Dasa Uranium Project

Phase 1 Feasibility Study

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Prepared For

Global Atomic Corporation

Qualified Persons

John Edwards, B.Sc. Hons., FSAIMM, Chief Metallurgist, METC Engineering

Andrew Pooley, B. Eng (Hons)., Mining Engineering, FSAIMM, Managing Director, Bara Consulting

Dmitry Pertel, M.Sc., MAIG, CSA Global Consultants Canada Limited

George A Flach, P.Geo., VP Exploration – Global Atomic Corporation

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Report Prepared By

METC Metallurgical Engineering Technology & Construction

BARA CONSULTING

epoch resources (pty) ltd

CSA

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Report prepared for

Client Name	Global Atomic Corporation
Project Name/Job Code	Dasa Uranium Project, FS NI 43-101 Report 0093-00-00-REP-0001
Contact Name	Ronald S. Halas
Contact Title	Chief Operating Officer
Office Address	8 King Street East, Suite 1700, Toronto, Ontario, M5C1B5, Canada

Report issued by

METC Engineering	METC Engineering (Pty) Ltd
	12 Autumn Road
	Rivonia
	Johannesburg, South Africa
	T : +27 (0) 87 808 8862 E: info@metcengineering.com

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Author/Qualified Person Listing

Dmitry Pertel,	MSc., MAIG
John Edwards	B.Sc. Hons., FSAIMM, Chief Metallurgist, METC Engineering
Andrew (Jim) Pooley	B. Eng (Hons)., Mining Engineering, FSAIMM, Managing Director, Bara Consulting
George A Flach	P.Geo. VP Exploration – Global Atomic Corporation

METC Engineering Internal Reviewer and Peer Review Listing

Ross Manning	M.Sc., P. Eng. METC Engineering, Technical Director
Nico Maritz	METC Engineering, Engineering Director

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Report issued by ..... - 3 -
Report information ..... - 3 -
Author/Qualified Person Listing ..... - 3 -
METC Engineering Internal Reviewer and Peer Review Listing ..... - 3 -
Certificate of Qualified Person – Dmitry Pertel ..... - 22 -
Certificate of Qualified Person – John Edwards ..... - 24 -
Certificate of Qualified Person – Andrew Pooley ..... - 25 -
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Certificate of Qualified Person – Dmitry Pertel

I, Dmitry Pertel, am employed as the Principal Geologist with CSA Global Pty Ltd with an office address at Level 2, 3 Ord Street, West Perth, Western Australia 6005.

I am an author of this report titled “Dasa Uranium Project Phase 1, Niger, NI 43-101 Technical Report” for the Feasibility Study, prepared for Global Atomic Corporation and with an Effective Date of 15 November 2021 (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 Master’s degree in Geology.

I am a Member of the Australian Institute of Geoscientists (AIG). I have worked as a Geologist for a total of 34 years since my graduation.

My relevant experience for the purpose of the Technical Report is:

Development and reporting of Mineral Resource models

Review and report QAQC procedures and protocols, site visits and laboratory inspections

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) and Section 14, and I have made contributions in Sections 1, 2, 24, 25 and 26, of the Technical Report.

I am independent of the Issuer applying the testset out in Section 1.5.(4) of NI 43-101.

I have prior involvement in the Dasa Uranium Project. I have:

- 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 Global Atomic Corporation and with an Effective Date of 1 June 2019
- Co-authored CSA Global’sreport titled “NI 43-101 Technical Report on the Preliminary Economic Assessment – Dasa Uranium Project, Central Niger”, prepared for Global Atomic Corporation 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 ResourceUpdate, 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 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.

At the Effective Date of this Technical Report, 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 30th day of December 2021 at Perth, Western Australia.

["SIGNED"]

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{Dmitry Pertel}

Dmitry Pertel, M.Sc., MAIG CSA Global 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 report titled “Dasa Uranium Project Phase 1, Niger, NI 43-101 Technical Report” for the Feasibility Study, prepared for Global Atomic Corporation and with an Effective Date of 15 November 2021 (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 36 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 (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 Global Atomic Corporation office in Niamey, Niger.

I am responsible for Sections 13 and 17 of the Technical Report and I have made contributions to Sections 1, 2, 21, 22, and 24 to 26 (inclusive).

I am independent of the Issuer applying the test set out in Section 1.5(4) of NI 43-101.

I have prior involvement in the Dasa Uranium Project. I have 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.

At the Effective Date of this Technical Report, 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 30th day of December 2021 in Johannesburg, South Africa.

[“SIGNED”]

{John Edwards}

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John Edwards

Certificate of Qualified Person – Andrew Pooley

I, Andrew Pooley, am employed as the Managing Director 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 report titled "Dasa Uranium Project Phase 1, Niger, NI 43-101 Technical Report" for the Feasibility Study, prepared for Global Atomic Corporation and with an Effective Date of 15 November 2021 (the "technical report"), 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 27 years since my graduation from university.

My uranium experience relates to 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 (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 I have made contributions to Sections 1, 18, 21, 24 and 26.

I am independent of the Issuer applying the test set out in Section 1.5(4) of NI 43-101.

I have no prior involvement in the Dasa Uranium Project.

I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

At the Effective Date of this Technical Report, to the best of my knowledge, I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission of which would make the Technical Report misleading.

DATED this 30th day of December 2021 in Johannesburg, South Africa.

["SIGNED"]

{Andrew Pooley}

_____

Andrew Pooley

Certificate of Qualified Person – George A Flach, P.Geo.

I, George A. Flach, am employed as the Vice President Exploration at Global Atomic Resources with an office address at 8 King Street East, Suite 1700, Toronto, Ontario, M5C 1B6, Canada (telephone +1 416 368 3949, email: gaflach@globalatomiccorp.com.

I am an author of this report titled “Dasa Uranium Project Phase 1, Niger, NI 43-101 Technical Report” for the Feasibility Study, prepared for Global Atomic Corporation and with an Effective Date of 15 November 2021 (the “technical report”), and do hereby certify that:

I am a registered member (and in good standing) of the Professional Geoscience of Ontario, membership number 0680.

I hold an Honours degree in Earth Sciences - Geology from the University of Waterloo, ON, Canada.

I am familiar with NI 43-101 and, by reason of education, experience in exploration, evaluation and data management, and professional registration; I fulfil the requirements of a Qualified Person as defined in NI 43-101. My experience includes 35 continuous years in the engineering and mining industry.

I have visited the Dasa Project on multiple occasions.

I am responsible for Sections 6, 10 to 12 (inclusive) and 23, and I have made contributions to Sections 1, 2, and 24 to 26 (inclusive) of the Technical Report.

I have prior involvement with the property and issuer.

As of the Effective Date of the Technical Report (15 November 2021), 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.

I am not independent of the Issuer applying all the tests in section 1.5 of NI 43-101, I am a fulltime employee of Global Atomic Corporation.

I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

DATED this 30th day of December 2021 in Takoradi, Ghana

[“SIGNED”]

“George A. Flach”

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George A Flach, P. Geo.

Global Atomic Corporation – VP Exploration

Abbreviations and Units of Measurement

Abbreviation	Description
%	percent
°	degrees
°C	degrees Celsius
3D	three-dimensional
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
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)
CRM	certified reference material
CSA Global	CSA Global Consultants Canada Limited
DCF	discounted cash flow
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)
HPGR	high-pressure grinding rolls
ICP-AES	inductively coupled plasma – atomic emission spectroscopy

Abbreviation	Description
IDW	inverse distance weighted
IMU	inertial measurement unit
IRR	internal rate of return
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 tonnes) per annum
kV	kilovolt
L	litre(s)
lb	pound(s)
LG	low grade (stockpile)
LiDAR	light detection and ranging (survey)
LOI	loss on ignition
LOM	life of mine
LSL	loaded strip liquor solution (OK liquor)
m, m^2, m^3	metre(s), square metre(s), cubic metre(s)
M	million(s)
ml	millilitre(s)
mg, mg/L	milligram(s), milligrams per litre
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
OK	ordinary kriging
ONAREM	Niger National Geological Survey
OPEX	operating expenditure
PEA	preliminary economic assessment
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, QAQC	quality assurance, quality control, quality assurance/quality control
REF	radioactive equilibrium factor
RL	resistivity logging
ROM	run-of mine

Abbreviation	Description
SAG	semi-autogenous grinding
SD	standard deviation
SP	spontaneous polarization
t, tpa, tpd, tph	tonne(s), tonnes per annum, tonnes per day, tonnes per hour
t/m3	tonnes per cubic metre
Th	thorium
U	uranium
UOC	uranium oxide concentrate
US$	United States dollars
UxC	Ux Consulting Company LLC
VAT	value-added tax
VLG	very low grade (stockpile)
XLG	extra low grade (stockpile)
XRD	x-ray diffraction
XRF	x-ray fluorescence

1. Summary

1.1. Principal Outcomes

The Dasa Uranium Project Phase 1 Feasibility Study is based on Global Atomic Corporation processing 4.254 million tonnes uranium-bearing ore grading 5,184 ppm U_3O_8 at the Dasa Deposit. Processing will take place over a 12-year mine plan to produce 45.4 Mlb U_3O_8 of recovered Yellowcake, with an average steady-state metallurgical recovery of 94.15% (overall average 93.4%).

All dollar amounts are stated in United States Dollars throughout this Report. This Report is referred to as Phase 1 of the Dasa Project. 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 4.066 million tonnes uranium-bearing ore grading 5,267 ppm U_3O_8 to result in 47.2 Mlb U_3O_8 Probable Reserves.

The total capital cost carried in the economic model is $345 million, inclusive of $208 million in pre-production capital costs based on a Class 3 AACE standard and $137 million of sustaining capital costs.

Total operating costs for the Phase 1, excluding royalties, are estimated to be $710 million. The average all-in sustaining cost (AISC) is $21.93 / lb U_3O_8

Based on an average selling price of $35 /lb U_3O_8, the after-tax net present value (NPV) at 8% is $157 million, the internal rate of return (IRR) is 22.7%, and the assumed payback period is 3 years once production begins.

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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 2019 MRE (CSA Global, 2019), and the feasibility study completed by METC up to and including 15 November 2021 (the “Effective Date”).

The 2021 Feasibility Report replaces the previous 2020 PEA 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.

In August 2020, Global Atomic Corporation (GAC) appointed METC Engineering Pty LTD (METC) to undertake a feasibility study and NI 43-101 Technical Report (Feasibility Report) on the Dasa Project with the intention to take the project into development in 2022. 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. The primary purpose of this document (the "Report") is to advance the project to a feasibility study level at a AACE Class 3 accuracy prior to moving the project into the development phase. 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. Mr. George A Flach, VP Exploration – GAC, is a non-independent QP responsible for Sections 10, 11 and 12 of this report.

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 Adrar Emoles 3 Exploration Permit, which with the contiguous Adrar Emoles 4 Exploration Permit, are 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 maintains a 100% interest. 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 Adrar Emoles 3 (AE3) Exploration Permit which has a total area of 121.2 km^2. Under NI 43-101 guidelines, the Adrar Emoles 4 (AE4) Exploration Permit, which is contiguous with the southern boundary of the AE3 Exploration Permit and has a total area of 122.4 km^2 and is considered to be the same Property as it would reasonably share common infrastructure should a mineral deposit be developed on either concession.

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 are granted within the provisions of Mining Agreements negotiated between the Ministry of Mines and the applicant. Such an agreement covers a period of up to 20 years, being the exploration period (three years plus two three-year renewals) and the first 10-year validity period of a Mining Permit. The Mining Agreement is then renegotiated at each renewal of the Mining Permit. The Mining Agreement 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.

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

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 also extended on January 23, 2021, for the period ending December 23, 2023.

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 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.

The Mining Permit will be transferred to the Niger mining company upon its incorporation, expected to occur in Q1 2022. 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 August 19, 2021, the Republic of Niger Ministry of Mines provided official notice to GAC that it would not increase its interest beyond the 10% 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.

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 and again in June 2019 (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 x 10 m x 4 m blocks approximating half the drill density in the central parts of the deposit. The estimate has been completed by CSA Global's Principal Resource Geologist, Dmitry Pertel (MAIG) who was an Author and Qualified Person for the Mineral Resource Report.

Information from all main phases of exploration and evaluation and the results of quality assurance/quality control (QAQC) 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 QAQC data, and to inspect the sample preparation laboratory in Niamey.

Review and analysis of both the historical and recent QAQC 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 CSA Global 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 CSA Global. 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 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 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.

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

Table 1-1: Dasa Mineral Resources with an Effective Date of 1 June 2019

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
Total Indicated	26.30	1,752	101.6
Inferred (Open Pit)	18.93	1,357	56.6
Inferred (Underground)	3.38	4,151	31.0
Total Inferred	22.31	1,781	87.6

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 2019 MRE differs in several key areas compared to the previous MREs. The current Mineral Resources were interpreted within a more robust geological and structural model reflecting information collected from the recent drilling programs. All mineralized envelopes were interpreted and controlled by the developed lithological model of the deposit and clipped to the interpreted and modelled known fault planes. Additionally, significant additional XRF chemical analyses (4,983 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 the 2020 preliminary economic assessment (PEA) 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, tailing 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 (15th November 2021)

Mineral Reserve Category	RoM (Mt)	U_3O_8 (ppm)	U_3O_8 (kt)	U_3O_8 (Million lbs)
Proven Mineral Reserve	-	-	-	
Probable Mineral Reserve	4.1	5,267	21.5	47.2
Total Mineral Reserve	4.1	5,267	21.5	47.2

Notes on Mineral Reserves

Mineral Reserves are reported with an effective date of 15th November 2021

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.

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 longhole 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 22.5m vertical intervals with a footwall drive developed along strike 20m from the stopes. Stope access crosscuts will be developed at 16.5m 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 longhole 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 2,074 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: 2021 FS Project summary

Project overview	Unit	Value
Mining		
Total mined	,000's t	4,254
Metal mined	eU_3O_8 M lb	48.6
Average plant feed grade	eU_3O_8 ppm	5,184
Mine life		
Ramp-up	months	23
Years at steady state	years	12
Average production rate	k tpm	33.0
Metallurgical		
Metallurgical recovery	%	93.41
Payable metal	eU_3O_8 M lb	45.4
Payable metal per tonne	eU_3O_8 lbs/t	10.67

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 450-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 largely uninhabited region characterized by nomadic villages and three small towns. 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, 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.

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.

Previous project reports did not indicate 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.

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.

Formal consultation engagement undertaken as part of the 2020 ESIA took the form of a series of meetings in the communities around the project area, including Tagaza, Agatara, Issakanan, Sikiret/Tadant, Oufound, Mizeine, Ghalab, the Kelezeret Tribe and Inolamane.

The 2020 ESIA consultations assisted GAC in the selection of initial priority projects that will be monitored on a regular basis as the project moves into the construction and operations phases. A Stakeholder Engagement Plan will be developed to formalize community engagement and provide a Grievance Mechanism.

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

Table 1-4: Feasibility Study CAPEX Summary

Capital Costs	**Initial ($ M)**	**Sustaining Capital ($ M)**	**Phase 1 Total ($ M)**	**$/lb U_3O_8**	**$/feed tonne**
Mining	54.4	131.5	185.9	4.09	43.70
Processing	87.7	0	87.7	1.93	20.61
Infrastructure	14.6	0	14.6	0.32	3.44
Total Direct Capital Costs	156.7	131.5	288.2	6.35	67.75
Indirect and Owner's Costs	32.7	3.5	36.2	0.80	8.52
Total (including Indirect Costs)	189.4	135.0	324.4	7.14	76.27
Contingency	18.2	0	18.2	0.40	4.28
Reclamation	0	2.4	2.4	0.05	0.56
TOTAL CAPITAL	207.6	137.3	345.0	7.60	81.10

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

The total capital cost for the Dasa mine is $208 million (including $18 for contingencies) with a further $137 million for sustaining costs over the life of mine. The sustaining costs include provision of mine development cost, major equipment replacement and refurbishment. These items will 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 12-11 Process Plant Capital Cost Summary.

Operating costs:

The mining operating costs for the feasibility study project are $6.18/lb U_3O_8 based on an owner-operator model. Ramp and access development are capitalized prior to mineralized material production and expensed as a component of operating costs thereafter. The first two years of mine development will be undertaken by a mine contractor who will be responsible for all underground activities during this period. The mine contractor will have a total compliment of 178 people arranged on a day shift, night shift and “off” shift basis. Shift duration will be 10 hours from start to finish. Similar to the mining contractor, when Dasa personnel take over the operation of the mine, the mine will operate on a day shift, night shift, and an “off” shift basis working 10 days on and 5 days off cycle. The exception to this will be management, and services personnel, who will work 11 days every 2 weeks with alternate Saturdays being working days and every Sunday an off day. Shift duration for management and services personnel will be 8-hour shifts. The allocation of resources per activity are detailed in section 16 of this report.

Average process plant operating costs are calculated to be $6.01/lb or $64/t of mineralized material treated with the largest contributors to operating costs being sulphuric acid and electricity costs. Electricity costs are based on grid power rates supplemented by a 5 MW solar PV system with kWh procured on a producer price agreement basis. The processing facility will be operated and maintained by a staff of 64 people (21 people on permanent day shift and 43 people per shift on a 3-shift rotation basis) giving a total operating complement of 150 people. The shift pattern is 2 X 12 hr shifts with a relief shift. The day shift people will work either a 10 hour shift 5 days a week for plant operations or a 9 hour shift X 5 days per week for administrative and management personnel. The process plant will operate 24 hours per day, 365 days per year The Operating Cost Estimate Summary is presented in the table below.

Description	Phase 1 Cost ($ million)	Average Annual ($ million)	Unit Cost ($/t processed)	Unit Cost ($/lb U_3O_8)
Mining Cost	280.4	22.8	65.9	6.18
Processing Cost	272.9	22.2	64.2	6.01
General and Administration	157.4	12.8	36.9	3.46
Total Cash Costs Before Royalties	**710.7**	**57.8**	**167.0**	**15.64**

Notes:

1 – Total may not sum due to rounding
2 – Average annual costs based on 12.3 years

General and administration (G&A) costs include a 450-person camp and facilities, camp staffing and offsite costs. G&A and offsite cash operating cost totals $3.46/lb U_3O_8 ($37/t processed).

Including sustaining capital and royalty costs, the all-in sustaining costs (AISC) is $21.93/lb U_3O_8 ($234.14/t processed).

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 22-5. 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 mine plan.

Assumptions are as follows:

The sale price for U_3O_8 is $35/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 2021 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 the start of first construction.

Revenue is estimated from the production plan and the $35/lb U_3O_8 Base Case estimate of the price received for the product. The mine production has been forecast to begin 17 months from the start, while the first product from the plant is forecast to begin 20 months from the start. The plant recovery is estimated

to be 94.15% at steady state. The plant is assumed to have an operating capacity of 1 000 t/d of run-of-mine feed. 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. The operating profitability is determined as a ratio of EBIT to revenues, where EBIT refers to earnings before interest and taxation. The royalty is deductible for the determination of corporate income tax.

