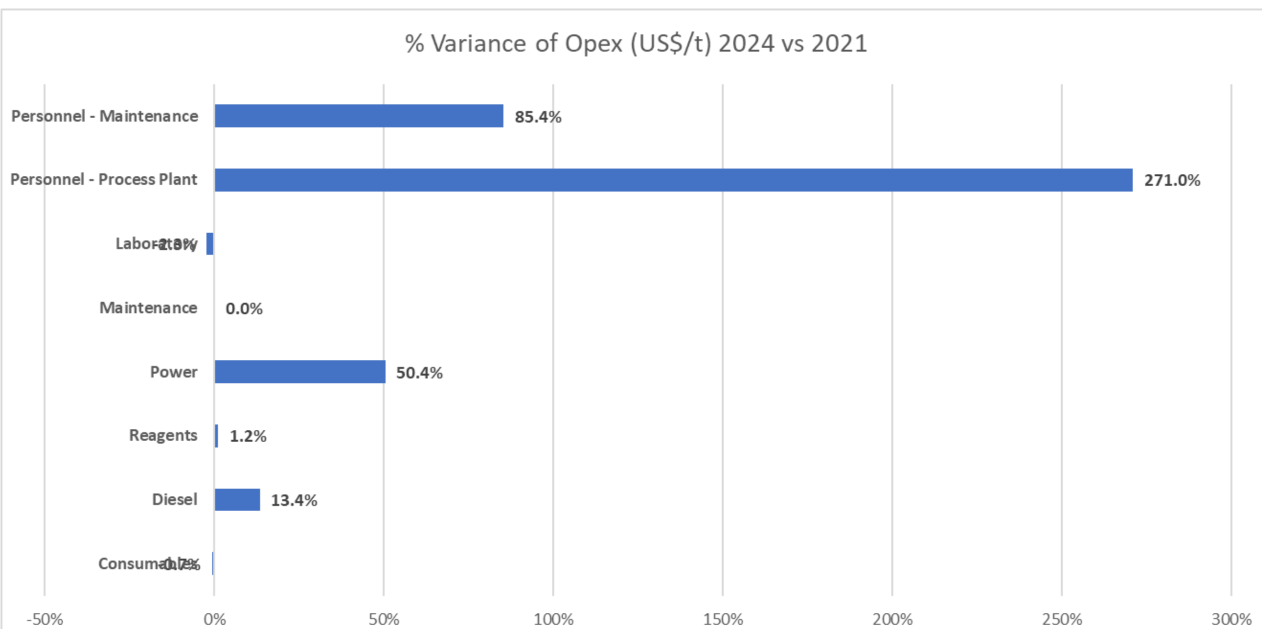

Figure 21-3: % OPEX Variance 2024 vs. 2021.

A comparison of the individual OPEX category contributions indicates a major difference being the higher weighting played by labour, and this is due to the client increasing the labour requirements.

- Reagents is being reported as an increase of < 2 % as compared to the feasibility study of 2021.
- Power and Diesel indicate a 50% and 13% increase respectively as compared to the feasibility study of 2021.
- Consumables are a lower contributor to the overall OPEX vs. 2021.

A breakdown highlighting individual components of the processing operating cost is shown in Figure 21-4 below.

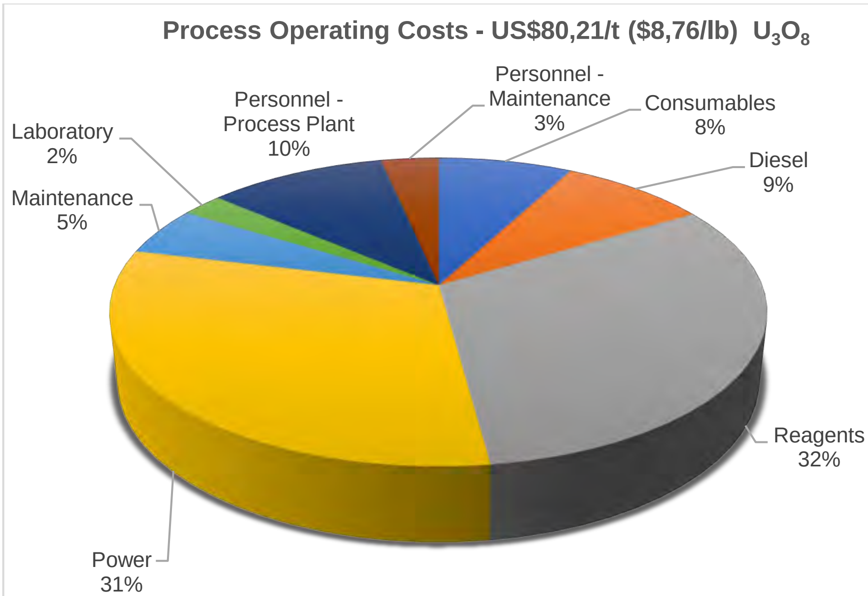

Figure 21-4: Plant Operating Cost Breakdown.

Reagents and Power collectively represent 65% of the total plant OPEX. Reagents are the most significant contributor to the OPEX, and its constituents is further delineated as follows.

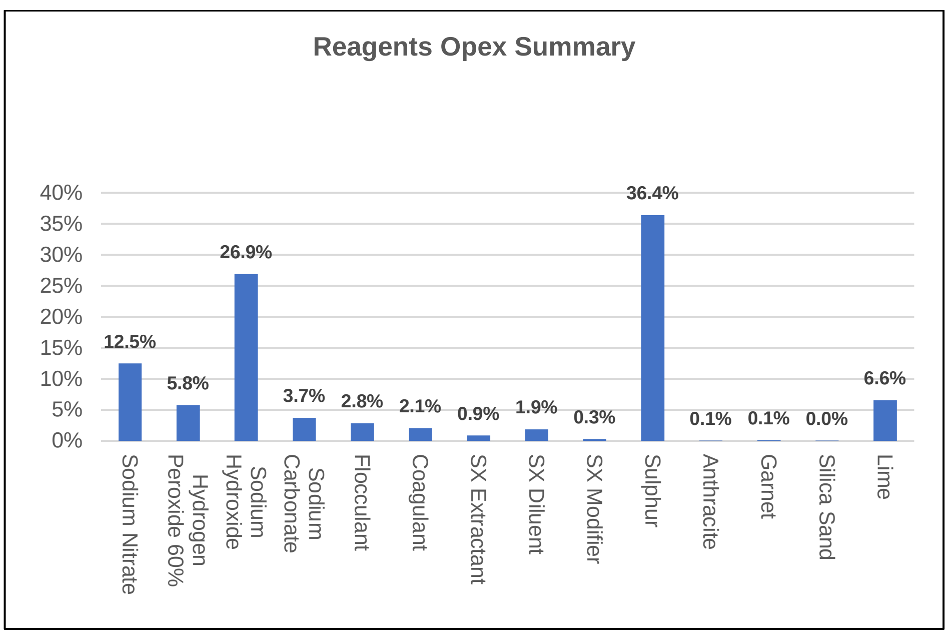

Figure 21-5: Reagent OPEX Summary.

**Barium Chloride is not reflected as the consumption rate (kg/d) has not been specified and currently does not comprise the reagent OPEX.*

General and Administration Costs

The General and Administration (G&A) cost components include the materials and supplies used by the administration and services groups. These costs comprise communications, office supplies, computer supplies and computer and software upgrades, light vehicle fuel and maintenance, insurance, camp accommodation operational costs, business travel, land lease and property costs. In addition, Marketing and Transportation costs have been included estimated based on $1 per lb of U_3O_8.

Annualized Dasa and Niamey site G&A costs are estimated at $ 26.25 per tonne of ore, or $2.46 per lb of U_3O_8. The full G&A costs ($36.93) are summarised in Table 21-23 below.

Table 21-23: Breakdown of Total G&A Costs.

Category	$ 000'	$/t milled
Management Salaries	23,214,540	2.89
Communications and IT	9,688,536	1.20
Insurances	18,626,250	2.31
Security	22,865,790	2.84
Camp Costs	124,008,095	15.41
Clinic	17,020,679	2.12
Safety, Health, and Environment	15,592,409	1.94
Community Relations	8.393,554	1.04
Marketing and Transportation	68,127,525	8.47
Other Support Costs	136,183,022	16.92
Total	**443,720,380**	**55.14**

22. ECONOMIC ANALYSIS

22.1. Cautionary Note

The reader is cautioned that the analysis and results reported in this chapter are forward-looking, and as a result there is no certainty that the forecast outcomes will be realized.

22.2. Introduction and Summary of Key Results

An economic assessment of the Project Feasibility Study for the Dasa Uranium Project has been developed. The project is in the Tim Mersoï Basin in Niger, a well-known and established uranium producing area.

The key metrics from this analysis are given Table 22-1 below.

Table 22-1: Key Metrics for the Phase 1 Dasa Project.

Project economics	**Unit**	**Value**
NPV @ 8 % after tax	$M	$917
IRR after tax	%	57.0%
Payback period from Jan 2024	years	4.2
Payback period from start-up	years	2.2
Cash flow (before capex & taxes)	$M	$2,948
Free cash flow	$M	$1,840
Operating costs		
Site cash cost (before royalties)	$/lb U_3O_8	25.62
Total cash cost	$/lb U_3O_8	30.73
All-in sustaining cost	$/lb U3O$_8$	35.47
Capital costs		
Initial capital costs	$M	$308
Sustaining capital costs	$M	$339

22.3. Model Development

A forward-looking discounted cash-flow model was developed in MS Excel to assess the economic potential of the project. The model accounts for the following line items:

- Revenue.
- Mining royalties.
- Operating costs.
- Taxes, and
- Capital costs.

The output from the model is the free cash flow, which is the revenue less the sum of the other items. Other metrics, such as the cash cost, the all-in sustaining cost, the NPV and the IRR are calculated.

The model is based on monthly mine and mill production as forecast in Chapter 16 for the underground mine plan.

Assumptions in this chapter are as follows:

- All monetary values refer to United States dollars.
- All values are in 2024 values, that is, no inflation has been included.
- Measures of units for mine production are metric tonnes, and for uranium as pounds U_3O_8.
- The input price is assumed constant at $75/lb U_3O_8.

22.4. Inputs to the Model

The inputs to the model are discussed for each line item in the model.

Revenue

Revenue is estimated from the production plan and the base case estimate of the price received for the product, U_3O_8. The base price has been estimated to be $75 /lb U_3O_8.

The mine production has been forecast to begin with development ore in Q4 2024 and stoping ore beginning Q4 2025. First ore through the mill is scheduled for January 2026. The mill ramp up is over a period of 11 months, starting with 50% of target 1,000 tonnes per day throughput and a recovery rate of 56% in the first month. By the 11th month, throughput is 1,000 tonnes per day and recovery rate is 94.15%. The mine plan has been set to provide an ore stockpile available for processing of between 3- and 4-months processing over the production period. Cash flow is based on an assumed revenue receipt 60 days after production.

Mining Royalty

The 2022 Niger Mining Code introduced a fixed royalty rate of 7% of revenues. Previously, the mining royalty was proportional to operating profitability. Under the Mining Convention applicable to Dasa, the project benefits from any reduction in taxes (including royalties) immediately, even though the other terms and conditions of its Mining Convention remain in force until its expiry in September 2027. The royalty is deductible for the determination of corporate income tax.

Operating costs

Operating costs for the mine and the plant have been forecast by the engineering teams based on labour, energy, and reagent contributions. The mining costs average to $77.08 /t milled and the processing costs are $84.69/t milled.

Corporate Income Tax

The corporate income tax rate in Niger is 30%. A three-year grace period is provided as an incentive. Exploration and development expenses are depreciated at a rate of 20%, plant at 10% and infrastructure at 5%. An initial balance of depreciable assets of $ 129.4 million is included in the calculation, including the historic exploration expenses incurred on the Dasa Project.

Niger has a value-added tax system with a rate of 19% and customs duties on imports of approximately 12%. The project is exempt from VAT and customs duties until commencement of production. Under the previous Mining Code, once production commences, the VAT rate was set at 0% and reagents, fuel and most consumables were exempt from customs duties throughout operations. The new Mining Code does not presently contain such exemptions but refers to provisions that may be included in regulations for strategic minerals. Uranium has historically been a strategic mineral, so it is expected similar exemptions will be enacted. The new Mining Code provisions do not impact the Dasa Project until after the existing Mining Convention expires in September 2027.

Changes in Working Capital

Working capital represents the amount required to fund the operations until the funds generated by the product sales are received, and calculated from product inventory, accounts receivable, supplies inventory less accounts payable. Changes in working capital are included in the cash flow projections, and initial working capital is recaptured at the end of the mine life.

Capital Expenditure

The mining and processing capital estimates have been prepared according to a class 3 estimate as defined by the AACE. Initial capital expenditures include contingencies ranging from 5% to 25%, for an overall average contingency of 14.1% before working capital investment. Sustaining capital expenditures include a contingency of 15% on mine development and capital expenditure. A closure cost provision of $0.23/pound U_3O_8 has been included so that it accumulates to the estimated closure costs of $25 million over the production period. The direct and indirect costs contributing to the initial and sustaining capital costs are given below.

Table 22-2: Initial and Sustaining Capital Expenditures.

Item	Amount, 000 $
Mining	58.8
Processing	83.2
Owners' costs & infrastructure	129.1
Contingency	37.2
Total Initial Capital Expenditure	**308.3**
Sustaining and closure costs	338.6
Total Initial plus Sustaining Capital expenditure	**646.9**

22.5. Results

Project Economics

The forecast of the economics of the project over the life-of-mine are given in Table 22-3 and the financial indicators are given in Table 22-4. The NPV_8 is $ 917 M, and IRR is 38.4% for the base case estimates.