The sliding scale is as follows:

If operating profitability < 20%	mining royalty rate is 5.5%
If operating profitability > 20% and < 50%	mining royalty rate is 9.0%
If operating profitability > 50%	mining royalty rate is 12.0%

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 $65.9/t mined and the processing costs are $64.1/t milled.

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 $47.5 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. Once production commences, the VAT rate is set at 0%.

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 1-5: Initial and sustaining capital expenditures

Item	Amount, 000 $
Mining - project	61,889
Processing - project	143,151
TSF - project	2,597
Total Initial Capital Expenditure	**207,637**
Sustaining and closure costs	137,347
Total Initial plus Sustaining Capital expenditure	**344,985**

1.15. Economic Results at $35/lb U_3O_8

The forecast of the economics over the Phase 1 Dasa Project are shown in Table 22-3 and the financial indicators are shown in Table 22-4.

The NPV_8 is $157 M, and IRR is 22.7% for the Base Case estimates. The cash flows by year are summarized in Table 22-5

Table 1-6: Forecast of the project economics over the Phase 1 Dasa Project.

Item	Phase 1 Total, $M
Revenue	1,589
Mining royalties	148
Operating costs	710
Operating income	**731**
Taxation	54
Capital costs	345
Free cash flow	**332**

Note: values may not add up due to rounding.

Table 1-7: Financial indicators over the Phase 1 Dasa Project.

Item	Unit	Value
NPV_8 after tax	$M	$157
IRR after tax	%	22.7%
Payback period	years	3
Cash flow (net of capex)	$M	$677
Free cash flow	$M	$332
Operating costs		
Cash cost including royalties	$/lb U_3O_8	18.91
All-in sustaining cost	$/lb U_3O_8	21.93
Capital costs		
Initial capital costs	$M	$208
Sustaining capital costs	$M	$137

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 22.5.4 of this report – the model is most sensitive to the price of U_3O_8. as shown in the following table:

Table 1-8: Economic Sensitivity with Varying Price of U_3O_8

U_3O_8 price (per pound)	**$35**	**$40**	**$50**	**$60**
Before-tax NPV_8	$187M	$309M	$556M	$804M
After-tax NPV_8	$157M	$259M	$468M	$676M
After-tax IRR	22.7%	30.6%	44.6%	57.2%

The Phase 1 Dasa Project production takes place over 12 years at full production plus some additional at either end. The initial Phase 1 production is from the very high-grade parts of the Flank Zone grading 6,464 ppm over the first 7 years to produce an average 4.8 million pounds per annum U_3O_8. Production of such high-grade ore results in a low cash cost of $12.36/lb U_3O_8 (before royalties). In the latter 5 years, production grades are 3,300 ppm to result in an average annual production of 2.3 million pounds per annum U_3O_8. Production of the medium grade ore results in a cash cost of $24.20/lb U_3O_8 (before royalties). The two distinct production periods are summarized in Tables 1-9 and 1-10.

Table 1-9: Production Metrics throughout Phase 1

	Production		
	Year 2 to 8	**Year 9 to 13**	**Phase 1[(1)]**
Tonnes Processed	2 514 849	1 681 099	4 253 626
Grade (ppm)	6 464	3 300	5 184
U_3O_8 Pounds Produced	33 553 851	11 514 987	45 406 166
Average Annual U_3O_8 Pounds	4 793 407	2 302 997	

Table 1-10 Production Costs throughout Phase 1

Operating Costs	$/lb U_3O_8 Recovered			$/tonne
	Year 2 to 8	Year 9 to 13	Phase 1(1)	
Mining Cost	4.71	10.11	6.18	65.92
Processing Cost	4.81	9.33	6.01	64.13
Overhead Cost	2.83	4.75	3.46	36.93
Cash Cost Before Royalties	**12.36**	**24.20**	**15.64**	**166.98**
Royalties	3.67	2.14	3.27	34.87
Total Cash Cost	**16.02**	**26.34**	**18.91**	**201.85**
Sustaining Capital	2.62	2.33	3.02	32.29
AISC	**18.65**	**28.67**	**21.93**	**234.14**

It is expected that long-term output may be maintained either by increased high grade tonnage by extension drilling of the Flank Zone or expansion of plant throughput to accommodate a greater volume of medium grade ore.

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 in the Flank Zone and at depth within the graben.
- The cut-off grade for the Phase 1 mine plan was based on a U3O8 price of $35/lb; use of a higher price would likely increase the economically mineable tonnes and the overall reserve quantum.
- Process plant availability was set at 86% based on estimated power interruptions; inclusion of a full power back-up system would increase plant availability likely in excess of 90%.
- 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 mine development.
- The current mine design can be further optimized which would translate into lower costs.
- Opportunities exist to further improve the size of the mill precipitation circuit which would result in cost savings.

Recommendations

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

In addition, relating to further exploration and development for the long term on the Adrar 3 and Adrar 4 Exploration Permit areas, the Qualified Persons and METC Engineering recommend the following be completed to support ongoing exploration and higher-level engineering studies at the project:

Further exploration and evaluation programs could upgrade the confidence of the extent and quality of mineralization at the deeper parts of the Dasa deposit (inside the graben). Additional, successful infill drilling would allow an upgrade in resource classification.

Conduct extension drilling, targeting the main Flank Zone fault along strike.

Complete more work and structural drilling that will support defining all structures and potential faults within the mining area

2. Introduction

2.1. Introduction

Mr. Dmitry Pertel, M.Sc., MAIG, Mr. Andrew Pooley, BEng (Hons) Mining, Mr. John Edwards, BSc (Hons) Mineral Processing Technology, and Mr. George A Flach, P. Geo. 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

Global Atomic Corporation (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. 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 15 November 2021 entitled "Phase 1 Dasa Project Feasibility Study and Issues a Maiden Mining Reserve".

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.

METC Engineering (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. The primary purpose of this document (the "Report") is to advance the project to a feasibility study level at a AACE Class 3 accuracy prior to moving the project into an execution phase.

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 2019 MRE (CSA Global, 2019), and the feasibility study completed by METC up to and including 15 November 2021 (the "Effective Date").

The 2021 Feasibility Report replaces the previous 2020 PEA 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 PEA was previously reported in a NI 43-101 Technical Report with an effective date of 1 June 2019 (CSA Global, 2019) 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. Mr. George A Flach, VP Exploration – GAC, is a non-independent QP responsible for Sections 6, 10-12 & 23 of this report.

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. Mr. George A Flach, VP Exploration – GAC, is a non-independent QP.

Table 2-1: Qualified Person section responsibility

Section	Section title	Qualified Person(s)
1	Summary	All
2	Introduction	All
3	Reliance on Other Experts	None
4	Property Description and Location	None
5	Accessibility, Climate, Local Resources, Infrastructure and Physiography	None
6	History	George A Flach
7	Geological Setting and Mineralization	Dmitry Pertel
8	Deposit Types	Dmitry Pertel
9	Exploration	Dmitry Pertel
10	Drilling	George A Flach
11	Sample Preparation, Analyses and Security	George A Flach
12	Data Verification	George A Flach
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	None
20	Environmental Studies, Permitting, and Social or Community Impact	None
21	Capital and Operating Costs	John Edwards and Andrew Pooley
22	Economic Analysis	John Edwards and Andrew Pooley
23	Adjacent Properties	George A Flach

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 (QAQC) 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

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 as reported in the PEA undertaken by CSA Global in 2020. 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.

The primary dataset used to inform the Mineral Resource is the digital drillhole database provided by GAC and the subsequent review by the CSA Global QP who reviewed the data, completed relevant QAQC checks and is satisfied the data is adequate for estimation of Mineral Resources.

Process test work by Insight R&D Reports as detailed below:

- 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, 2020
- Insight R&D, Dasa Pilot Plant Campaign 3, Insight Project Report Number 2006-03-3-R0, February 4, 2020
- Insight R&D, Dasa Pilot 1st Filtration Tests, Insight Project Report Number 2010-01-1R0, April 8, 2021

- Insight R&D, Tailings Density Report, Jan 20, 2021
- Insight R&D, Dasa Pilot 1st Filtration Tests, Insight Project Number 2010-01-1-R0, April 8, 2021
- Insight R&D, Dasa Leach Filtration, Second Test Campaign, Insight Project Number 2103-01-1-R0, July 2021
- Insight R&D, Yellow Cake Production Route, Insight Project Report Number 2103-01-2-R0, July 28, 2021
- Insight R&D, Insight SDU Filtration Test Results, June 29, 2021, Spreadsheet Report
- Insight R&D, Insight RND Tailings Density Report Jan 20-2021 Spreadsheet Report
- Insight R&D, Insight Yellow Cake Settling Density Sieve Comp Feb 28 21 Spreadsheet Report

The authors have taken reasonable steps to verify the information provided where possible.

3. Reliance on Other Experts

Section 4.2 of this report (Mineral Tenure) – ownership information has been provided by Global Atomic Corporation - the reliability of the information has not been interrogated.

Section 4.2 of this report (Permitting Considerations) – the permitting status has been provided by Global Atomic Corporation – the accuracy and reliability of the information has not been interrogated.

Section 13.5 of this report (Feasibility Study Test Work (2020 – 2021)) – was undertaken in Canada, the QP did not visit the facility, however the reports published and referenced in section 13.5 have been reviewed and scrutinised to the extent possible in the feasibility study and the QP does not have reason to challenge the outcomes as reported.

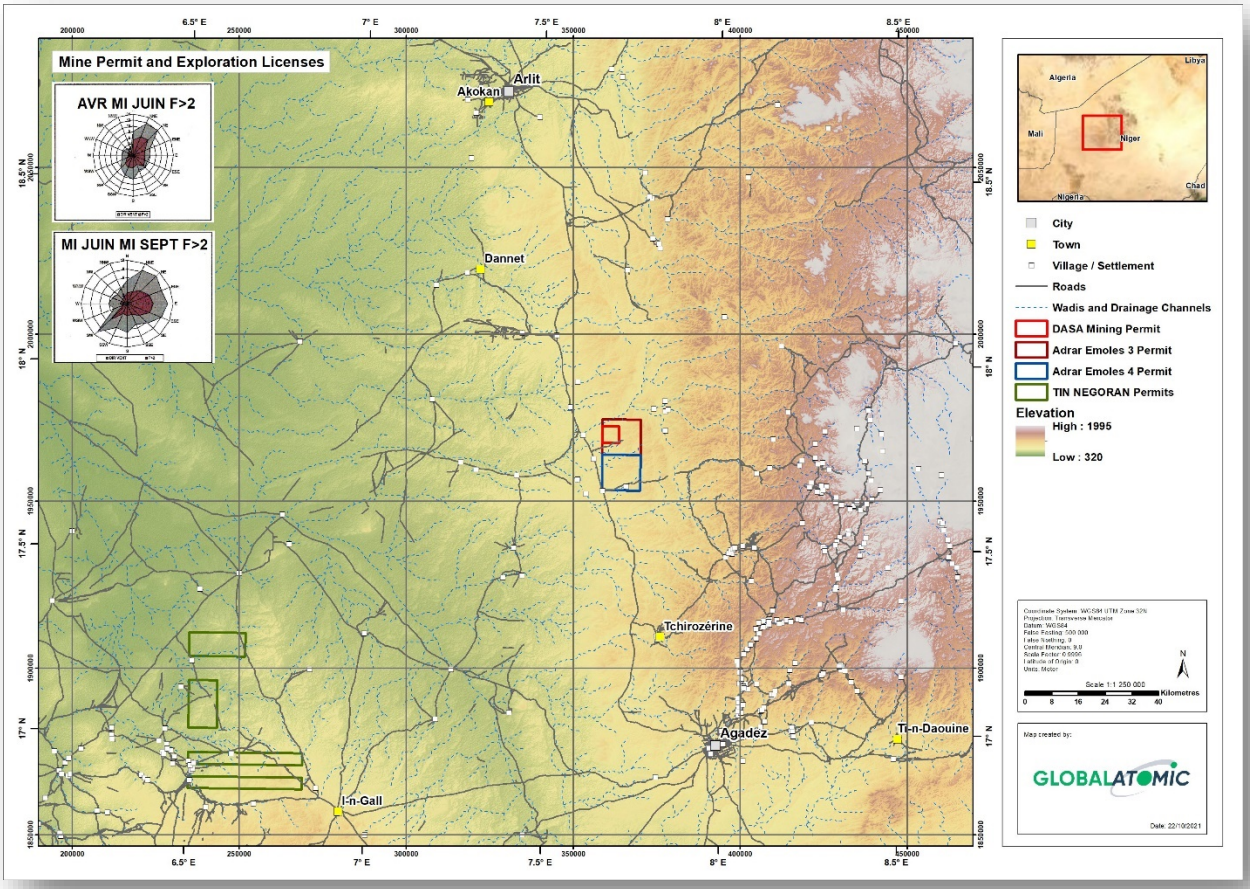
4. Property Description and Location

4.1. Location of Property

GAC's exploration operations are located in the north central part of the Republic of Niger (Figure 4-1), 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.

The Dasa Project is located in the southwest of the Adrar Emoles 3 (AE3) Exploration Permit which itself has a current total area of 121.2 km^2. The centre of the Dasa Project is positioned at longitude 7.8° east and latitude 17.8° north. Under NI 43-101 guidelines, the Adrar Emoles 4 (AE4) Exploration Permit, which is contiguous with the southern boundary of the AE3 Permit and has a total area of 122.4 km^2, is considered to be the same Property as it would reasonably share common infrastructure should a mineral deposit be developed on either concession, Table 4-1).

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



4.2. Mineral Tenure

Exploration Permits and Mining Permits are granted within the provisions of a Mining Agreement negotiated between the Ministry of Mines and the applicant. Such an agreement covers a period of up to 20 years, being the exploration period (three years plus two three-year renewals) and the first 10-year validity period of a Mining Permit. The Mining Agreement is then renegotiated at each renewal of the Mining Permit. The Mining Agreement 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.

GAC, through its 100% owned subsidiary GAFC, entered into and holds a 100% interest in two Mining Agreements 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.

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 23, 2021, for the period ending December 23, 2023.

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 23, 2021, for the period ending December 23, 2023.

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.

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.

The Mining Permit will be transferred to the Niger mining company upon its incorporation, expected to occur in Q1 2022. 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 August 19, 2021, the Republic of Niger Ministry of Mines provided official notice to GAC that it would not increase its interest beyond the 10% carried interest.

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 is removed from the perimeter of the AE3 Exploration Permit. On January 21, 2021, the Exploration Permits for the remaining area of AE3 and AE4 were extended for a further period to December 17, 2023.

Exploration expenditures and corporate overhead costs incurred in Niger related to the Dasa Deposit by GAFC to the date of formation of the Niger mining company will be transferred to the Niger mining company. Subject to negotiation with the Government of the Republic of Niger, such amounts will be reimbursed to GAC as a priority distribution from Dasa Mine operations. As of September 30, 2021, the amount of such cumulative expenditures was $47.5 million.

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: GAFC Date granted: 23/12/2020 Valid: 10 years subject to 5 year renewals until Deposit is depleted. Area: 25.01 km^2			Tenement type: Exploration Company: GAFC Date granted: 21/01/2021 Valid: to 23/12/2023 Area: 121.2 km^2			Tenement type: Exploration Company: GAFC Date granted: 21/01/2021 Valid: to 23/12/2023 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. Permitting Considerations

In 2020 GAC retained Groupe Art & Genie to conduct an Environmental and Social Impact Assessment (“ESIA”) specific to the Dasa Project Area associated with the Phase I Mine Plan subject of this Feasibility

Report. The ESIA updated previous environmental and socio-economic base-line information in support of the Company's Mining Permit application for the Dasa Project. GAC engaged with the Ministry of Environment, Urban Health and Sustainable Development ("MESUDD") to confirm the Terms of Reference and scope of both the ESIA and the Environment and Social Management Plan ("ESMP") at the start of the ESIA process. The MESUDD and its constituent agencies are responsible for developing guidelines for the ESIA, analysis and evaluation of the ESIA, validation of the ESIA through stakeholder meetings and the negotiation of the ESMP.

The 2020 ESIA serves to update the ECS completed by Groupe Art & Genie in 2011 and includes hydrological and air quality baseline data compiled during the interim period and addresses environmental, social, and economic impacts associated with construction of the mine, mining operations and reclamation and closure. The ESIA includes an analysis of alternative design and operating scenarios in order to reduce environmental and social impacts associated with the Mine.

The ESIA included public consultation to inform stakeholders of the proposed project, and technical studies related thereto. Community meetings were held in area nomadic villages and small towns as part of the ESIA process and were organised by MESUDD and attended by Government and GAC personnel. A copy of the ESIA is kept at the MESUDD to enable stakeholders the opportunity to access and review the documentation. 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.

An important component of the ESIA is the Environmental and Social Management Plan ("ESMP"). The ESMP governs the Company's activities and defines the protocols for Project monitoring and reporting to government ministries and agencies from construction through operations and mine closure. The ESMP also outlines the management procedures, mitigation, monitoring and reporting protocols GAC will implement to avoid, reduce, and manage environmental and social impacts during the construction, operations and closure phases of the Dasa project.

The ESIA was completed and submitted to the MESUDD for review and approval in October 2020 in accordance with Law No. 98-56 of December 29, 1998, on the framework law on environmental management and Law No. 2018-28 of May 14, 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 is issued for an initial period of ten (10) years subject to automatic five (5) year renewals until the deposit is depleted.

GAC also entered into two agreements with the National Office of Environmental Assessment ("BNEA") in December 2020; the "Partnership Agreement" which establishes the framework for the implementation and monitoring of the ESMP, capacity building and budgets for the relevant ministries and agencies and, the

"Environment and Social Charges Book" which establishes implementation, monitoring and reporting protocols related to the implementation of 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.

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

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

The sectors which have received support since 2008 are detailed in Table 4-2 below. To date, education and training has been limited to employees of the project and annual training contributions to the Ministry of Mines.

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.

Table 4-2 also indicates an anticipated increased level of support and the introduction of new programs during the construction and underground development phases in the period 2022 - 2024. 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 identified in consultation with communities.

Table 4-2: Summary of community support initiatives undertaken and planned by GAC

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

Formal consultation engagement undertaken as part of the 2020 ESIA took the form of a series of meetings in the communities around the project area, including Tagaza, Agatara, Issakanan, Sikiret/Tadant, Oufound, Mizeine, Ghalab, the Kelezeret Tribe and Inolamane.

Environmental concerns included potential effects of uranium mining, contamination of the food chain, human and animal health risks; occupation and loss of pastoral areas and crop lands, vegetation,

and loss of wildlife habitat; impacts to water resources, management of waste from mining operations and restoration and rehabilitation of the mine.

Social concerns included population displacement, marginalization of local communities, the security of cultural and tourist sites, employment opportunities for young people from the local communities and management of labour risks.