Table 22-3: Forecast of the Project Economics Over the Life-of-Mine.

Item	LoM Total, $M
Revenue	5,041
Mining royalties	348
Operating costs	1,745
Operating income	**2,948**
Taxation	492
Capital costs & working capital	616
Free cash flow	**1,840**

Note: values may not add up due to rounding.

Table 22-4: Financial Indicators Over the Life-of-Mine.

Item	Unit	Value
NPV_8 after tax	$M	$917
IRR after tax	%	57.0%
Payback period from start of production	years	2.2
Cash flow (before capex)	$M	$2,948
Free cash flow	$M	$1,840
Operating costs		
Cash cost	$/lb U_3O_8	30.73
All-in sustaining cost	$/lb U_3O_8	35.47
Capital costs		
Initial capital costs	$M	$308
Sustaining capital costs	$M	$339

NPV is based on discounting to commissioning date, January 1, 2026, less undiscounted remaining capital costs.

The cash cost per pound of U_3O_8 represents the sum of the costs of mining, processing, mining royalties and site and offsite general and administrative costs, divided by the pounds of recovered U_3O_8. Site cash costs are calculated excluding royalties and selling expenses. The all-in sustaining cost per pound of U_3O_8 represents the sum of the costs of mining, processing, mining royalties, site and offsite general and administrative costs and the sustaining capital expenditures, divided by the pounds of recovered U_3O_8.

The unit costs by year are shown as follows in Figure 22-1 below:

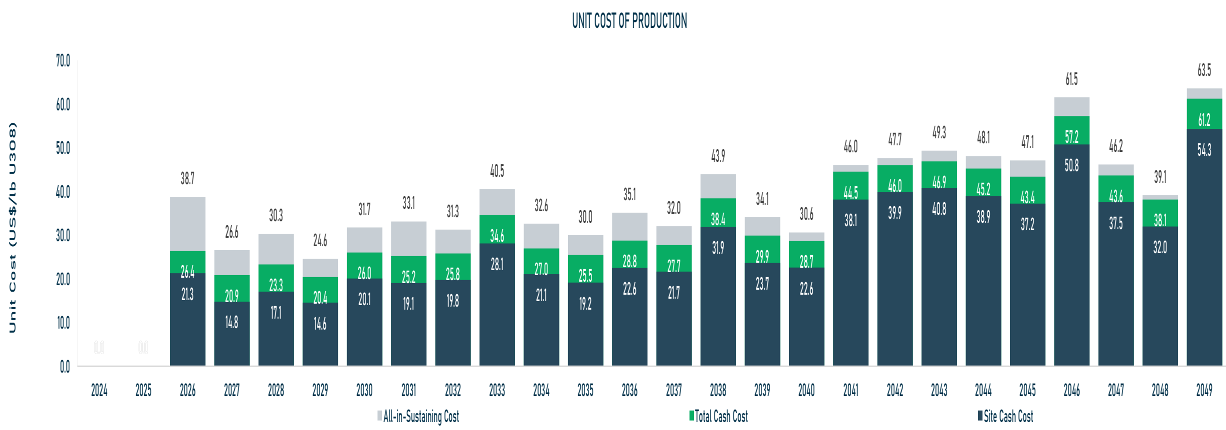

Figure 22-1: Financial Indicators Over the Life-of-Mine.

Free Cash Flow and Payback Period

The free cash flow is calculated as revenue minus operating costs, capital expenditures, taxation, royalties, and changes in working capital. The annual free cash flow and cumulative free cash flow are shown in. The cumulative free cash becomes positive 2.2 years after the start of production and 4.2 years inclusive of the remaining construction period.

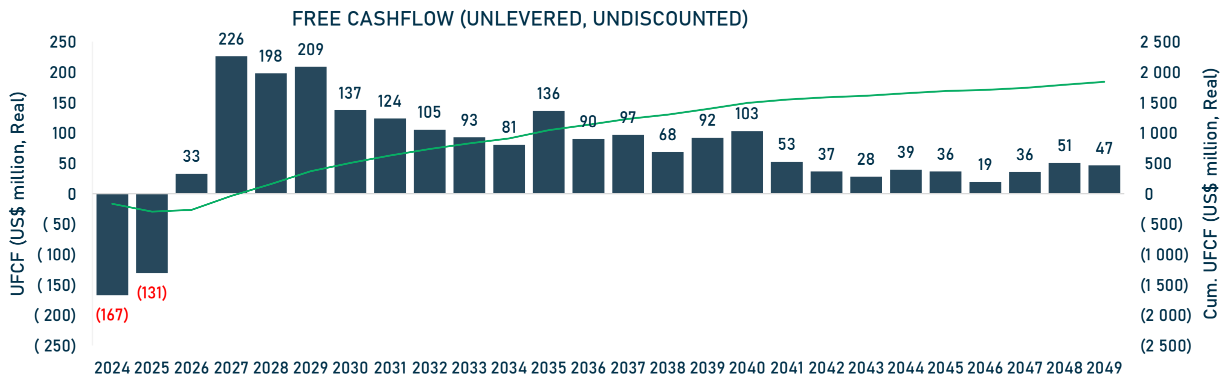

Figure 22-2: Annual Free Cash Flow Profile for the Dasa Phase 1 Project.

Composition of Cash Flow

The free cash flow and the composition of free cash flow are shown in Figure 22-3 below.

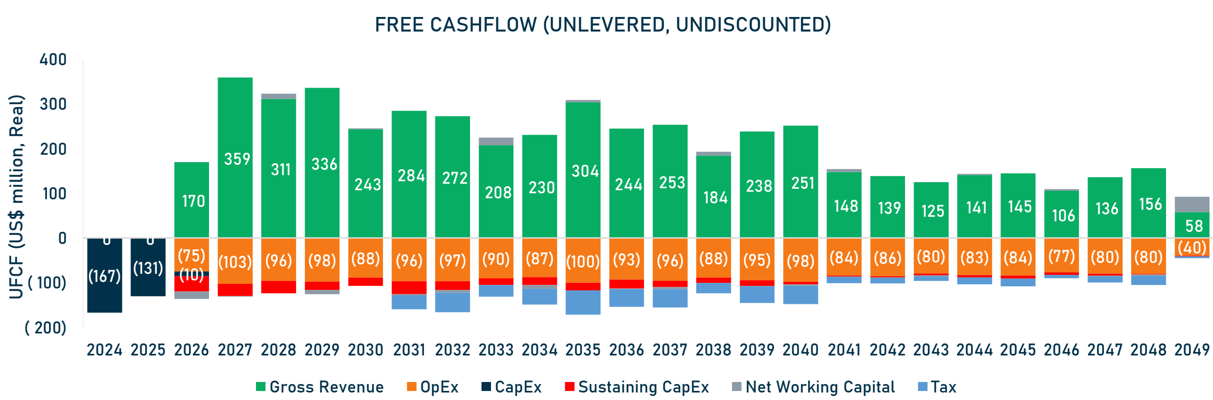

Figure 22-3: Free Cash Flow (Unlevered, Undiscounted).