Community development concerns included youth training and jobs for young people (8 villages), water wells (7 villages), sanitation (6 villages), construction of a health post (4 villages), market gardening (3 villages), business opportunities and a food bank (2 villages) and support for animal health and protection of the koris (1 village).

The 2020 ESIA consultations assisted GAC in the selection of initial priority projects that will be monitored on a regular basis as the project moves into the construction and operations phases. A Stakeholder Engagement Plan will be developed to formalize community engagement and provide a Grievance Mechanism.

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 GAC exploration “Dajy” 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.

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



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 daily scheduled connections and at one time also handled international tourist flights from Europe.

Arlit also has an airport with a 2,000m 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 1970s.

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, it is believed that there is sufficient capacity for the Sonichar plant to provide enough power to 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 can be expected to be 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 1950s 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 1960s, 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-1960s.

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 (Figure 4-2). 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 world class 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 x 800 m and 400 m x 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 x 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 (Figure 4-2), 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 (Figure 4-2) 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 (Figure 4-2). 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 (Figure 42).

Within the AE3 and AE4 concessions, mineralization was known to exist within 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 x 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.

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

A 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). 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). 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 (Section 14 of this Report).

6.6. Historical 2018 Preliminary Economic Assessment

CSA Global completed a PEA of the Dasa Project in 2018 (CSA Global, 2018b). The 2018 Dasa PEA was based on CSA Global's 2018 Dasa MRE which has now been replaced by the 2019 Dasa Mineral Resource update presented in this Report. The 2019 Mineral Resource update differs materially from the 2018 Mineral Resource and as such, GAC and CSA Global no longer considered the 2018 PEA current and it is replaced by the 2020 PEA Update presented in this Report.

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.

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 has been reported on since the 1950's with several reports published (a listing of these reports can be found at the end of Section 7.7 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 110,000 tonnes of uranium. Orano (2019a) reported production of 1,128 tonnes of uranium from Cominak (presently closed) and 1,783 tonnes of uranium from Somaïr during 2018.

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.

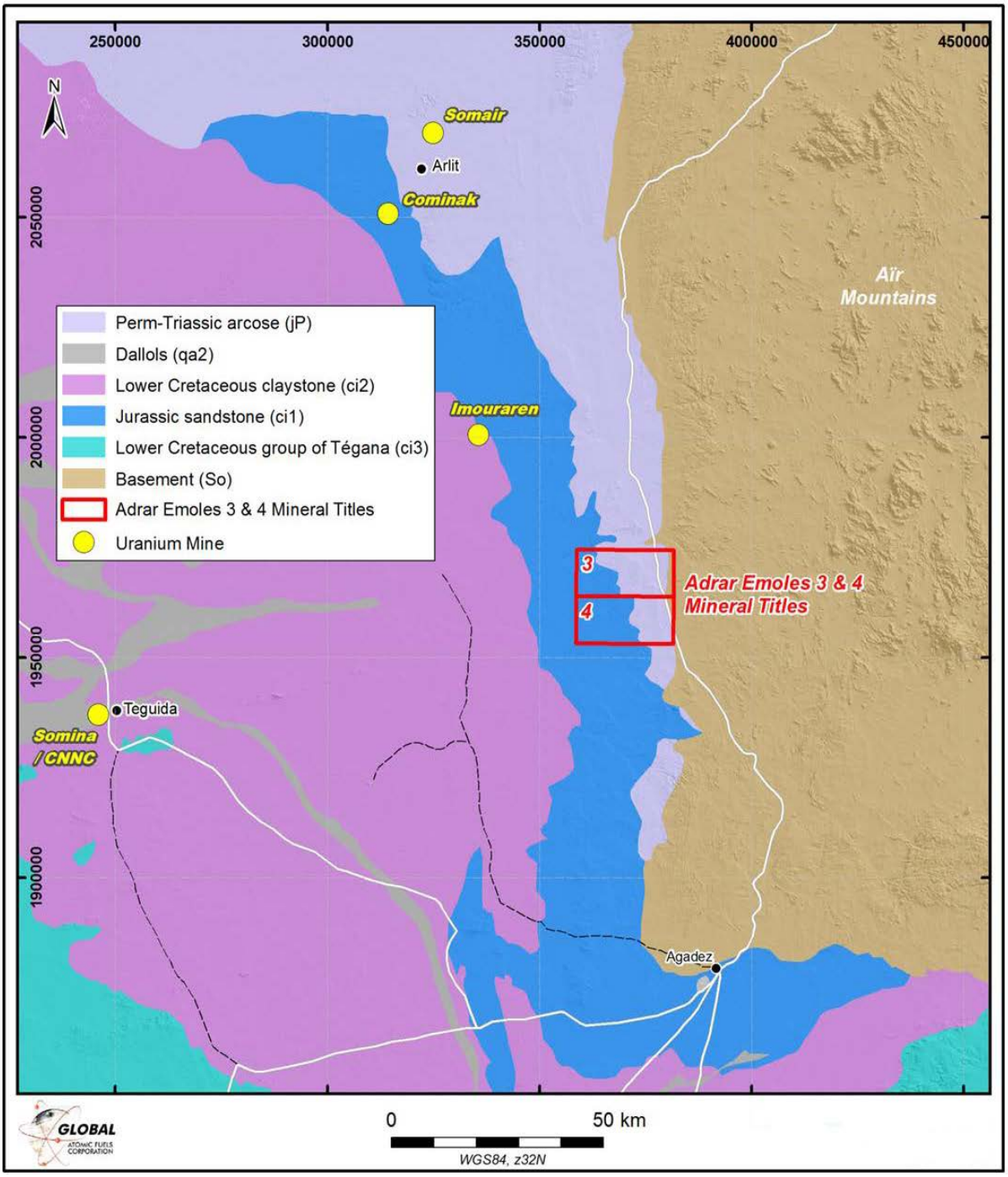


Figure 7-1: Regional geology map

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

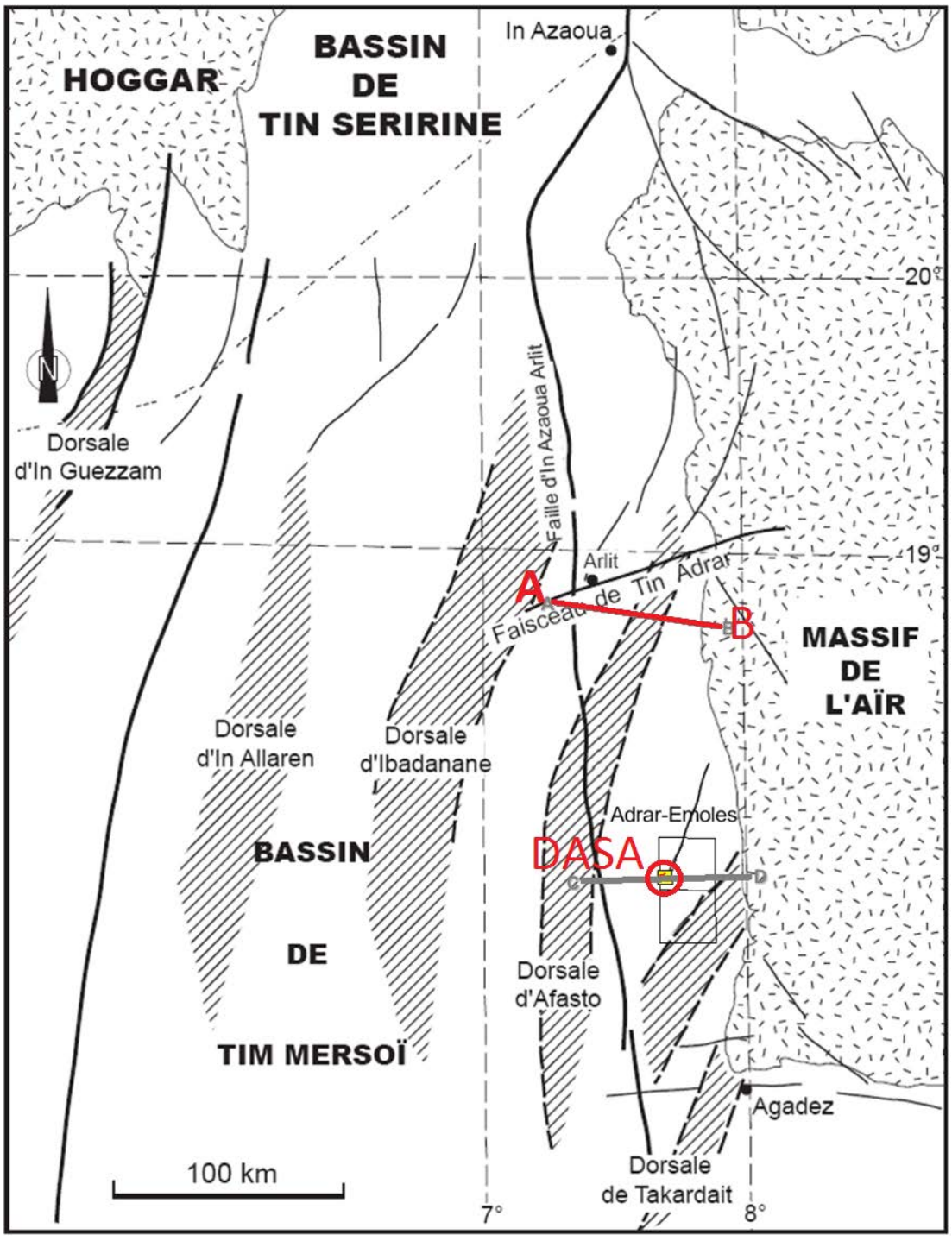


Figure 7-2: Major structures in the Tim Mersoï Basin, shaping it in a succession of ridges and basins

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 intrusives 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.

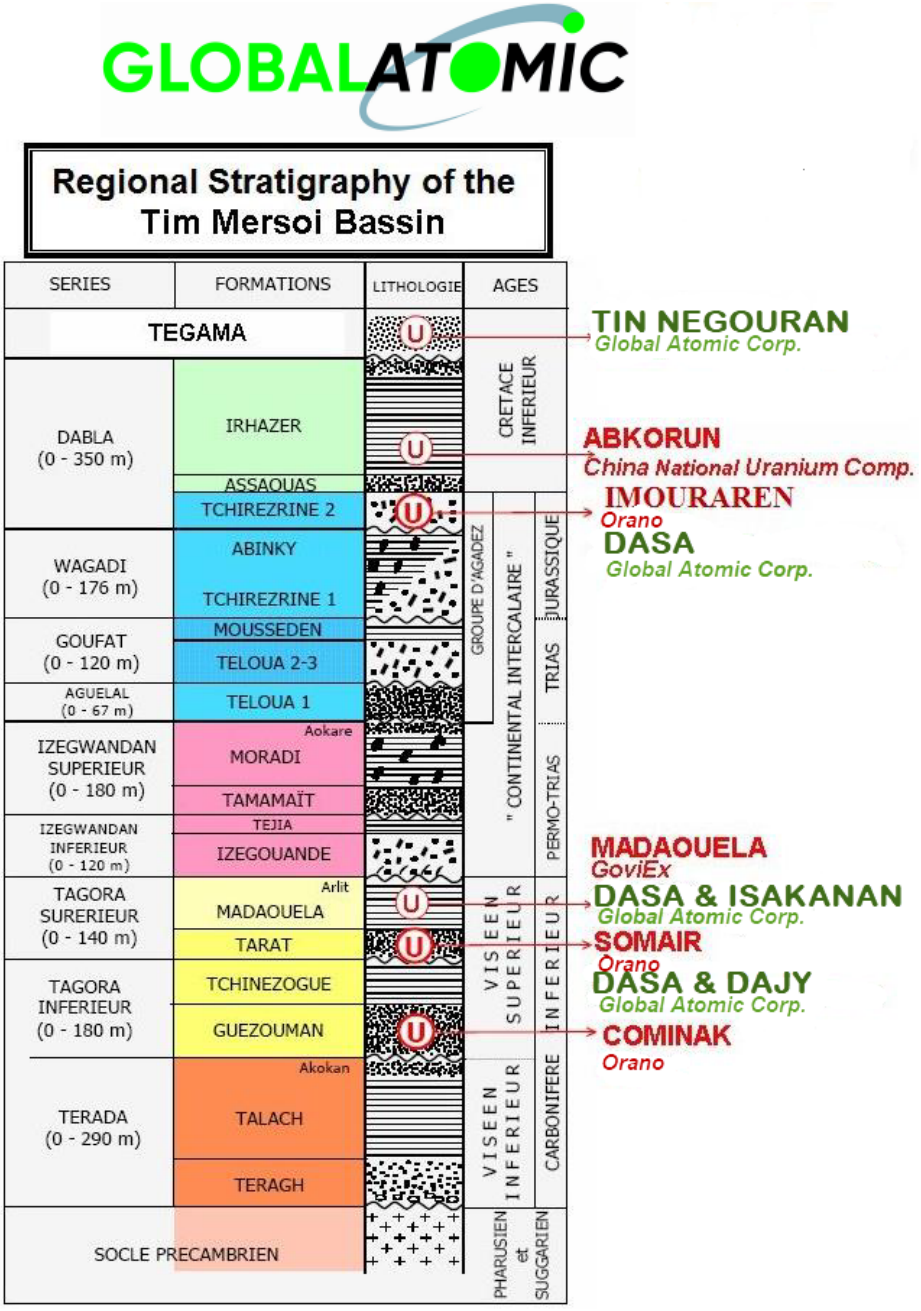


Figure 7-3: Stratigraphic column of the series and formations in the southern part of the Tim Mersoï Basin and including the Dasa Project and Property

Source: GAC

There are a number of 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).

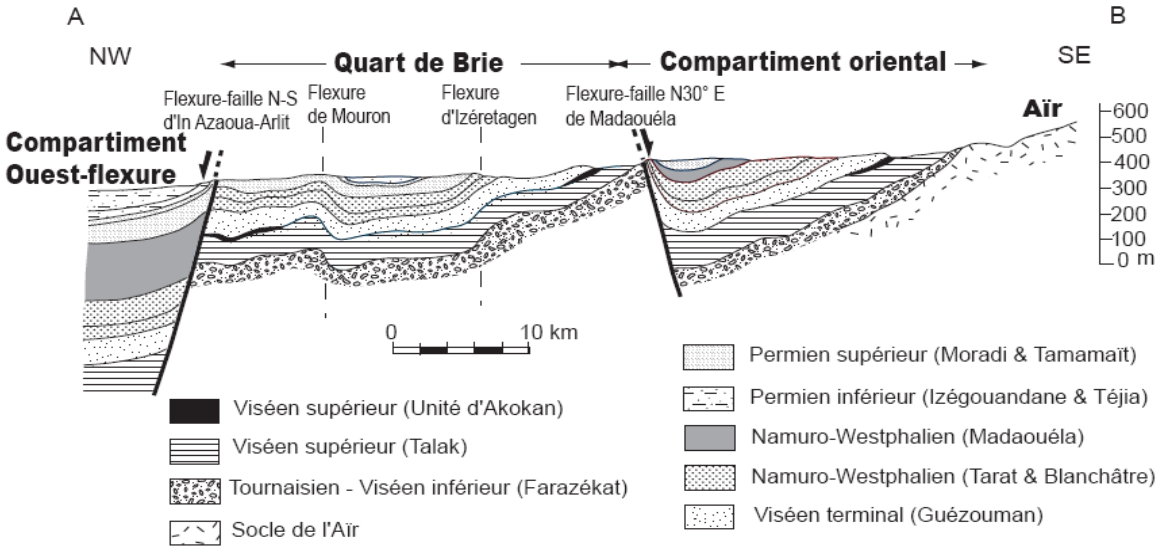


Figure 7-4: Northwest-southeast profile in the Arlit area (see Section A-B in Figure 7-2)

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		Equigranular sandstones medium to conglomeratic with iron stains	Lacustrine with fluvial intercalations		
	Moradi		reddish argillites and very fine grained sand lenses carbonatic cemented Arkosic channels strongly oxidized Conglomerates with quart zand clay pebbles			
	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

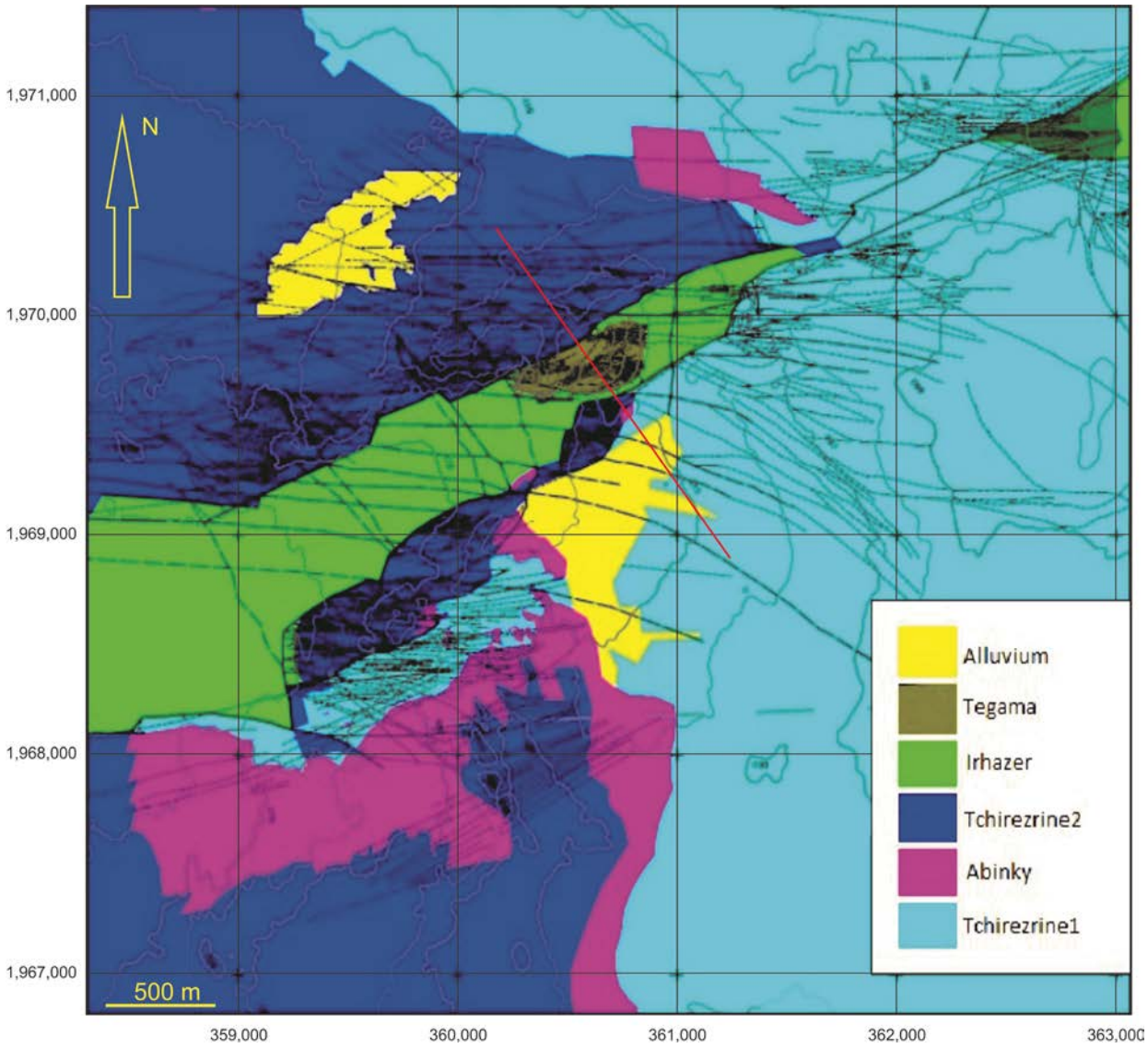


Figure 7-6: Dasa structural map (Figure 7-7 section line in red)

Source: GAC

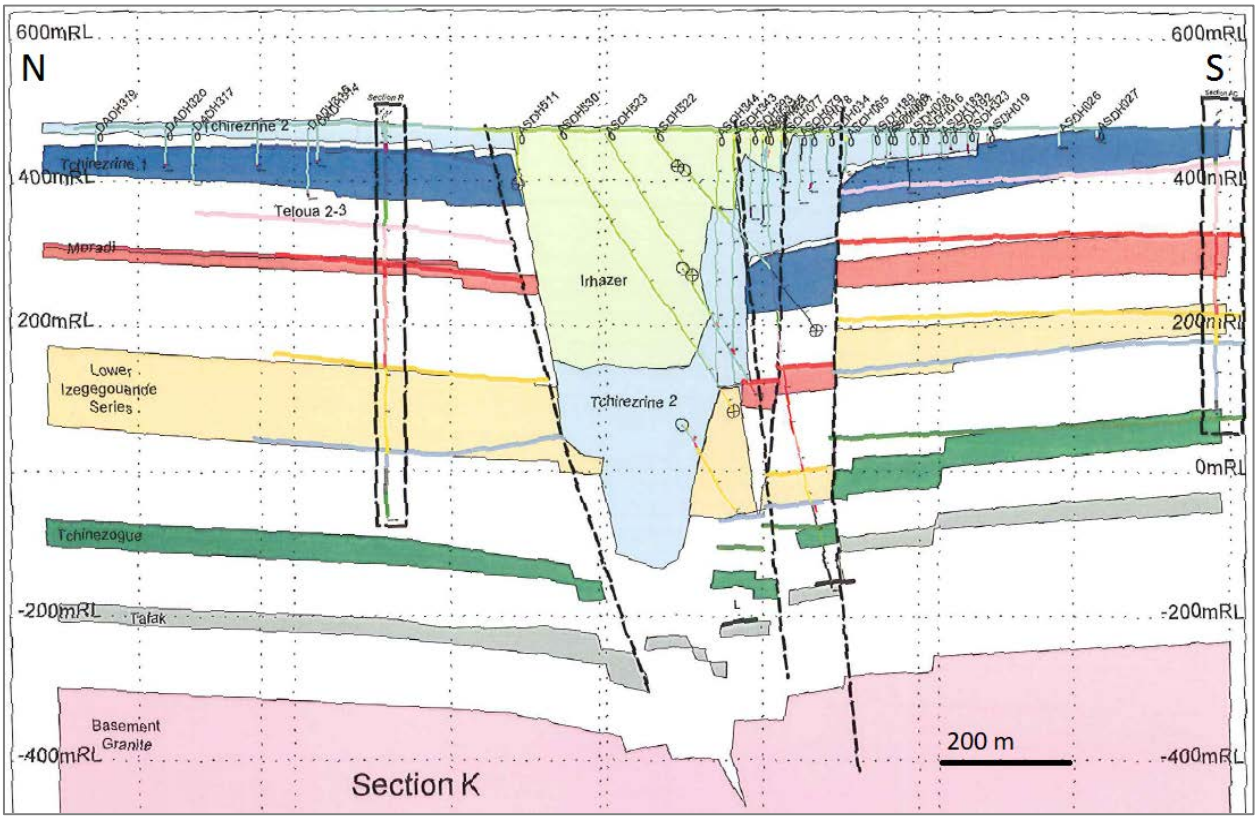


Figure 7-7: Dasa schematic geological section (see red section line on Figure 7-6)

Source: GAC

Subsections 7.4.1 to 7.4.10 (below) provide a brief 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 at this time.

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.

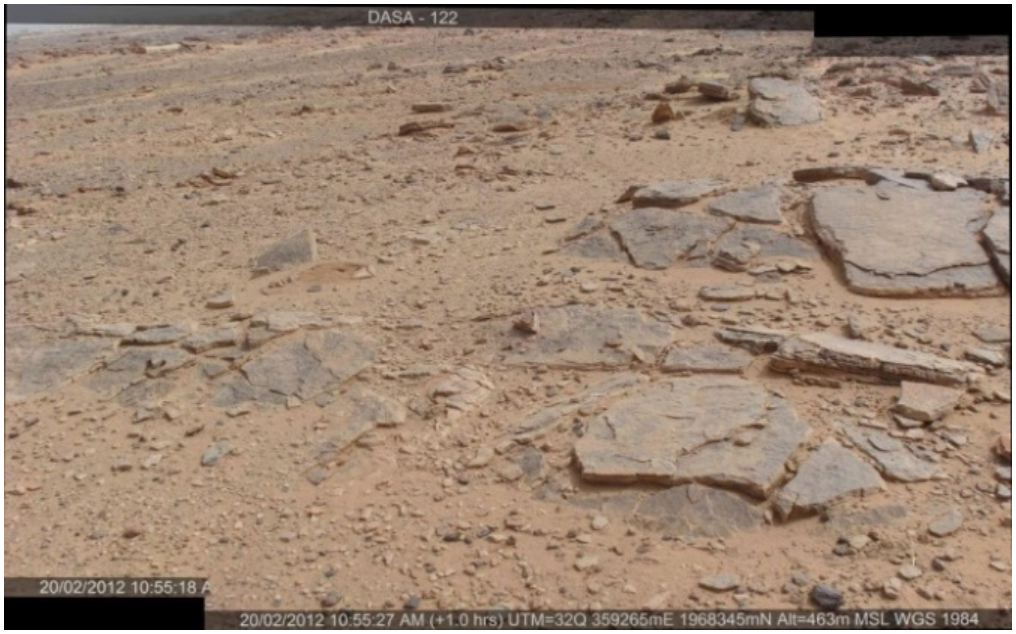


Figure 7-13: Siltstone outcrop, Assaouas Formation, southern outcrops

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, Figure 7-16).



Figure 7-15: Heavily quartz veined Tegama sandstone; mount inside the Assouza Graben

Source: GAC

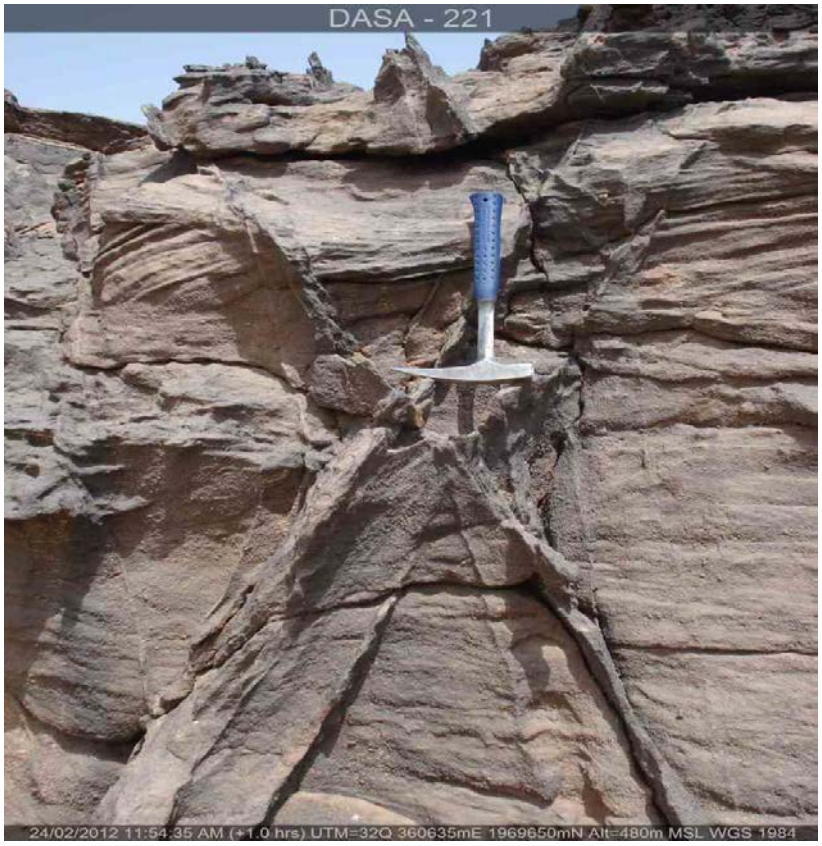


Figure 7-16: Conjugate fracture veined in quartz in coarse cross-bedded Tegama sandstone

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 1980s 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 at 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:

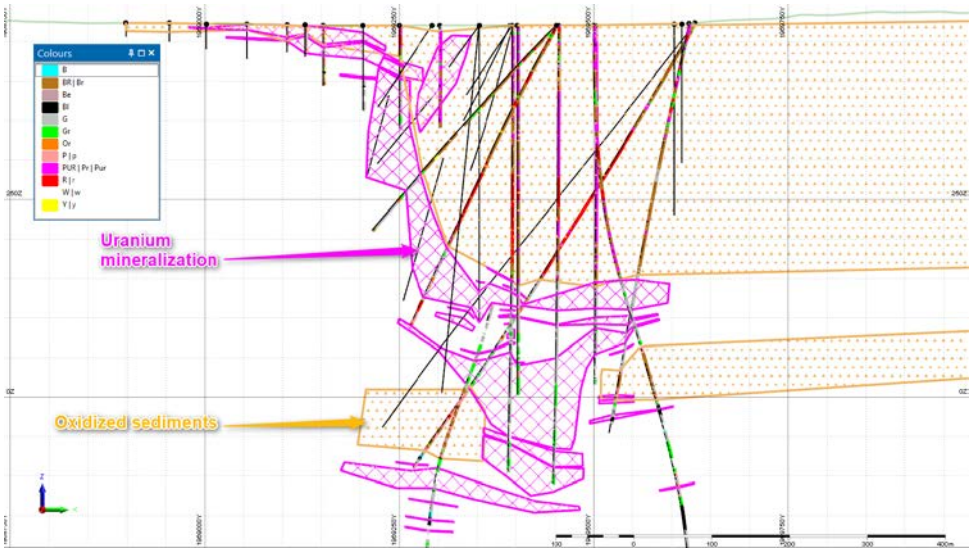
There is a strong control by stratigraphy and lithology – with mineralization mainly hosted within the Tchirezrine 2 sandstones, particularly in the coarser-grained micro-conglomeratic facies of greyish-greenish colour containing frequent sulphides and organic matter such as plant remains.

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 work 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)

Cogema (1977a, 1977b)

Jean Martin von Siebenthal (2013)

Cazoulat (1985)

Gauthier (1972, 1974)

Gerbaud (2006)

https://www.uni-hohenheim.de/atlas308/startpages/page2/french/content_fr/conframe_fr.htm

Guiraud (1981)

Greigert and Pougnet (1967)

Hirlemann and Robert (1977, 1980)

Hirlemann and Faure (1978)

Joulia (1957, 1959, 1963)

Joulia and Obellianne (1976)

Konaté et al. (2007)

Lang et al. (1991)

Molebale (2012)

Sempéré (1981)

Valsardieu (1971)

Wright (1993, 2010, 2012)

Yahaya (1992)

Yahaya and Lang (2000).

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

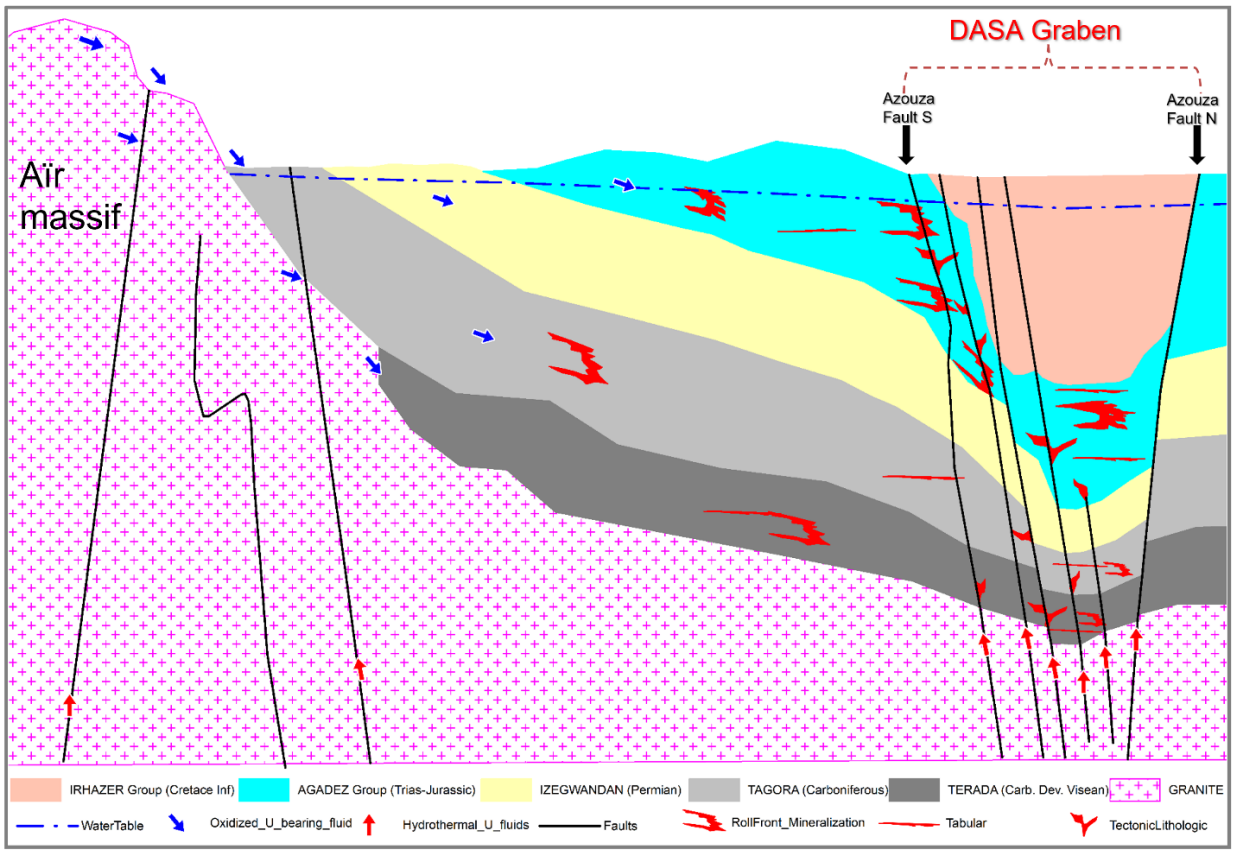


Figure 8-1: Global Atomic Corp DASA Uranium Genetic Model, November 2021 (graphic, not to scale)

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, acquired the Tin Negoran 1, 2, 3 and 4 Mining Agreements 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 Agreements. 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.

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 North East	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 North East	Northeast of actual Dasa deposit
G096	361165	1969340	4,467	412	Dajy	South of actual Dasa deposit
G097	362183	1971953	2,811	475	Azouza North East	Northeast of actual Dasa deposit
G120	361256	1969305	5,417	428	Dajy	South of actual Dasa deposit
G129	362697	1970250	2,360	421	Azouza North East	Northeast of actual Dasa deposit
G130	365843	1972250	2,327	276	Azouza North East	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 3D 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 x 100 m; 100 m x 50 m; to 25 m x 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 x 100 m; 50 m x 50 m; to 25 m x 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 x 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 WGS84/32N)	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).

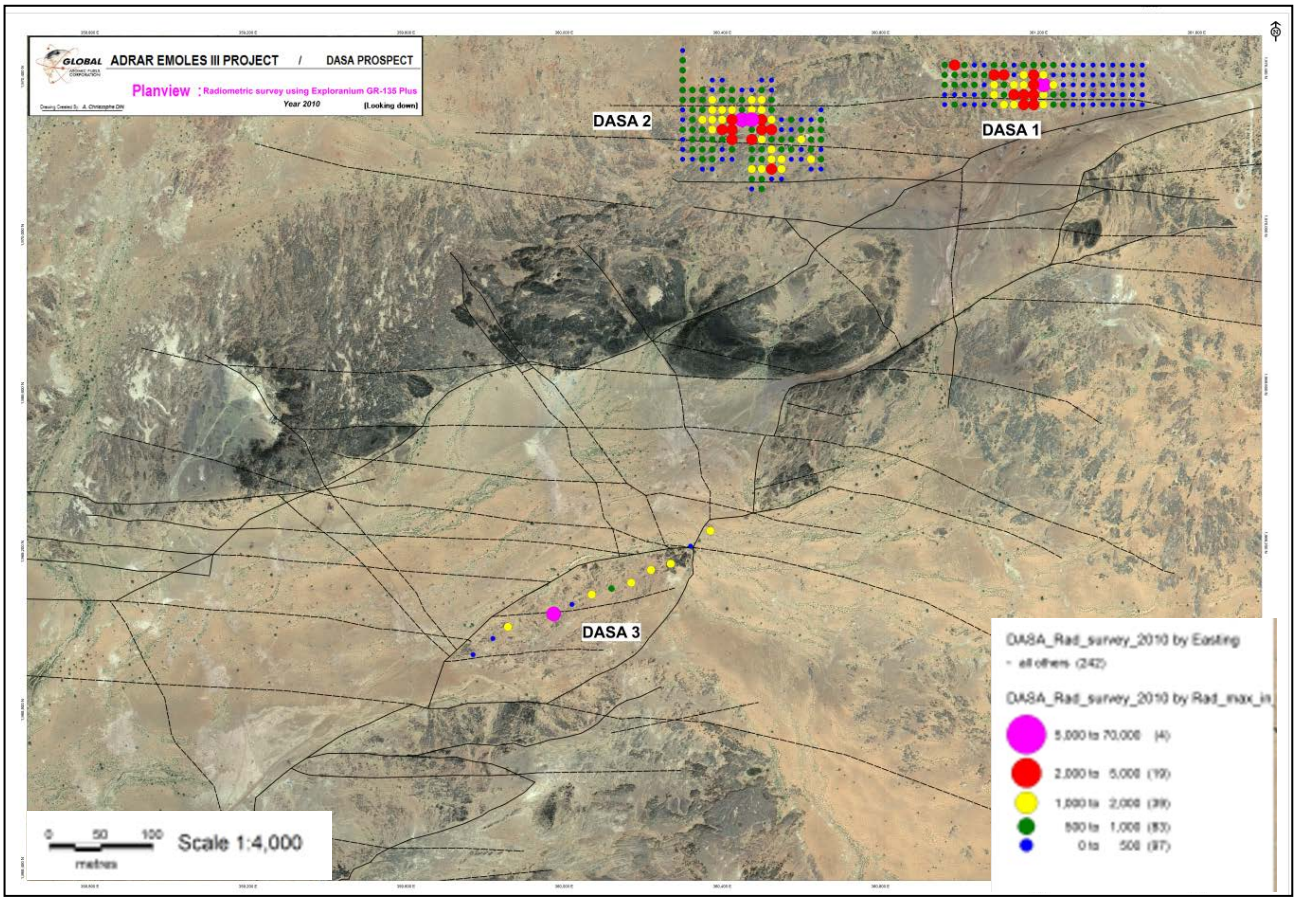


Figure 9-1: Radiometric sampling points at the Dasa Project, AE3 Concession

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 (3D) 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 in order 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 through the use of 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 in order 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, 1,029 holes (Figure 10-1) including 873 rotary holes and 156 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	1,579
2011	607	38,381	18	986	625	39,367
2012	197	36,504	41	6,251	238	42,755
2013	17	10,734	28	16,621	45	27,355
2014	0	0	12	8,064	12	8,064
2015	0	0	1	501	1	501
2017–2018	6	3,632	53	22,846	59	26,479
Total	873	90,393	156	55,706	1,029	146,100

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 59 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 26,479 m. The average drilling depth was 415 m.