Sensitivity Analysis Update

The sensitivity of the NPV_8 and the IRR to the changes in the input estimates to the financial model is shown in Figure 22-4, Figure 22-5 and Table 22-5.

The model is most sensitive to the price of U_3O_8.

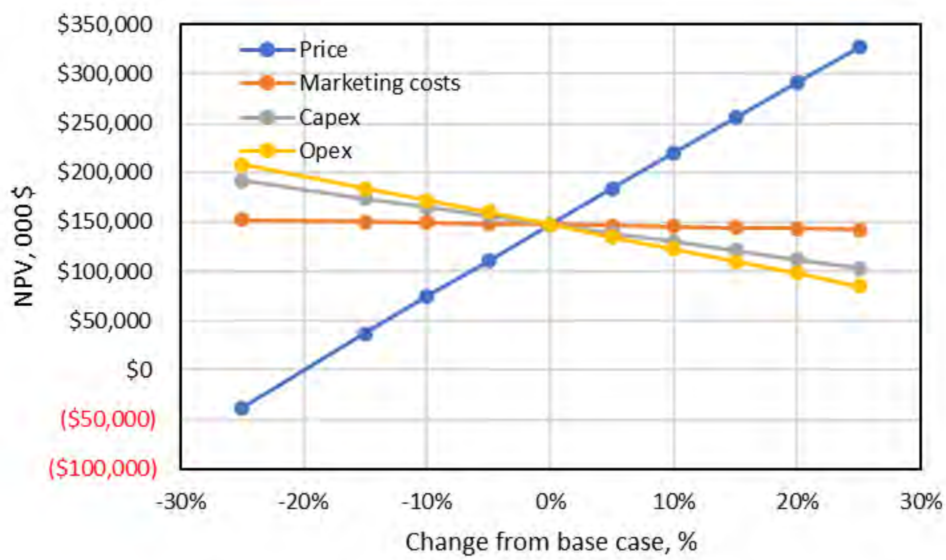

Figure 22-4: Sensitivity of the NPV_8 to Changes from the Base Case in the Input Costs for the Financial Model for the Dasa Project.

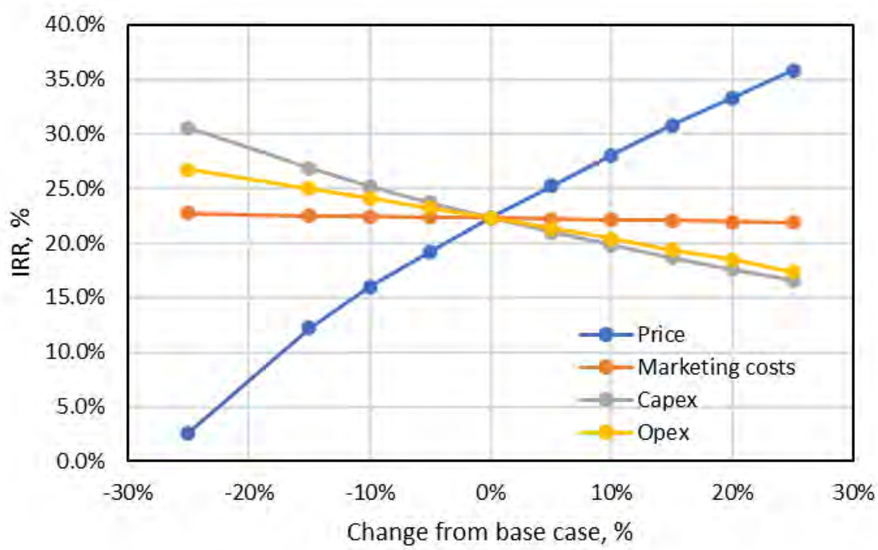

Figure 22-5 - Sensitivity of the IRR to Changes from the Base Case in the Input Costs for the Financial Model for the Dasa Project.

Table 22-5: Economic Sensitivity to Varying Price of U_3O_8

U_3O_8 price (per pound)	**$60**	**$75**	**$90**	**$105**
Before-tax NPV8	$656 M	$1,122 M	$1,572 M	$2,022 M
After-tax NPV8	$551 M	$917 M	$1,269 M	$1,621 M
After-tax IRR	**38.2%**	**57.0%**	**74.8%**	**92.9%**

Summary of Economic Model

The economic model is summarized in Table 22-6 below.

Table 22-6 Summary of Economic Forecast for the Dasa Project.