The 2010 to 2018 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.

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

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

Zone	Hole	From (m) – To (m)	Length (m)	Grade (ppm/% eU_3O_8)
	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
	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

Zone	Hole	From (m) – To (m)	Length (m)	Grade (ppm/% eU_3O_8)
	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

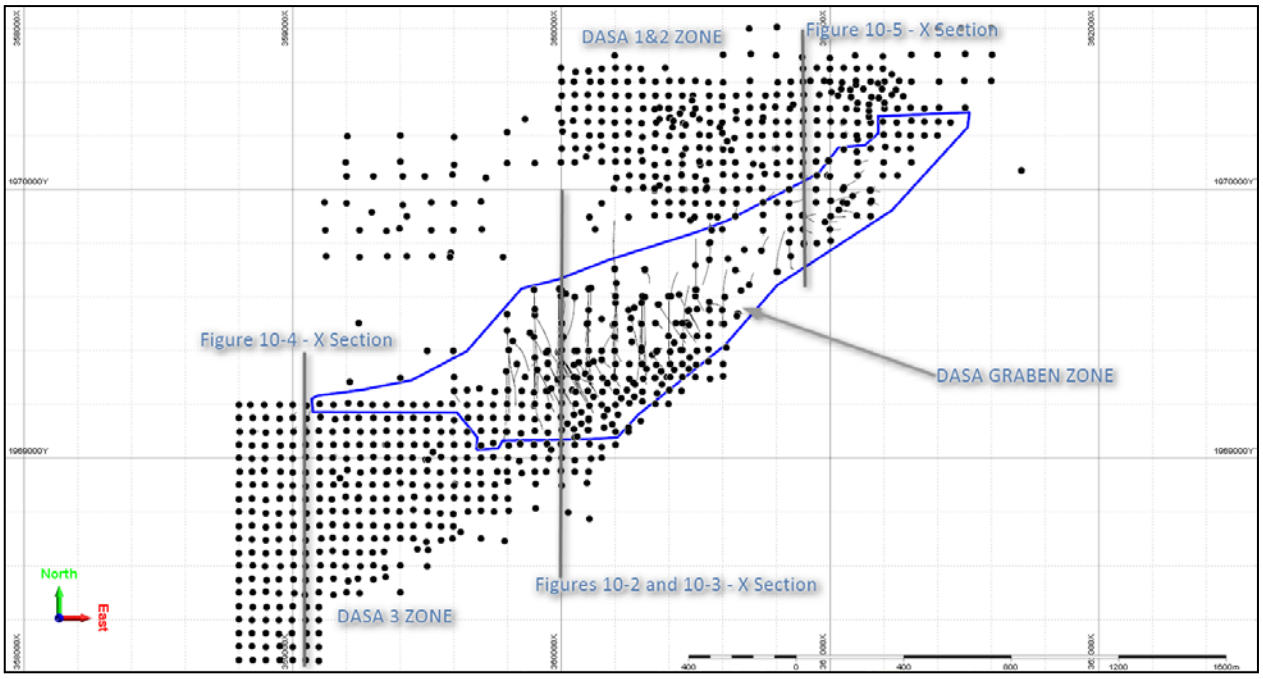


Figure 10-1: Dasa drill holes location map

Pertel (2019)

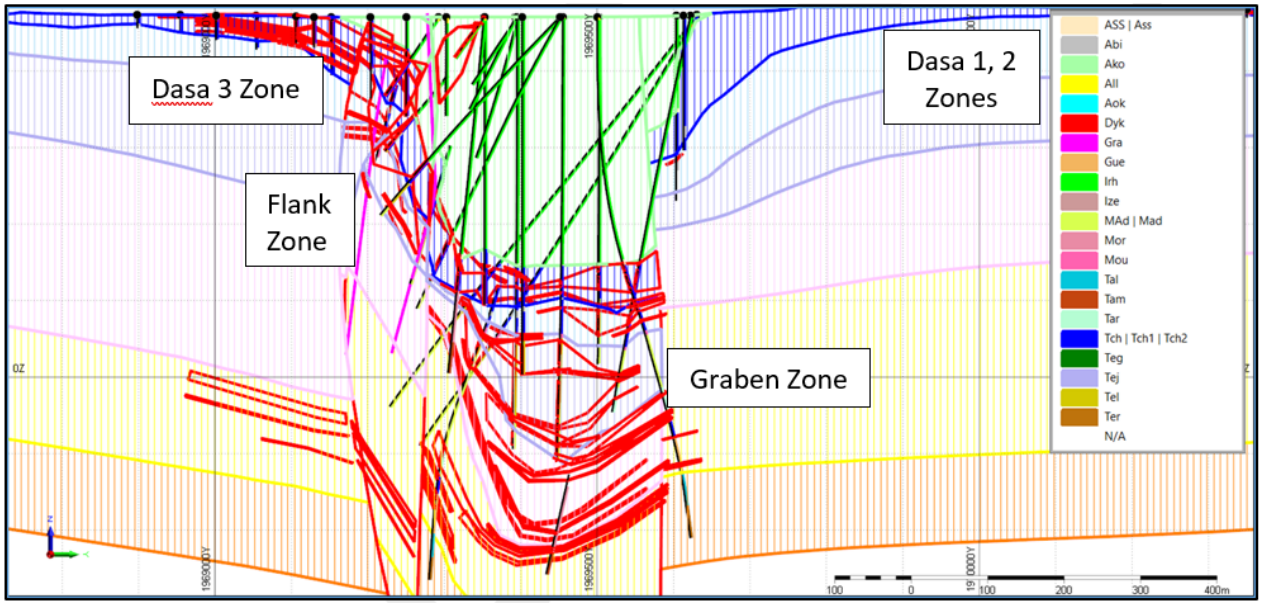


Figure 10-2: Dasa Project schematic drill section – geology (Section 360000mE, looking west – see Figure 10-1)

Source: Pertel (2019)



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 (2019)

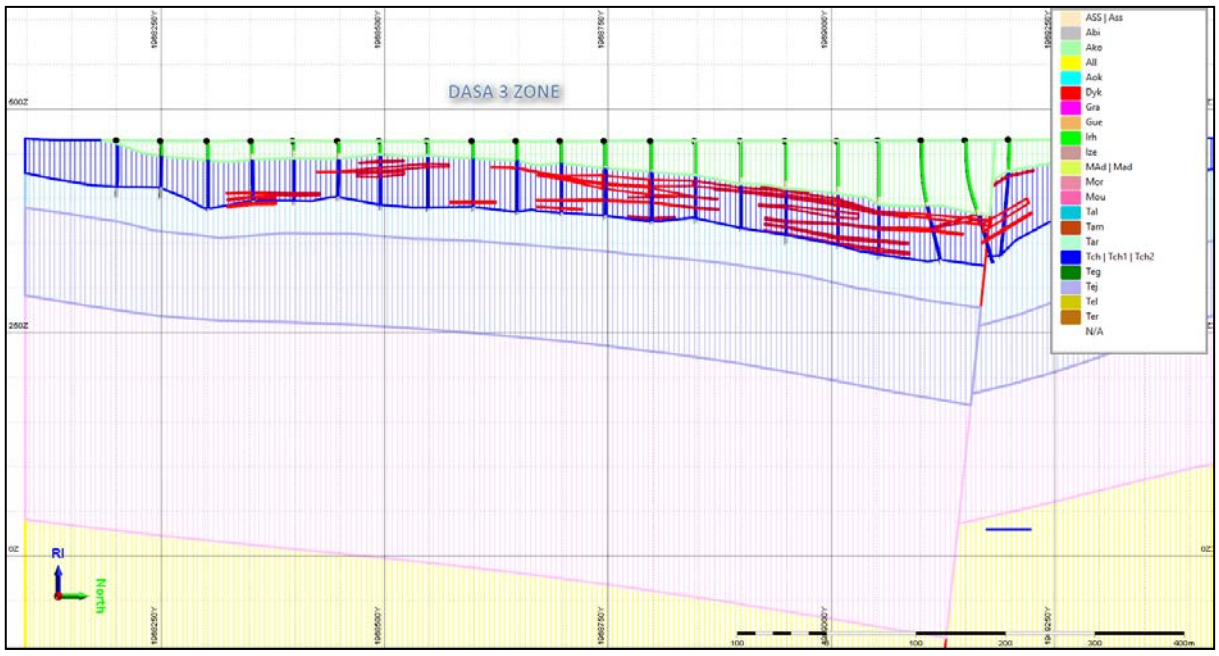


Figure 10-4: Dasa Project schematic drill section – geology (Section 359000mE, looking west – see Figure 10-1)

Source: Pertel (2019)

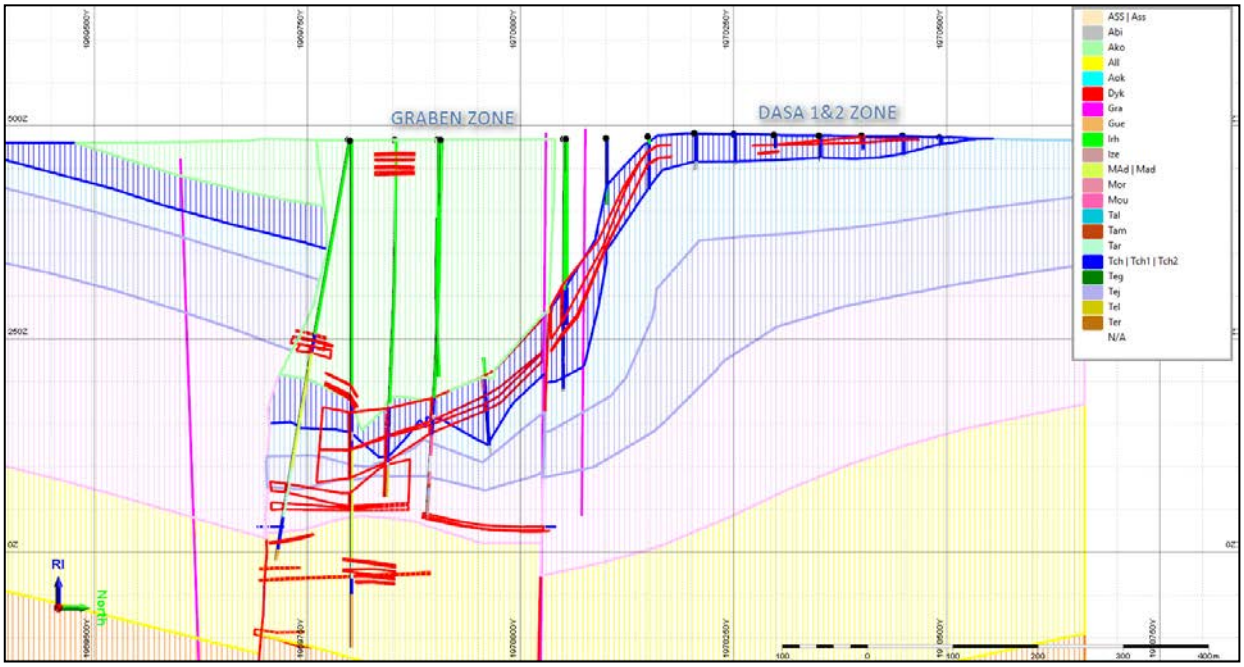


Figure 10-5: Dasa Project schematic drill section – geology (Section 360900mE, looking west – see Figure 10-1)

Source: Pertel (2019)

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 hole's 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 drillholes 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 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 10-6) at the dedicated core photography facility.



Figure 10-6: 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 QAQC 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. Five probes were used in the 2017–2018 program: UEP1805, DGGG1307, DGGG1734, DIL801 and DIL1125.

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” x 2” Nal crystal
DIL38 #1126	0.1305	38	0.0047	0.000004	0.043	2,120	1” x 2” Nal crystal
DIL38 #1250	0.1362	38	0.0047	0.000004	0.043	2,120	1” x 2” Nal crystal
DIL38 #80	0.126	39	0.0047	0.000004	0.043	2,120	1” x 2” Nal crystal
BDVG #735	0.1119	42	0.0047	0.000004	0.043	140	1” x 2” Nal crystal
DGGG1307, PM	0.8089	42	0.0047	0.000004	0.043	150	2 cm x 5 cm Nal

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
DGGG1304, PM	0.8089	42	0.0047	0.000004	0.043	150	2 cm x 5 cm Nal
DGGG9354, PM	0.8089	42	0.0047	0.000004	0.043	150	2 cm x 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 has 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. 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.

CSA Global 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. The Qualified Person reviewed the calibration results and was satisfied.

CSA Global 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 intervals of 36 holes drilled at Dasa. 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:

$$GTcore = Ucore \times Tcore = Factor\ (CPS \times Tprobe) = GTprobe$$

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 has found that the coefficient of correlation between the chemical grades and 10 m calculated grade composites is 0.755, with a 2-sigma precision on the mean of 5.8%: a relatively close clustering of data along a linear relationship.

Melabar Geoconsulting provided CSA Global with a detailed report describing the uranium equivalent grade calculation based on GR results. The report was reviewed by CSA Global, 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 10-7.

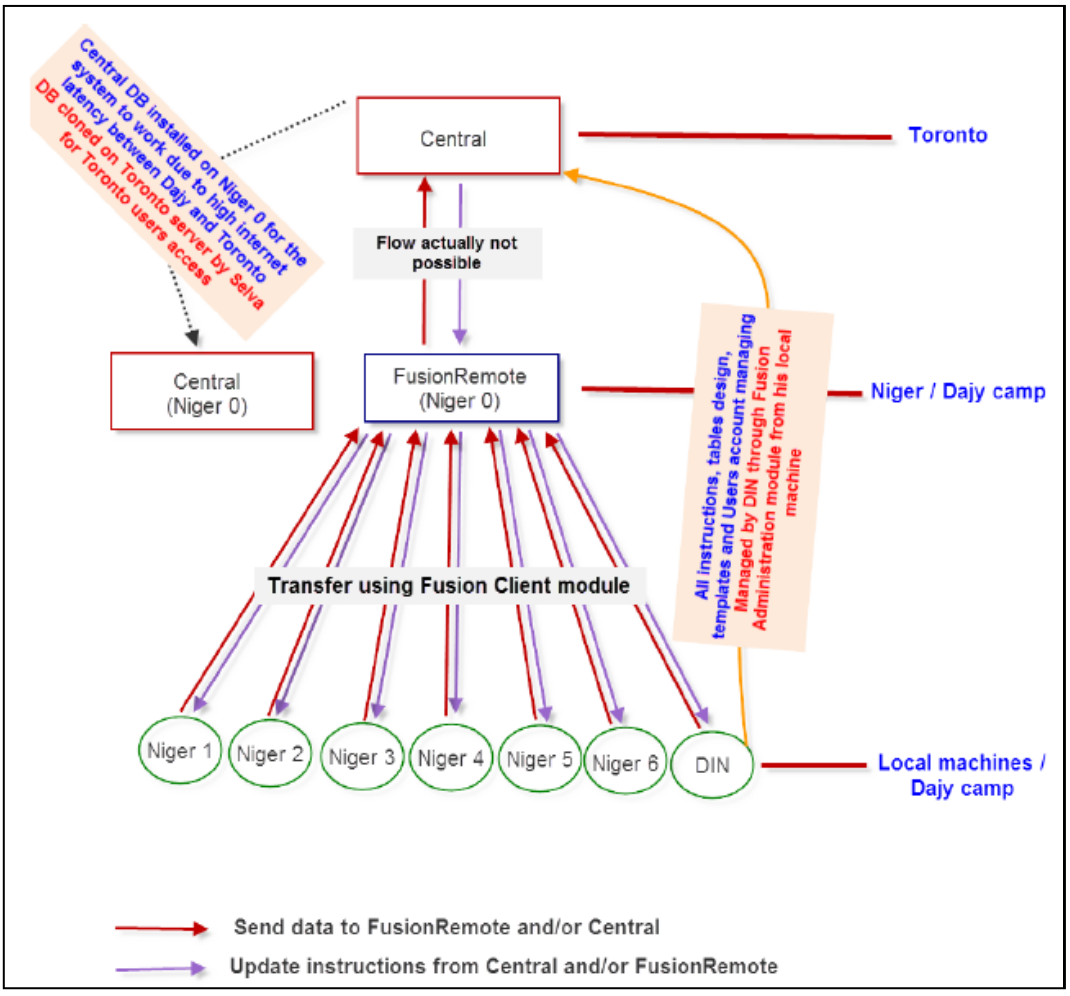


Figure 10-7: GAC data collection and handling system/ Constellation Software FUSION

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
1	Sand
2	Alluvium
3	Clay
4	Mudstone
5	Siltstone
6	Fine sandstone
7	Medium-grain sandstone
8	Coarse-grain sandstone
9	Very coarse-grain sandstone
10	Micro conglomerate
11	Conglomerate

Code	Lithology
12	Limestone
13	Marl
14	Muddy sandstone
15	Sandy mudstone
16	Calcareous sandstone
17	Carbonate mudstone
18	Arkosic sandstone
19	Analcimolite
20	Dolomite
21	Muddy siltstone

Code	Lithology
22	Granite
23	Diorite
24	Amphibolite
25	Gneiss
26	Schist
27	Organic matter sandstone
28	Pyritic sandstone
29	Coal
30	Graywacke
31	Analcimolitic sandstone

Table 10-7: Codes of alteration and mineralization

Alteration	Mineralization
Carbonate: Ca Iron: Fe Chlorite: Cl Sulphides: Su Manganese: Mn Clay: Cy	Uranium Pitchblende: Pe Uraninite: Ur Coffinite: Co Carnotite: Ct 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 is stored in the core boxes at the base camp. 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 sought 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 or half core and appropriately bagged and labelled (before 2013 only half core was retained and hence only quarter 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 or half 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. 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).