	Total	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049
Ore mined (000 Tonnes)	8,046.6	12.3	97.9	228.0	375.0	372.2	315.0	322.1	384.0	374.5	346.7	297.3	382.4	350.3	384.0	366.6	384.0	381.8	307.1	366.8	296.8	338.6	343.6	278.6	324.6	312.5	103.7
Ore Processed (000 Tonnes)	8,046.6			273.9	364.6	365.6	364.6	352.7	364.6	365.6	364.6	304.5	364.6	365.6	364.6	364.6	364.6	365.6	364.6	364.6	299.5	341.0	339.6	285.8	324.6	312.5	103.7
Grade (ppm)	*4,113*			*5,992*	*6,499*	*5,424*	*6,306*	*4,643*	*5,058*	*4,928*	*3,445*	*5,130*	*5,195*	*4,271*	*4,558*	*3,041*	*4,187*	*4,505*	*2,500*	*2,455*	*2,728*	*2,597*	*2,752*	*2,266*	*2,723*	*3,246*	*2,982*
Contained U3O8 (million lbs)	73.0			3.6	5.2	4.4	5.1	3.6	4.1	4.0	2.8	3.4	4.2	3.4	3.7	2.4	3.4	3.6	2.0	2.0	1.8	2.0	2.1	1.4	1.9	2.2	0.7
Recovery	*93.4%*			*78.4%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*	*94.2%*
Production U3O8 (million lbs)	**68.1**			**2.8**	**4.9**	**4.1**	**4.8**	**3.4**	**3.8**	**3.7**	**2.6**	**3.2**	**3.9**	**3.2**	**3.4**	**2.3**	**3.2**	**3.4**	**1.9**	**1.9**	**1.7**	**1.8**	**1.9**	**1.3**	**1.8**	**2.1**	**0.6**
Revenues	5,041.4			169.6	359.0	311.2	335.8	243.3	284.3	272.5	207.7	230.4	303.6	244.5	253.1	184.5	238.2	251.5	147.5	138.7	124.7	141.1	144.7	105.9	135.8	156.2	57.8
Mining costs	620.2			17.9	23.8	22.3	20.9	21.8	25.8	26.8	27.9	23.2	28.2	26.9	28.2	29.0	29.6	31.3	28.4	30.4	27.0	28.3	28.9	26.9	28.0	27.4	11.3
Processing costs	681.5			25.0	32.2	31.1	32.0	29.9	30.7	30.6	29.0	28.8	30.8	29.9	30.2	28.6	29.8	30.2	28.0	27.9	26.5	27.5	27.6	25.8	27.2	27.4	14.9
Site & Niamey costs	375.6			17.4	16.9	16.9	16.6	16.6	16.6	16.5	16.3	16.3	16.3	16.3	16.3	15.8	15.8	15.7	15.7	15.7	15.7	15.7	15.7	15.6	13.7	12.6	8.7
Selling expenses	68.1			2.8	4.9	4.1	4.8	3.4	3.8	3.7	2.6	3.2	3.9	3.2	3.4	2.3	3.2	3.4	1.9	1.9	1.7	1.8	1.9	1.3	1.8	2.1	0.6
Royalties	348.1			11.7	24.8	21.5	23.2	16.8	19.6	18.8	14.4	15.9	21.0	16.9	17.5	12.8	16.4	17.4	10.2	9.6	8.6	9.7	10.0	7.3	9.4	10.8	4.0
EBITDA	**2,947.8**			**94.7**	**256.4**	**215.3**	**238.3**	**154.8**	**187.8**	**175.9**	**117.4**	**143.0**	**203.4**	**151.2**	**157.4**	**96.1**	**143.4**	**153.5**	**63.3**	**53.2**	**45.1**	**57.9**	**60.6**	**29.0**	**55.7**	**75.9**	**18.3**
Initial capital expenditures	297.6	167.1	130.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Sustaining capital expenditures	333.4			45.0	27.1	27.7	18.8	18.6	29.4	19.6	14.8	17.7	16.8	19.9	14.1	12.2	12.5	5.9	2.4	2.6	3.8	4.8	6.7	5.5	4.3	1.6	1.5
Closure fund payments	15.9			0.6	1.2	1.0	1.1	0.8	0.9	0.9	0.6	0.7	0.9	0.8	0.8	0.6	0.7	0.8	0.5	0.4	0.4	0.4	0.4	0.3	0.4	0.5	0.2
Working capital	-30.7			16.3	1.9	-11.3	9.5	-1.9	2.3	6.3	-17.0	8.6	-5.0	1.2	4.4	-8.3	0.7	2.7	-6.8	0.7	1.8	-2.4	0.3	-3.6	1.2	2.3	-34.4
Pre-tax cash flow	**2,331.6**	**-167.1**	**-130.5**	**32.9**	**226.3**	**197.9**	**208.9**	**137.3**	**155.2**	**149.1**	**119.0**	**116.0**	**190.6**	**129.4**	**138.1**	**91.6**	**129.5**	**144.2**	**67.2**	**49.5**	**39.2**	**55.0**	**53.1**	**26.7**	**49.9**	**71.5**	**51.0**
Income taxes	492.1			0.0	0.0	0.0	0.0	0.0	31.6	43.7	26.3	35.3	54.6	39.7	41.3	23.2	37.3	41.3	14.6	12.8	11.0	15.6	16.7	7.5	14.3	21.0	4.3
Post-tax cash flow	**1,839.5**	**-167.1**	**-130.5**	**32.9**	**226.3**	**197.9**	**208.9**	**137.3**	**123.7**	**105.5**	**92.7**	**80.6**	**136.0**	**89.7**	**96.8**	**68.4**	**92.1**	**102.9**	**52.5**	**36.7**	**28.2**	**39.5**	**36.4**	**19.2**	**35.6**	**50.5**	**46.7**

23. ADJACENT PROPERTIES

There are no third-party properties currently in production adjacent to Dasa project site.

Global Atomic Fuels Corporation has held the AE3 and AE4 Exploration Permits since 2008. The Dasa deposit was carved out of the AE3 Exploration Permit and is now held under a separate Mining Permit comprising 25 km^2. Historic resources have been identified on the remaining Adrar Emoles Exploration Permit, although significantly lower grade than the Dasa deposit. A resource known as the Isakanan resource has also been identified on the AE4 Exploration Permit. Application has been made for an extension or renewal of the AE3 and AE4 Exploration Permits and with additional drilling and test work, such deposits could become economic for processing through the Dasa processing plant in the future.

The Imouraren uranium project is located approximately 40 km northwest of the AE3 Exploration Permit and the Dasa deposit, and approximately 80 km south of Arlit and about 160 km north of Agadez. The Imouraren project is held by Imouraren SA (66.65% owned by Orano Expansion, 10% by the Republic of the Niger and 23.35% by SOPAMIN). Orano (2022) reports that the deposit area holds a total (100% share) of 211 million tonnes of Proven and Probable Reserves grading 956 ppm as at 31 December 2022.

In 1963, the CEA discovered the uranium shows of Mont Imouraren. Development of the Imouraren uranium project between 1974 and 1977 by the Cogema-Conocoonarem association led to the identification of three world-scale deposits (Imfout, Imatra, Imola) in an area of 40 km^2 (El Hamet and Idde, 2009). Orano (then Areva) was granted an exploration licence in 2006 for Imouraren. By 2006, more than 55 km of development drilling and bulk sampling had been completed (Kinnaird and Nex, 2016). The resumption of work in 2006 enabled the discovery of two other shallow deposits (Imca 25 and Imaren), situated 5 km from the previous ones in an area of 1.2 km^2 (El Hamet and Idde, 2009). Following a Feasibility Study completed at the end of 2007, Orano was awarded an operating permit to mine the deposit in early 2009 (Orano, 2019b). The project proposed an open pit mine with annual production capacity of 5,000 tonnes and lifespan of 35-years. However, since 2015, production start-up work has been suspended. Orano has been investigating the potential mining of Imouraren by the In Situ leaching methodology but has not yet made a production decision.

The following geological description of Imouraren is taken from El Hamet and Idde (2009).

“The Imouraren licence is situated in the eastern part of the Tim Mersoï basin. The geological formations are Carboniferous to Cretaceous continental, terrigenous, detritic formations lying on the Pan African basement dipping gently to the west. The deposit lies within the Jurassic Tchirézrine 2 formation; this formation is confined at the top and bottom by two formations with very fine grain size distribution and low permeability, Irhazer-Assaouas and Abinky. The mineralization is concentrated in heterogranular sandstone facies of fluvial origin with intercalating levels of analcime. The sandstones of Tchirézrine are general poorly cemented; the cement is made up of secondary silica, greenish clay, analcime, kaolinite, limonite, and haematite. The primary source of uranium seems to be in the Aïr volcanism, as indicated by analcimolite. In addition to the standard factors (stratigraphic, sedimentological, palaeogeographical, and tectonic), mineralization control seems to be the result of two phenomena: dispersion by oxidation of a syngenetic mineralization and re-concentration of an epigenetic mineralization by roll-type phenomena."

The Imouraren mineralization is a special case, differing from other deposits. It is 90% composed of hexavalent secondary uranium minerals (uranotile, meta-tyuyamunite) and 10% of primary minerals (coffinite, pitchblende), appearing in sandstone cement, at the centre of analcime grains and pebbles and in epigenized organic debris. Uranotile (Ca $(H_3O)2(UO_2)2(SiO_4)23H_2O$) is the most abundant mineral and is manifested in small fibroradiated clusters underlining the stratification or filling in the imprints of organic debris. These uranium minerals are often associated with copper sulphides and silicates (chalcocite and chrysocolla) and even with native copper; vanadium is present but often linked with chlorites in the form of montroseite.

Unlike the other deposits in the region, the Imouraren uraniferous mineralization is weakly carbonated (0.2% to 0.5% calcite). Iron minerals (pyrite, haematite, goethite), sulphates (gypsum, barytine) and phosphates (apatite) are not very abundant. Organic matter is rare or absent. The mineralization is spread vertically over three horizons in the whole sandstone facies of Tchirézrine 2 at a cumulative average thickness of 55 m at depths of between 105 m and 165 m. Laterally, the mineralization is subdivided into three zones from north to south: Imatra, Imfout and Imola. In the west Arlit-In Azaoua flexure area, two new average size and shallow (25–35 m) deposits, called Imca 25 and Imaren, were found during recent exploration activity.

The reader is cautioned that the Qualified Person has been unable to verify the information presented in this section and this information is not necessarily indicative of the mineralization on the Property that is the subject of this Technical Report.

24. OTHER RELEVANT DATA AND INFORMATION

The Feasibility Study presented in this report presents an optimized mine plan and recovery process for the Dasa deposit based on the extraction 8.047 million tonnes of mineralised material from a sub-vertical section of the deposit on the flank of the graben, from an average depth of 300 m below surface. The average processed grade of 4,113 ppm represents the ability to produce 73 million lb U_3O_8 for an expected revenue of $5.041 billion at a price of $75/lb U_3O_8. Value opportunities exist in extending the mine-life beyond the 24-years, based on the large mineral resource inventory that exists at Dasa. A large volume of mineralised material in the Inferred Resource category is present in the flat-lying portions of the graben between 400 metres and 800 metres below surface that could be mined in future decades. In addition, the deposit remains open along strike and at depth.

Increasing the processing plant throughput by optimizing plant availability and potential capacity expansions would have the effect of significantly improving the Dasa Project NPV.

No additional information or explanation is necessary to make this Technical Report understandable and not misleading.

25. INTERPRETATION AND CONCLUSIONS

METC Engineering has completed a Feasibility Report (FS) for the Dasa Uranium Project located in the central part of the Republic of Niger, West Africa. The results of this work indicated that an underground mine, process plant and all associated infrastructure is an economically viable project and warrants development to the phase of construction and development. The work has been complete at feasibility study level to an AACE Class 3 level of accuracy. The mineral resource estimate is based on estimates by AMC Consultants effective May 12, 2023. The mineral reserve estimate has been undertaken by Bara Consulting and is based on the outcomes of the feasibility study.

The sections below provide the key conclusions for the study.

25.1. Geology and Resources

The mineral resource estimate (MRE) was prepared with an effective date of May 12, 2023 (AMC Consultants, 2023) and is repeated in this report. The MRE has been prepared in accordance with CIM Definition Standards for Mineral Resources and Mineral Reserves (10 May 2014).

No new exploration work has been completed at the Project since the MRE's effective date of May 12, 2023.

The results of this Mineral Resource estimate are summarized in Table 25-1 below.

Table 25-1: Dasa Mineral Resources with an Effective Date of May 12, 2023

Mineral Resource category	Tonnes (Mt)	eU_3O_8 (ppm)	Contained Metal (Mlb)
Indicated	10.09	4,913	109.3
Inferred	4.45	5,243	51.4

Notes:

Mineral Resources are classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (10 May 2014).

The MRE was prepared by Dmitry Pertel, MAIG, (AMC Consultants).

The Effective Date of the MRE is May 12, 2023.

Mineral Resources are estimated based on an underground mine.

A cut-off grade of 1,480 ppm eU_3O_8 has been applied for the resources.

A bulk density of 2.36 t/m^3 has been applied for all model cells.

Rows and columns may not add up exactly due to rounding.

No Measured Resources or Mineral Reserves of any category were identified.

It can be reasonably expected that the following applies to the project.

- The majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued infill drilling.
- This Mineral Resource is a reliable estimate of the mineralization present at the Dasa Project, as supported by the current lithological model, analytical data, and geophysical logging results.
- The data used as inputs to the model have been collected and compiled at high standard and indicate that the Project is a high-quality mineral asset.
- Mineralization potential exists within the Project along strike to the north and south, as well as within the graben providing significant upside potential.
- Additional exploration and infill drilling work is warranted at the Project to enlarge the resource and upgrade current Mineral Resource to a higher classification.
- Infill drilling in critical areas would significantly reduce any potential risk in future Mineral Resource updates.

25.2. Mining

The following observations are based on the results of the Dasa underground optimizations performed during the feasibility study:

- The work undertaken during the feasibility study has demonstrated that the Dasa deposit supports a high-grade low-volume underground mine extraction strategy that is amenable to standard productivity mechanized mining methods.
- A Mining method trade-off study has been undertaken and has concluded that a long hole open stoping method with cemented hydraulic backfill in a transverse configuration is the most appropriate method for the Dasa deposit.
- Based on the mining evaluation undertaken and the Mineral Resource in the Measured and Indicated categories, the Dasa Project can support a mine for approximately 24-years.
- Significant groundwater inflows are expected into the mine during its operation, and strategies to handle this water should be studied further.

Bara Consulting concludes the following:

- The mineral resource model classified as Indicated is sufficiently reliable to support engineering and design studies to evaluate the economic viability of a mining project to feasibility study levels of accuracy.
- The mining scenario modelled, has demonstrated economic feasibility, based on a long-term uranium price of $75/lb.
- The Project has a robust resource base with opportunities to expand along strike and further extend the life of mine.

25.3. Metallurgy and Processing

Results of the metallurgical test work show the Flank Zone mineralized material is readily amenable to the pugging and curing process as used successfully by other uranium mines in the area and is similar to the Orano operation at Arlit, Niger.

Dry milling to a particle size of P_{95} 550 µm followed by pugging with sulphuric acid (produced in an acid plant on site) and recycled nitric acid, followed by three hours of conveyor belt curing, enable high uranium dissolution with minimal negative silicate solubility and can produce superior leaching to conventional heap or tank leach processes.

Re-pulping and subsequent solid/liquid separation on belt filters produce a disposable residue and a clean uranium liquor. This liquor can be clarified using solvent extraction with the resultant OK liquor being easily precipitated with caustic and hydrogen peroxide. Filtration and drying can produce final uranyl peroxide (UO_4), “yellowcake” product cake for packing and dispatch.