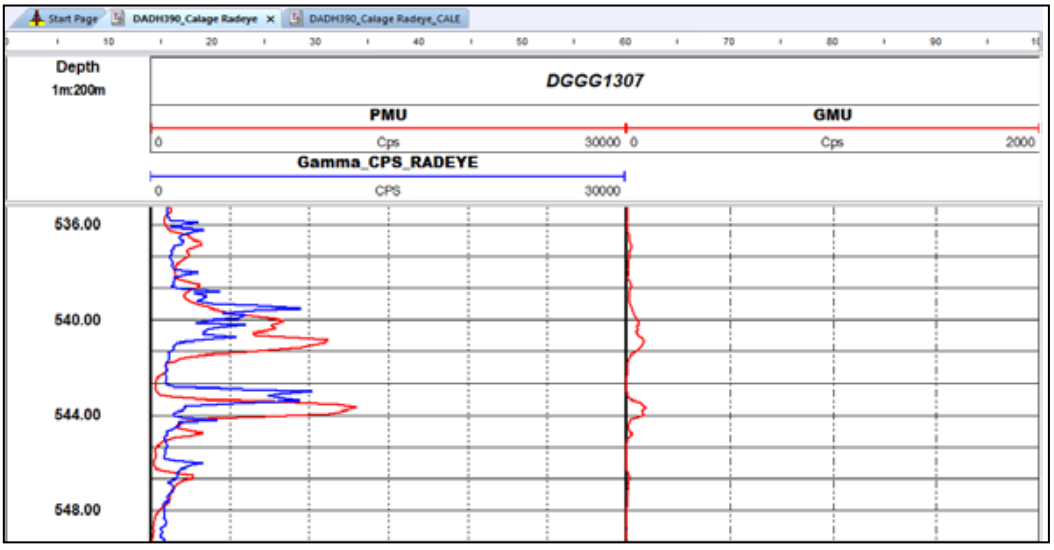


Figure 11-1: Hole DADH390, RadEye (blue) vs Probe (red) depth before rescaling

Source: GAC

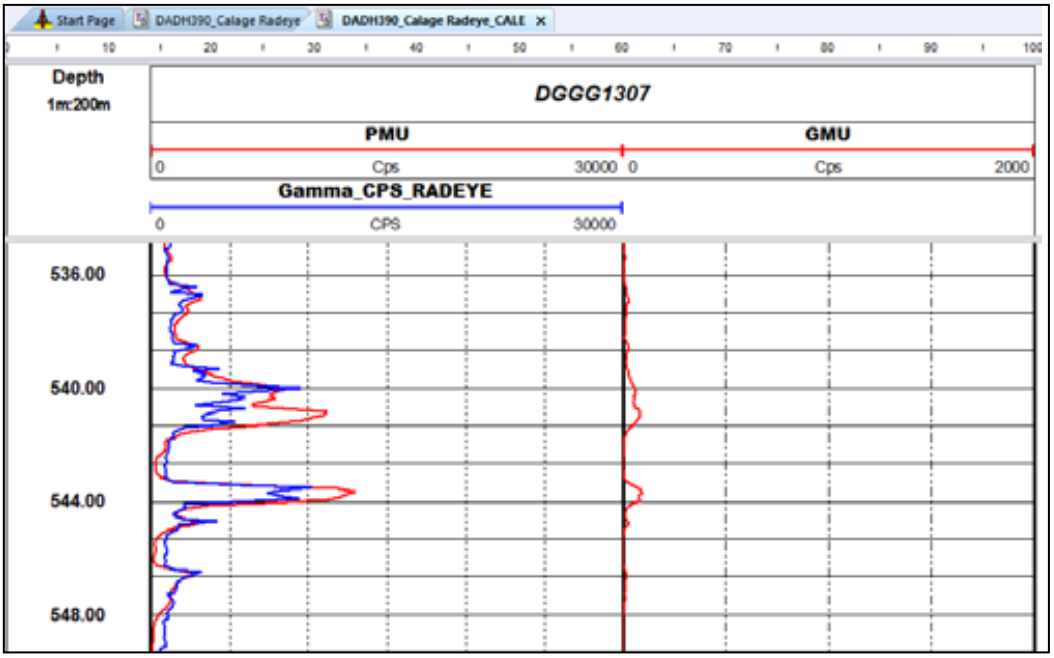


Figure 11-2: Hole DADH390, RadEye (blue) vs Probe (red) depth after rescaling

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.

11.4. 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).

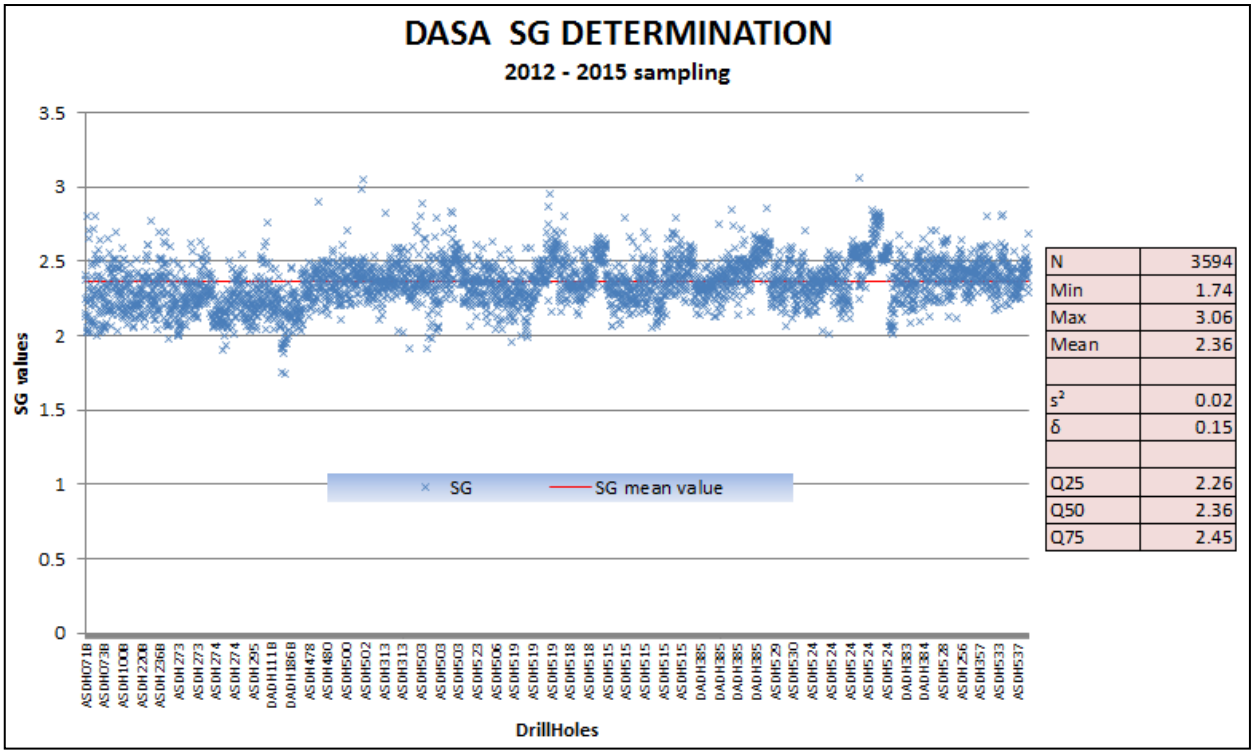


Figure 11-3: Average bulk density determination from core samples

Source: GAC

11.5. 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 the Table *11-2*. Results show that the quality of sampling and assaying is acceptable. Comparison of original samples and duplicates is provided in the QAQC reports on Dasa Project (GAC 2012, 2013) (Figure 11-4).

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 a number of 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.

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.

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.

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.

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 x 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 U_3O_8 achieved uranium recoveries around 93% at a grind P_{80} 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 were 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

Borehole	Average Grade, U_3O_8 ppm (by XRF)	Mass, kg
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 Standstone	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		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 1.5M 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 diuranate, 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 P80 ~ 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 on the basis of the demonstrated PEA bench scale process viability. The Feasibility Study test work programs focussed on:

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 five 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 80kg.

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 Standstone	14.4	0.0	0.0

Type	Lithology	Campaign 1 (%)	Campaign 2 (%)	Campaign 3 (%)
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
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 (~ 20kg) 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 x 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
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 x 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 this 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.

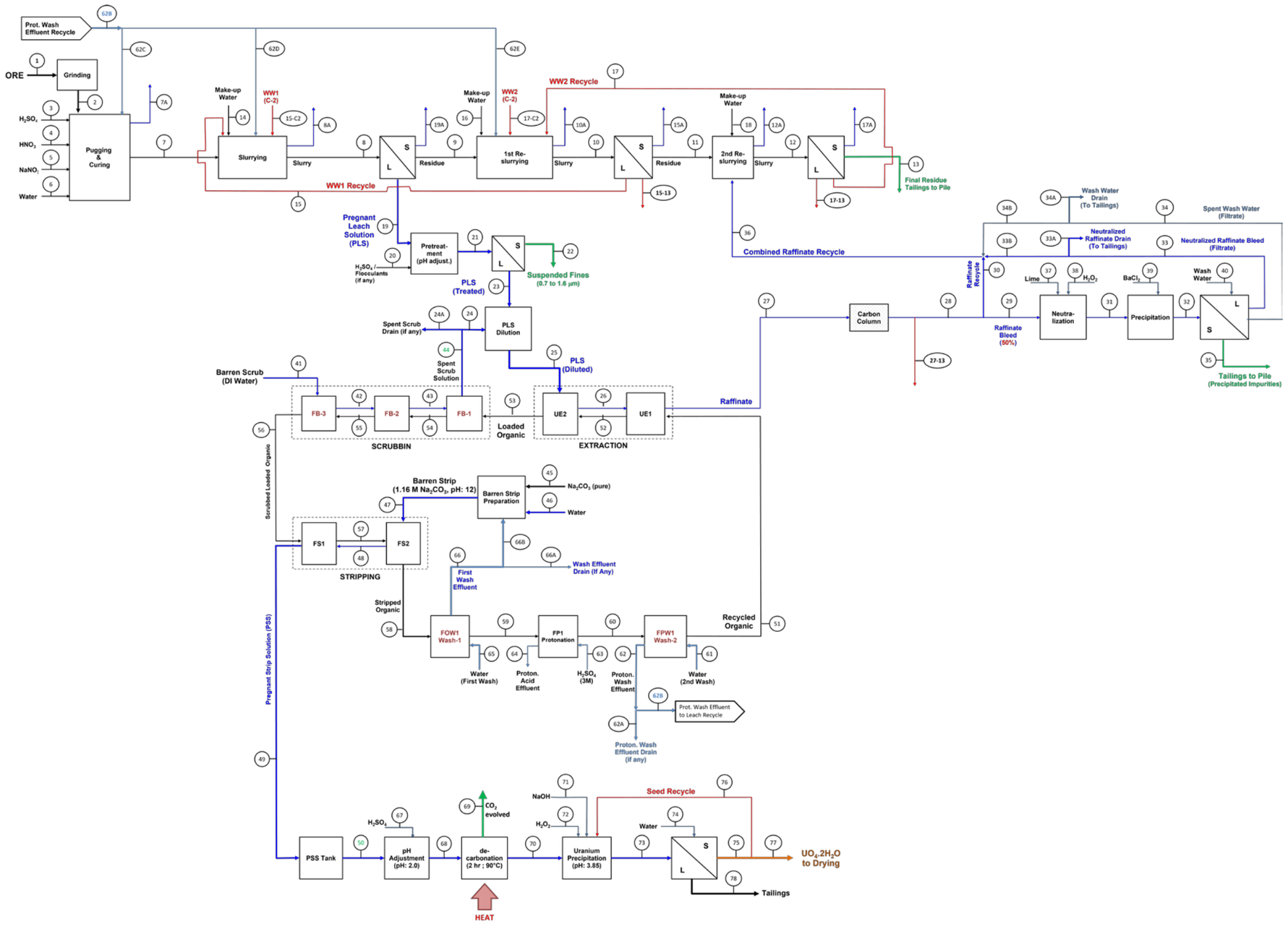


Figure 13-1: Dasa pilot plant flowsheet

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 10mm 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.

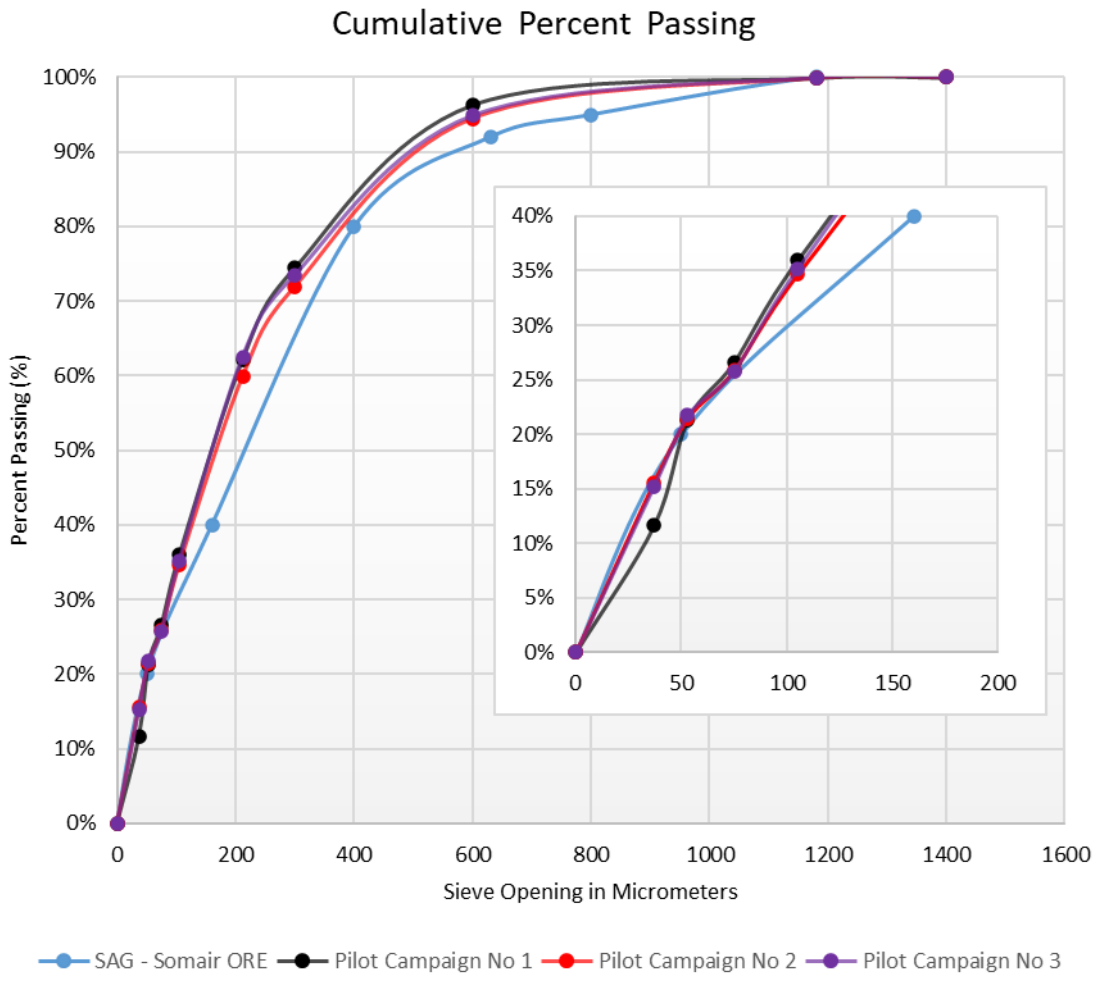


Figure 13-2: Campaign 1, 2 and 3 particle size distributions vs Somair reference

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 1.5M 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, 1.16M 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

	Strip Agent	Loaded Organic (mg/L U)	Stripped Organic (mg/L U)	Stripping Efficiency (%)
Campaign 1	1.50M NaCl	2 651	246	90
Campaign 2	1.16M Na_2CO_3	2 537	0	100
Campaign 3	1.16M 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 1.3g/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 20wt% $Ca(OH)_2$) 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 H2SO4.

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

	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:

1. Insight R&D, Dasa Pilot Plant 1st Filtration Tests, Project Report Number 2010-01-1-R0, April 8, 2021

2. Insight R&D, Dasa Leach Filtration 2nd Test Campaign, Project Report Number 2103-01-1-R0, July 8, 2021
3. 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

	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. Additional work is required to find the best operating window (time, temperature, pH, CO_2 levels) to

increase crystal size and improve filtration rates, as under current filtration rates, large filters would be required in the plant.

Recent test work at 60°C with sequential caustic addition improved settling rates eight-fold. Some additional test work including reseed, and application of flocculant is still in progress which may lead to further improvements.

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 life of mine 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 amount 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.

The following are the major process flowsheet differences between the PEA and FS:

- Application of a stronger sodium carbonate based SX stripping optimised the extraction of uranium from the organic solution with a better uranium recovery.
- The production of sodium di-uranate produces a more practical intermediate product with lower operating cost and easier to operate.
- The feasibility study incorporated a raffinate solution bleed 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 13-18. 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 13-18: 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.

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 13-19: Dasa Pilot Plant Test work overall uranium recovery summary at 80kg/t H2SO4 and 2.5kg/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	%	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.

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) and is repeated below. 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 1 June 2019.

14.1. Software Used

The Dasa uranium deposit Mineral Resources were updated by the Qualified Person and other CSA Global geologists under supervision of the Qualified Person, using Micromine version 2018.6 (18.0.947.6 x64) software.

14.2. Database Compilation

GAC supplied CSA Global with the updated database in Microsoft Excel format. The database included all the exploration results for all exploration stages, including all holes drilled in 2018.

The main analytical database comprises estimated uranium equivalent grades (eU3O8) based on the gamma-logging of the drillholes. A separate file was supplied with all results of chemical assays.

The uranium oxide equivalent grades were calculated 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. All other correction factors and parameters are shown in Table 10-3.

The eU_3O_8 grades were calculated for each 10 cm interval using LAS files. Some historical holes were however not gamma-logged to the total depth but had results of the chemical assays which were also used for interpretation and modelling. The available data is summarized in Table 14-1.