Extensive test work was undertaken on available samples with multiple pilot plant runs having been completed. The test work confirmed that the previous metallurgical ore treatment process considered, can be upscaled and replicated in a commercial operation. The test work also identified some process improvements, namely:

- Increased water replacement in the circuit to alleviate a build-up of impurities in raffinate solution and the negative impact these impurities have on dewatering the tailing stream, and on the horizontal vacuum belt filter performance.
- Improved extraction efficiency, using sodium carbonate as a stripping reagent to replace the sodium chloride (previously selected).
- 2 stage precipitation (sodium di-uranate, followed by hydrogen peroxide) to reduce reagent consumption and improve the plant water balance.
- Utilising a combination of flocculants and coagulants to improve tailings belt filtration.

Build-up of impurities in raffinate solution and the negative impact these impurities have:

- In dewatering the tailing stream, and
- On the horizontal vacuum belt filter performance

25.4. Economic Assessment Update

The forecast of the economics of the project over the life-of-mine are given in Table 25-2 and the financial indicators are given in Table 25-3. The NPV_8 is \$ 917 M, and IRR is 57.0% for the base case estimates.

Table 25-2: Forecast of the Project Economics Over the Life of Mine.

Item	LoM Total, $M
Revenue	5,041
Mining royalties	348
Operating costs	1,745
Operating income	**2,948**
Taxation	492
Capital costs	616
Free cash flow	**1,840**

Note: values may not add up due to rounding.

Table 25-3: Financial indicators over the life-of-mine.

Item	Unit	Value
NPV_8 after tax	$M	$917
IRR after tax	%	57.0%
Payback period	Years	2.2
Cash flow (before capex)	$M	$2,948
Free cash flow	$M	$1,840
Operating costs		
Cash cost	$/lb U_3O_8	30.73
All-in sustaining cost	$/lb U_3O_8	35.47
Capital costs		
Initial capital costs	$M	$308
Sustaining capital costs	$M	$339

The cash cost per pound of U_3O_8 represents the sum of the costs of mining, processing, mining royalties and site and offsite general and administrative costs, divided by the pounds of recovered U_3O_8. The all-in sustaining cost per pound of U_3O_8 represents the sum of the costs of mining, processing, mining royalties, site and offsite general and administrative costs and the sustaining capital expenditures, divided by the pounds of recovered U_3O_8.

These results are based on a selling price of $75.00 / lb U_3O_8 and any improvement on this has significant positive impacts on the project as demonstrated in the sensitivity analysis. The sensitivity of the NPV_8 and the IRR to the changes in the input estimates to the financial model is shown in Figure 25-1 and Figure 25-2. The model is most sensitive to the price of U_3O_8.

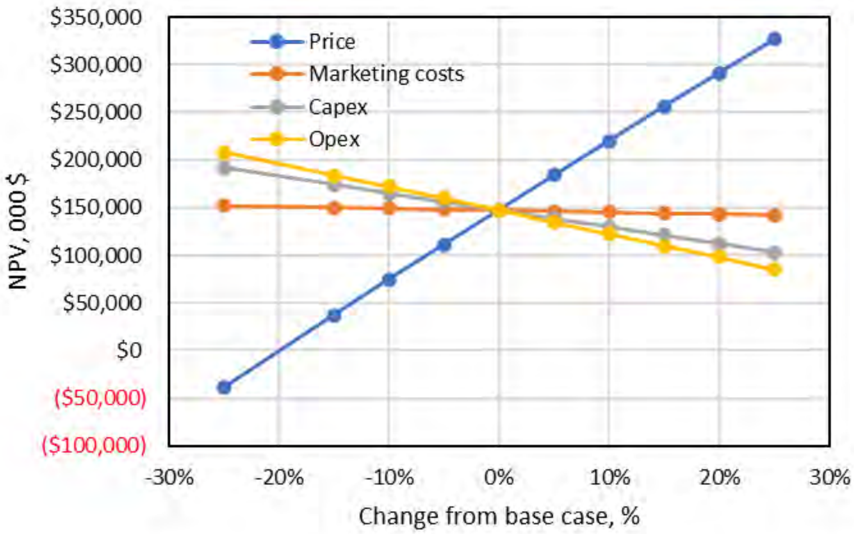

Figure 25-1: Sensitivity of the NPV8 to changes from the base case in the input costs for the financial model for the Dasa Phase 1 project.

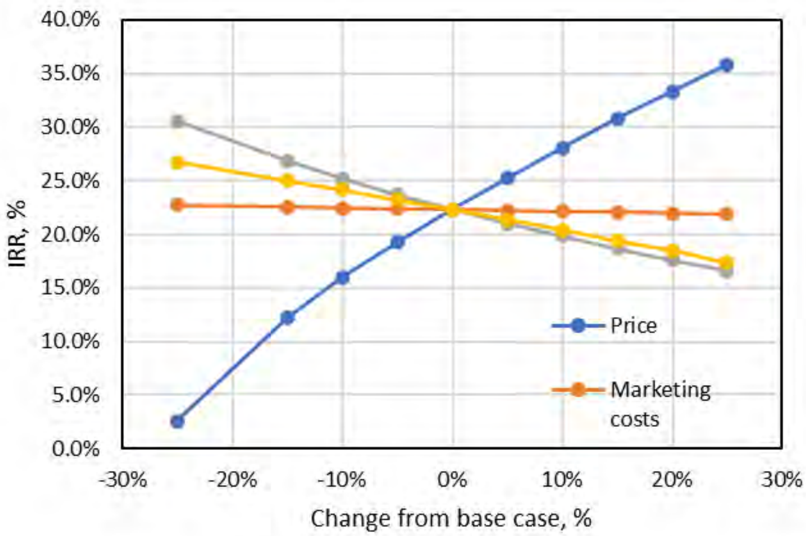

Figure 25-2: Sensitivity of the IRR to changes from the base case in the input costs for the financial model for the Dasa Project.

25.5. Risks

A review of the Project risks identified the following:

General

Legal, title, taxation, marketing, political, or other relevant issues could potentially affect the Dasa Project, however none of these risks are currently considered. Potential technical, environmental, socio-economic, and permitting risks are discussed below.

Tenure and Permitting

The relevant mining and environmental permits have been issued (as detailed in section 4 of this report) and as such, the project exposure to risks related to tenure and permitting should be considered minimal.

Mineral Resource Estimate

The risks associated with the mineral resource estimate remain unchanged from the 2020 PEA Technical Report and no further geological drilling has been undertaken since the 2020 PEA.

Mining

A project risk assessment was undertaken and the following residual risks relating to mining have been identified, that, in the Qualified Persons opinion, could impact the Project outcome:

- Exposure to high levels of radiation.
- Working in an underground mine with a high concentrate Uranium orebody, will pose significant health risks to personnel in terms of radiation exposure. In order to reduce the radiation exposure levels an adequate ventilation system has been designed, which will provide for a large volume of air and frequent air replacement through the underground mine. It is further recommended that during the implementation phase, a specific SOP should be drawn up and continuous personnel monitoring introduced to monitor and manage this risk.

Larger than anticipated ground water inflows entering the underground mine.