Table 14-1: Summary of supplied and used data

Category	Supplied	Gamma-logged	Chemical assays	Used for MRE
Drillhole collars	1,028			
Drillholes		1,017	93	1,018
Metres drilled	150,399	138,230	6,333	138,844
Survey records	11,622			11,622
Records in assay data file		1,138,299	9,784	
Assayed/probed intervals for U_3O_8		1,382,299	9,772	1,382,999 (including 700 chemical assays and 1,382,299 probed intervals)
Records in geology logging file	9,039			9,039

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 DXF file).

Import of the various datasets into Micromine proceeded without error.

14.3. Data Validation

The analytical and geological database was 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° 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. All these holes were excluded from the resource estimation process.

No other errors have been identified in the databases, and no corrections were introduced to the database. Those intervals that were not gamma-logged, but assayed, were added to the combined data file with uranium grades.

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 parameters of 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 show an obvious cut-off grade that could be used for interpretation of uranium mineralization. A decision was made to employ the nominal cut-off grade of 100 ppm for the subsequent update of interpretation of mineralized bodies. The adoption of 100 ppm cut-off grade also reduces the residual effect of any radium halos by their exclusion.

Once the uranium mineralization was interpreted for all mineralized lenses and wireframed, classical statistical analysis was repeated for the composited samples within the interpreted envelopes to meet the following objectives:

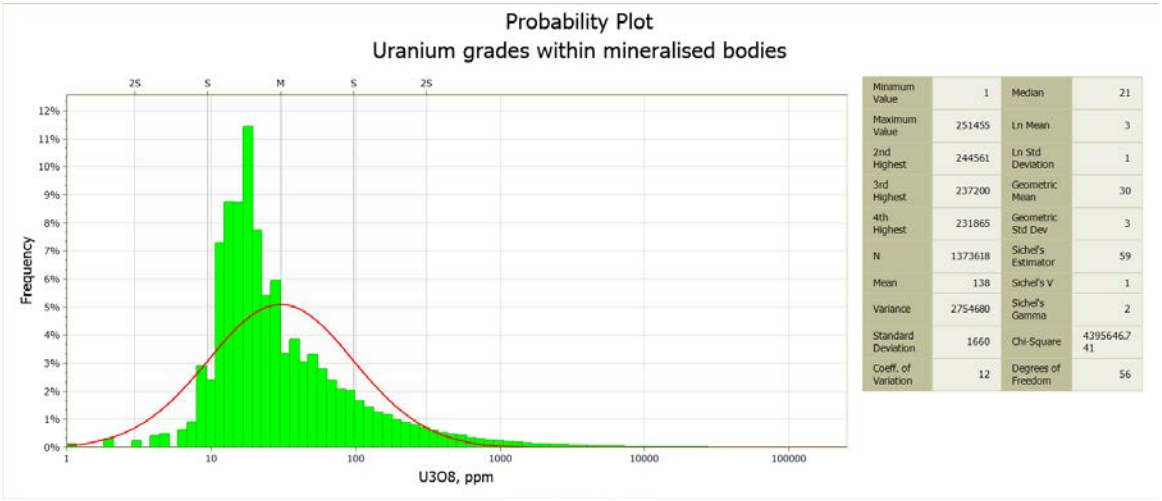
To estimate the mixing effect of grade populations for uranium within the interpreted mineralized bodies

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.

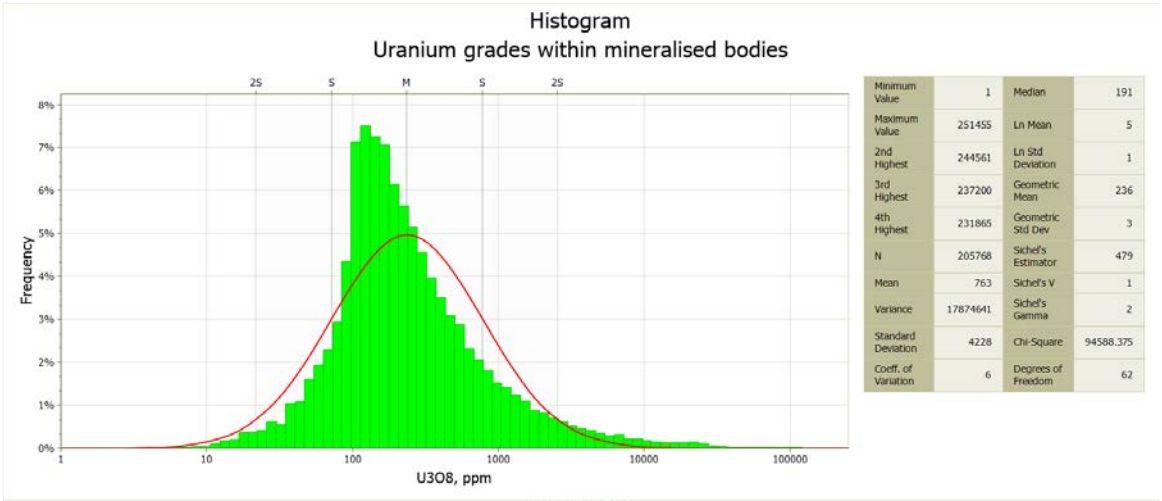
The input sample file was flagged to exclude those intervals that appeared outside the wireframed mineralized envelopes for uranium. The modelled histogram for the uranium grades restricted within mineralized envelopes does not demonstrate apparent mixing of grade populations for uranium (Figure 14-2).

The lognormal histograms and cumulative probability plots were analysed 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 10cm intervals. Thus, a decision was made that no top-cut grade values are applied on the analysed intervals because deconvolving of uranium grades from gamma-logging results usually takes into account abnormally high grades and, therefore, top cutting is not required.



Minimum Value	1	Median	21
Maximum Value	251455	Ln Mean	3
2nd Highest	244561	Ln Std Deviation	1
3rd Highest	237200	Geometric Mean	30
4th Highest	231865	Geometric Std Dev	3
N	1373618	Sichel's Estimator	59
Mean	138	Sichel's V	1
Variance	2754680	Sichel's Gamma	2
Standard Deviation	1660	Chi-Square	43956467 41
Coeff. of Variation	12	Degrees of Freedom	56

Figure 14-1:Log histogram for unrestricted uranium grades



Minimum Value	1	Median	191
Maximum Value	251455	Ln Mean	5
2nd Highest	244561	Ln Std Deviation	1
3rd Highest	237200	Geometric Mean	236
4th Highest	231865	Geometric Std Dev	3
N	205768	Sichel's Estimator	479
Mean	763	Sichel's V	1
Variance	17874641	Sichel's Gamma	2
Standard Deviation	4228	Chi-Square	94588.375
Coeff. of Variation	6	Degrees of Freedom	62

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)

Element	Min (ppm)	Max (ppm)	No. of points	Mean (ppm)	Variance	Standard deviation	Coefficient of variation	Median (ppm)
eU_3O_8	Unrestricted sample intervals							
	0	251,455	1,382,999	137	2,728,214	1,652	12	21
	Intervals within mineralized bodies							
	0	251,455	205,804	761	17,711,245	4,208	5.5	191
	0.5 m composites within mineralized intervals							
	2	218,487	42,158	765	16,918,201	4,113	5.4	194

14.5. Lithological Model

A full lithological model was developed for the Dasa deposit using logged lithological codes in the supplied database. 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 block separately, limited by the fault planes.

Example of the modelled lithology for one of the cross sections is shown on Figure 14-3.

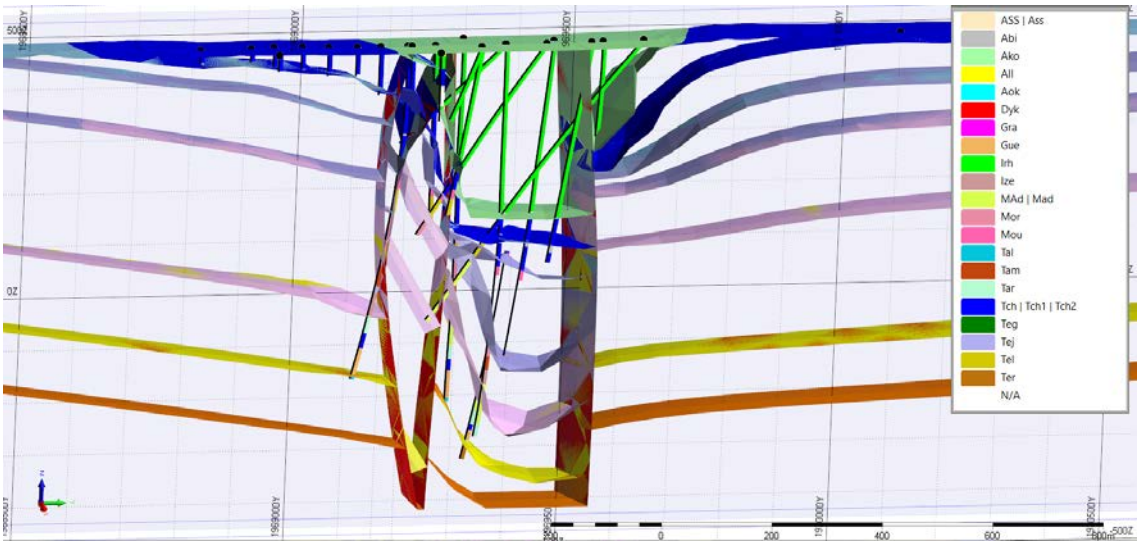


Figure 14-3: Lithological model and faults, Section 23 (359,900mE)

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 employed to calculate the mineralized intervals using a 100 ppm 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 also between the sections to make sure that the correct lens numbers would subsequently be assigned to the analytical data file.

The grade compositing process employed 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 56 SN cross sections which were 50 m apart with some infilled holes within the Flank Zone. When uranium grades were interpreted, 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 number of 327 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. It was initially 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 a half distance between the adjacent exploration lines. 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 interpolated 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 the Figure 14-4, where thick red lines along drillhole traces are grade composites, traces are colour coded according to lithology, red strings are interpreted mineralized bodies, purple lines are faults, and hatched areas are different lithological units.

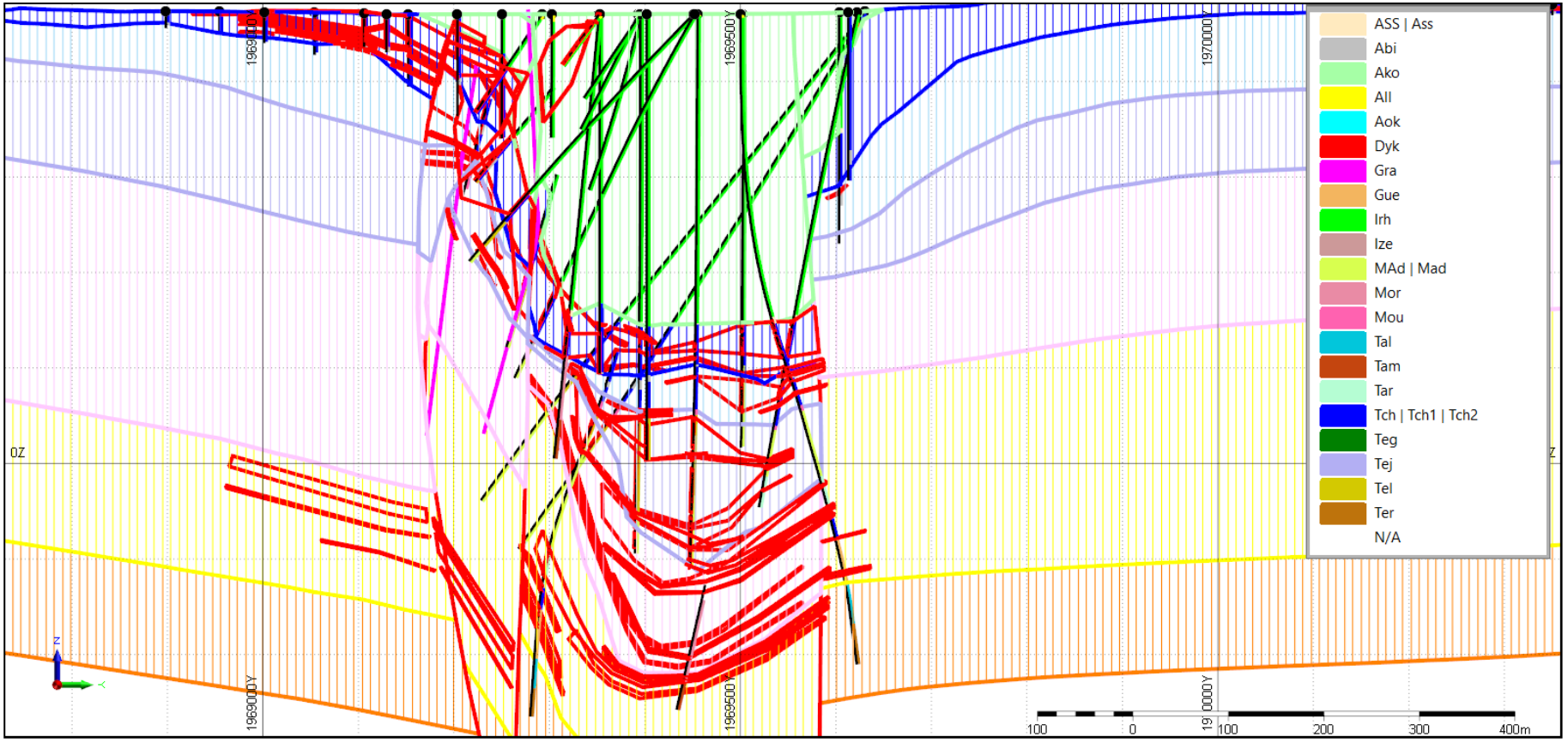


Figure 14-4: Schematic example of interpretation of the Dasa deposit – Section 360,000mE

Note: Hatched areas – lithological units, red are mineralized envelopes, black is drillhole trace with red hatches on the left – grade composites.

14.7. Wireframing

The interpreted strings were used to update 3D 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 3D view of the modelled mineralized bodies. A total of 327 mineralized wireframes were modelled for the deposit. Each wireframed lens had a different colour, and steeply dipping faults are shown with dark red colour in Figure 14-4. 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.

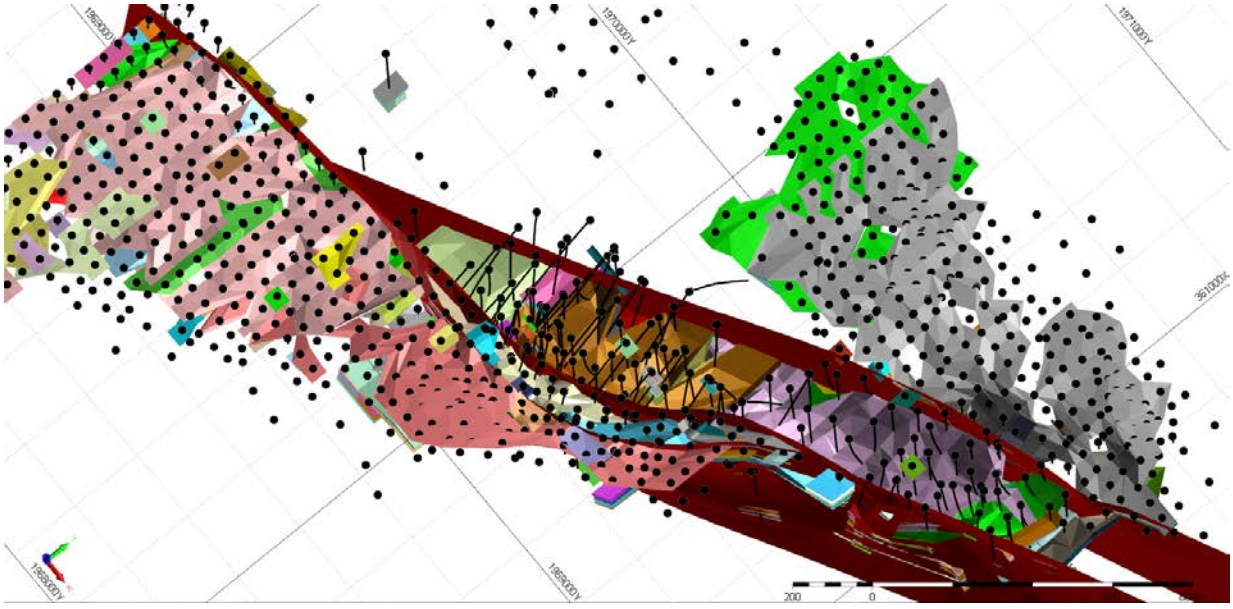


Figure 14-5: Oblique view of the wireframed uranium mineralized envelopes and fault planes for the Dasa deposit (looking northwest)

All wireframe models were validated so that they are all solids (closed) and that they do not contain intersecting triangles.

Table 14-3: Number of interpreted wireframes at the Dasa deposit

Number of wireframes	Volume (m^3)
327	79,027,830

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 the drillhole sample intervals. Samples were selected for individual envelopes and flagged accordingly for each modelled mineralized envelope.

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 mineralized envelopes.

Majority of intervals in the analytical data file were 10 cm based on gamma-logging. 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

Recent drilling confirmed that the Flank Zone is a set of steeply dipping mineralized bodies with variable northeast dipping 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 to update the model.

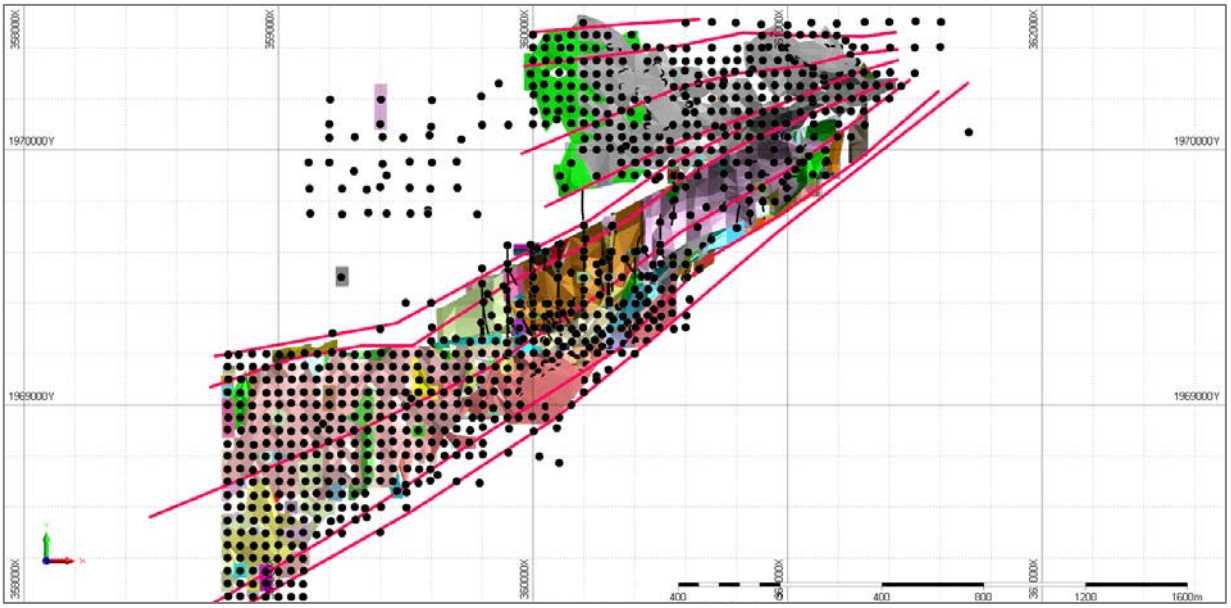


Figure 14-6: Strings along the deposit strike (plan view)

It was decided that a dynamic search would deliver 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:

A set of strings was digitized along the deposit strike in plan view as shown in Figure 14-6 (red lines).