- With uncertainty in the hydrological information, there is a risk that more than anticipated ground water will be entering the underground working areas. To reduce this risk, spare pumping capacity and equipment have been included in design.
- High ground water inflows in the decline reporting to development end, reducing decline advance rate.
- With uncertainty in the hydrological information, there is a risk that more than anticipated ground water will be entering the decline and report to the development end. Cover drilling for water and dewatering ahead of the decline development end has been recommended as a risk mitigation.

Mineral Reserves could be impacted by the following issues:

- Changes to geotechnical conditions that have been modelled for the site resulting in an impact to the mining design and subsequent impact on mining costs, grade, and extraction ratios.
- Increase in the estimated groundwater inflows into the mine resulting in the requirement to increase water handling and disposal capacity at increased costs.
- Changes to input assumptions for the cut-off grade calculation used in the stope optimiser software, which may impact on the volumes of payable Mineral Resources.
- Changes to the metallurgical recovery when scaled up to full size plant impacting on revenue.
- Changes in economic assumptions over time relating to uranium price and exchange rates.
- Country risk, including changes to tax laws and permitting requirements.

Geotechnical

A geotechnical study has been undertaken to feasibility study levels of accuracy and a geotechnical risk assessment has been undertaken, key risks identified are:

- Insufficient data.
- Data collection standards.

The mitigation for both of the identified risks is the same and are as follows:

- Conservative design decisions taken, providing conservative design criteria for input into the mine design.
- Collection of additional data on initiation of project implementation activities.
- Use the additional data to further validate the design work undertaken.

Processing

The metallurgical test work undertaken together with the mineral processing investigation undertaken in the feasibility study has enabled a recovery process to be determined that can be constructed and put into operation. Previous solid liquid separation processes concerns have been derisked after additional test work has been completed.

Environmental and Social

While fully permitted for construction and operation of the Dasa Mine, GAC recognises the current ESIA does not meet all international expectations and has completed a gap analysis against Equator Principle (EP4) and IFC Performance Standards. The following areas will be addressed in the ESIA Addendum scheduled for completion H2 2022:

- Site specific time series baseline data including weather data from an on-site station.
- Systematic biodiversity studies.
- Additional community studies.
- Impact Assessment will be supplemented with quantitative data, post mitigation residual impacts.
- GHG calculations are included in the FS and will be included in climate change considerations.
- Potential for Indigenous People to be affected (Tuareg) (PS7).

The ESIA Addendum will follow the completion of field work referenced above and will seek to achieve “International ESIA” standards of detail and disclosure.

25.6. Opportunities

Geology/Mineral Resource Estimate

Infill drilling in critical areas such as the inferred parts of the Flank Zone would significantly reduce any potential risk in future Mineral Resource updates and further economic assessment of the Project. This is equally true in the deeper parts of the deposit that may be amenable to underground mining.

GAC should consider progressing additional exploration to expand resources at Dasa, along strike in the Flank Zone and at depth within the graben.

Mining

The following opportunities have been identified in regard to the mining design for the Dasa Project:

- Further optimisation of the geotechnical design - larger stopes would lead to a reduction in development and hence reduce costs.
- Use of battery electric vehicles (BEV) in the mine – reducing the heat load, diesel contaminants in the air and hence a reduction in costs.
- Increases in the Mineral resources along strike would lead to a longer mine life and/or an increase in production potential. Specifically, the potential for additional Mineral resources along strike in zone 1, would mean the mine would operate at a shallower depth for longer. This could result in significant savings (or deferment of) capital expenditure to access and mining the deeper areas.
- Dewatering of the ground ahead of mining, (by establishing a surface wellfield) which would reduce the pumping infrastructure capital and operating costs.

The Mineral Reserves have been calculated with an assumed U_3O_8 price of $35/lb. Using higher prices will increase the tonnage and contained uranium content at Dasa.

Metallurgy/Mineral Processing

Results of metallurgical test work shows the mineralogy and metallurgy of the Dasa mineralization is amenable to pugging and curing process, similar to the Orano operation at Arlit, Niger. The additional test work completed has identified a reliable process route relative to the ore tested.

Detailed design along with experience from other plants have help identify plant areas that will require careful attention and these areas have had careful design considerations to prevent process upsets. These include optimised ore drying within the mills, careful reagent addition to prevent excessive leaching and generation of excess poor filterable tailings ore which require more attention in optimising tailings belt filter operation and wash efficiency.

The final plant design presented is good, in line with the characteristics identified from all test work completed.

Equipment has been sized relative to test work results. Opportunities exist when the process plant is operational to increase the equipment up time and hence increase the overall plant utilisation.

Equipment utilisation has been assumed to be a conservative 86%, partly as a result of intermittent power for the local grid. The opportunity exists to increase the process plant utilisation (and hence production of U_3O_8) by providing a more reliable power supply.

26. RECOMMENDATIONS

The Qualified Persons and METC Engineering recommend that, based, on the work done in the feasibility study including the favourable economic analysis, that the project move into the next phase, namely construction and development.

The following activities per discipline should be considered in the execution phase of the project as they will enhance the robustness of designs and refine the overall projects cost base.

26.1. Geotechnical

Bara Consulting recognises that the underground geotechnical stope stability analysis is a key driver for the determination of the stope configuration.

Development of a live geotechnical model, for the mine, should be updated regularly through the life of mine, with information from geotechnical drilling, Mineral Resource drilling and from development and mining observations once project activities proceed underground.

26.2. Processing

The feasibility study provides a detailed assessment of the process plant requirements to meet the recoveries achieved in the extensive test work program that was undertaken. The requisite level of design and capital cost determination undertaken in the feasibility study demonstrates that a process plant can:

- Be constructed with known, commercially available equipment.
- Achieve the desired recovery expectations.
- Be operated at acceptable operational cost levels.

26.3. Hydrology and Hydrogeology

Review the handling and management of ground water that could flow into the mine workings and determine the optimum solution to handle this water.

26.4. Mining

The feasibility study mining evaluation should be further optimised during the detailed design and project implementation phase of the project. Key areas requiring focus during the next phase are:

- Further refinement of stope optimisation and designs based on MRE update and additional geotechnical data.
- Further study of options and optimisation of the pumping arrangements for excess mine water based on the ground water modelling undertaken in the feasibility study.
- Further evaluation and optimisation of ventilation simulation and modelling.
- Completion of detailed designs for the mine support and services infrastructure to support project implementation.
- Update of the first principal capital and operating cost models based on the detailed design work to be undertaken.

26.5. Environmental

Recommendations for next steps include those outlined in Section 20.6 which includes the collection of additional baseline data and development of an ESIA addendum and associated management plans including an Environmental and Social Management System (ESMS).

26.6. Recommendations Budget

The proposed budget for these recommendations is shown in Table 26-1 below.

Table 26-1: Budget for Recommendations.

Item No.	Topic	Estimated Cost ($)	Timing (Year)
1	Geotechnical	65 000	2024, 2025
2	Processing	45 000	2022
3	Hydrology and Hydrogeology	55 000	2022
4	Mining	100 000	2024, 2025
5	Environmental	100 000	2022, 2023
	Total	**365 000**	

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