A set of 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 wireframe slices. These strings (Figure 14-7) represented general plunge for each modelled lens (purple 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 wireframe slices. These strings (Figure 14-8) represented general dip for each modelled lens (purple lines).

When all strings for azimuth, plunge and dip were digitized, they were “normalized”, 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 lenses plunge.

Inclination for each pair of points along strings was calculated and recorded in the string file which was digitized for the lenses dip.

All calculated values for strings were checked to have correct positive or negative values and corrected if it was necessary.

The resultant strings were used to interpolate azimuth, plunge, and dip values into each cell of the block model. Spherical search was employed for this process.

The resultant block model had assigned azimuth, dip, and plunge values for each model cell, which represented the general directions of mineralized lenses.

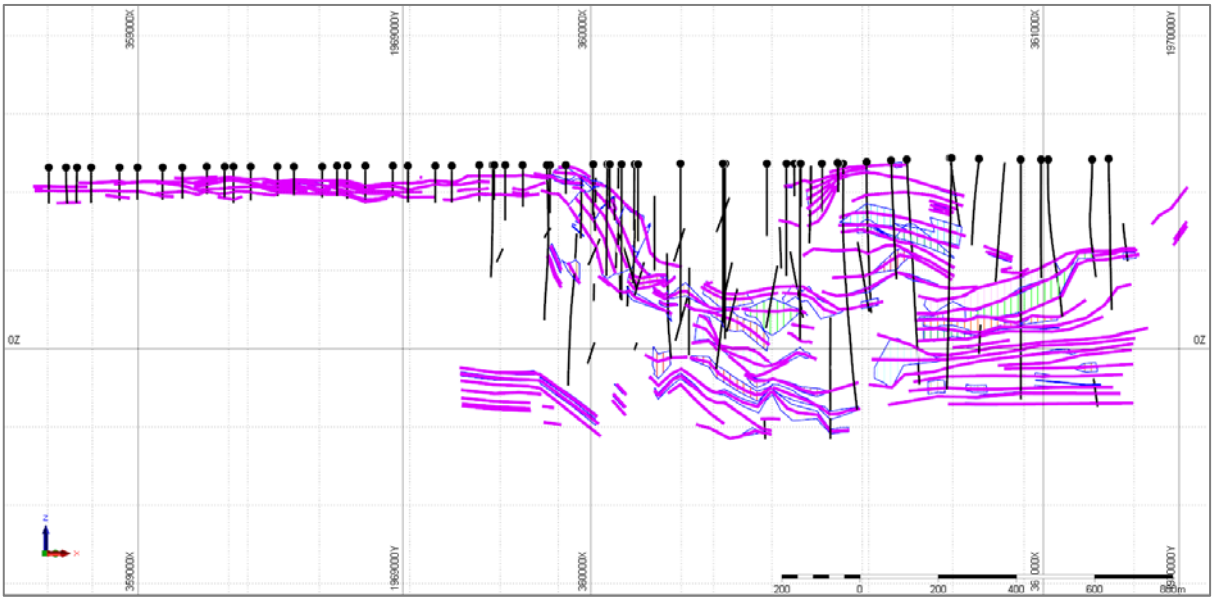


Figure 14-7: Strings for the plunge of the bodies (section view looking northwest)

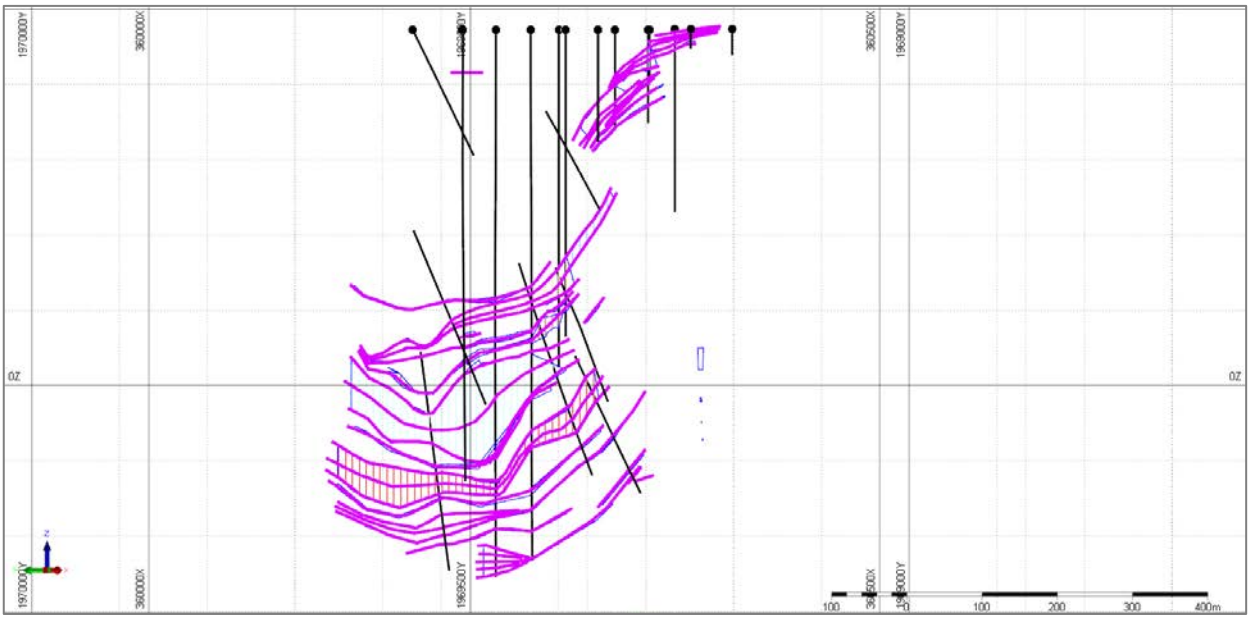


Figure 14-8: Strings for the dip of the bodies (section view looking northeast)

14.10. Geostatistical Analysis

The new analytical data was merged with the historical data, and the geostatistical analysis was repeated for the model update. It was found that the previously modelled semi-variograms did not significantly change their characteristics.

Downhole experimental variogram was modelled to estimate the expected nugget effect for uranium grades (Figure 14-9). The estimated nugget effect was then applied to model directional semi-variogram models.

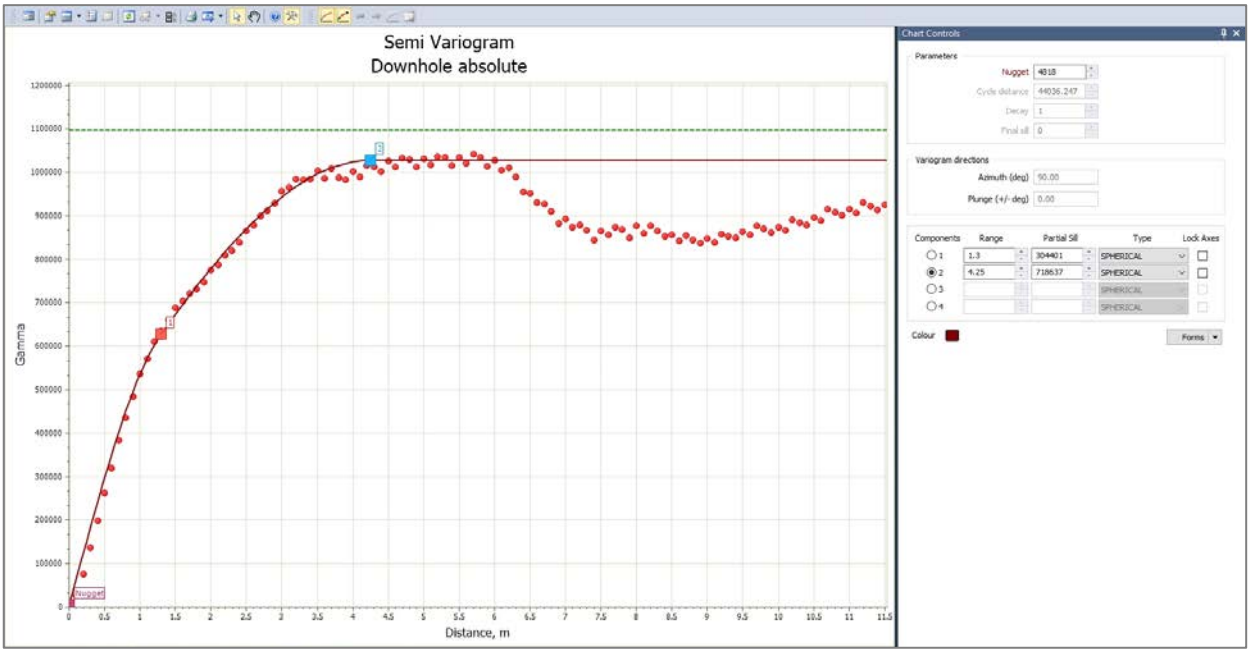


Figure 14-9: Downhole absolute semi-variogram model for uranium

A semi-variogram map was then generated in plan view to establish the direction of maximum grade continuity (Figure 14-10). The map clearly demonstrated that the azimuth of maximum continuity is 55°, which generally matches with the overall strike of the mineralized bodies. The directions for semi-variogram models were established as 55° azimuth, 0° dip; 145° azimuth, 0° dip; and vertical; but the grade interpolator with dynamic search applied variable directions to the semi-variogram models in line with the parameters estimated for each model cell.

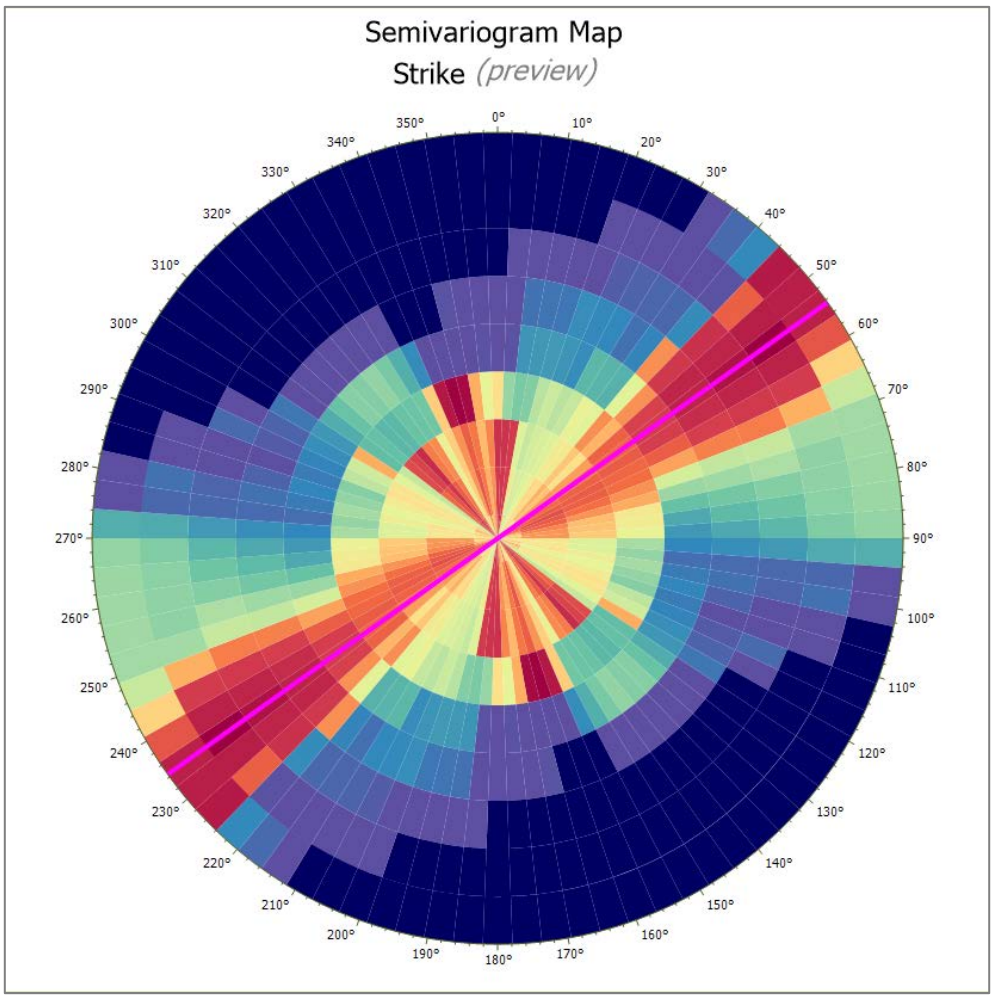


Figure 14-10:Semi-variogram map (plan view)

It was found that robust absolute semi-variograms are difficult to model most likely due to the high coefficient of variation of uranium grades. Therefore, relative semi-variogram models were calculated and modelled for the composited uranium sample file without applied top-cut grades (Figure 14-11).

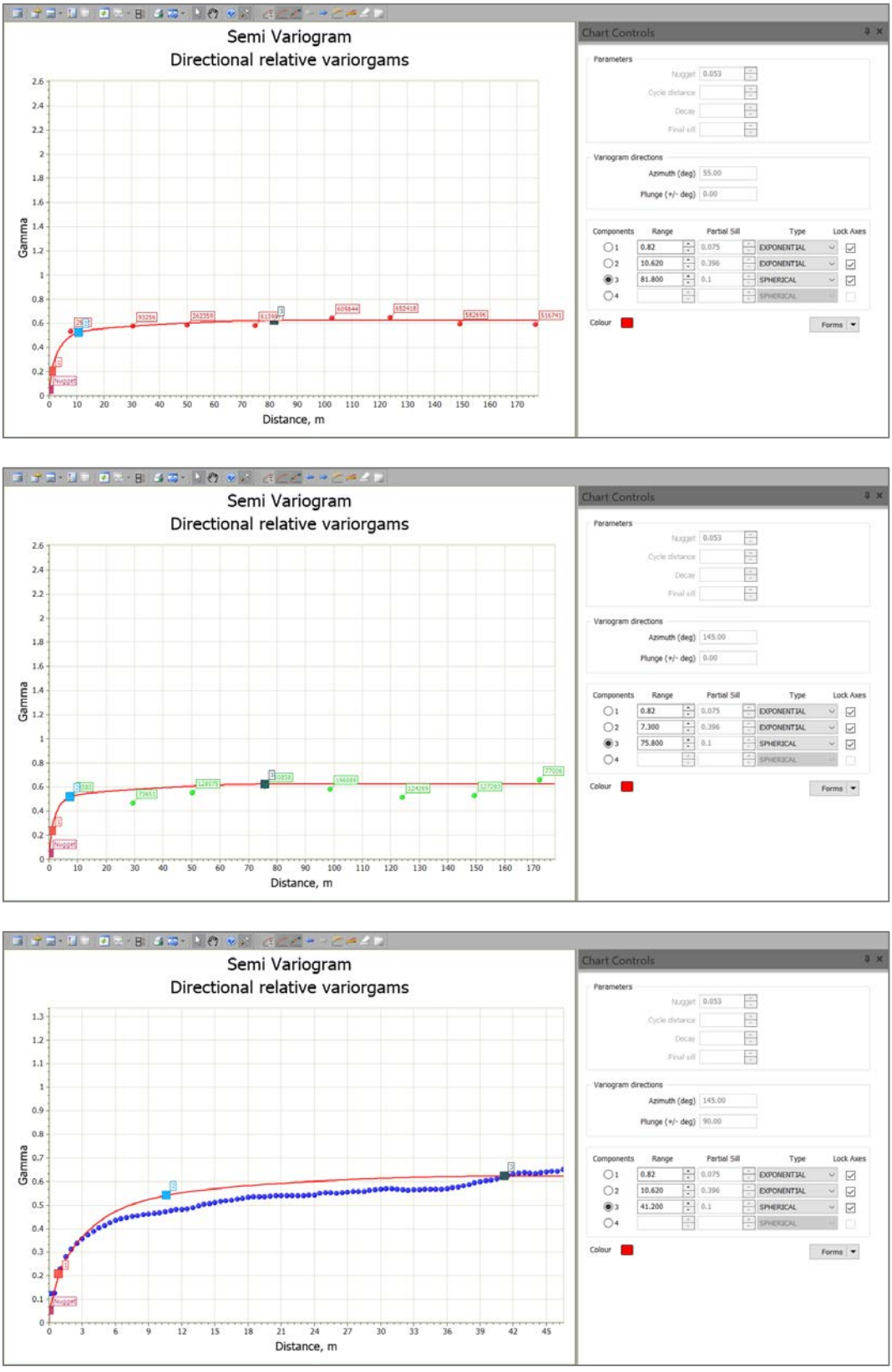


Figure 14-11:Relative semi-variogram models for uranium (three main directions)

All modelled experimental semi-variograms were exponential and spherical with three nested structures. The obtained semi-variogram ranges were used to determine the search radii. The latter were used in the grade interpolation processes.

Table 14-4: Semi-variogram characteristics

Element	Type	Axis	Azimuth	Dip	Nugget	Partial sills	Ranges
U_3O_8	Relative Exponential and Spherical	Main	55	0	0.053	0.075, 0.396 and 0.1	1, 10.6 and 81.8
		Second	145	0			1, 7.3 and 75.8
		Third	145	90			1, 10.6 and 41.2

14.11. Block Modelling

An empty block model was created within the closed wireframe models for the mineralized envelopes. Each modelled lens was assigned a unique code in the model file. The block model was then restricted 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 characteristics

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 chosen on the basis of the exploration grid and general morphology of the mineralized bodies and in order 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.

14.12. Grade Interpolation

Uranium equivalent (eU_3O_8) grades were interpolated into the empty block model using the Ordinary Kriging (OK) interpolation method. This was then rerun using Inverse Distance Weighted (IDW) method with the powers of two and three as cross checks. The search ellipse and semi-variogram models were oriented by the interpolator using the azimuth, plunge and dip values for each model cell accordingly.

The OK and IDW processes were performed at different search radii until all model cells were interpolated. The search radii were determined by means of the evaluation of the semi-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 semi-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 semi-variogram long ranges in all directions. The third interpolation run employed radii equal to full semi-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 semi-variogram ranges until all model cells were informed with uranium grade.

When model cells were estimated using radii not exceeding full semi-variogram ranges, a restriction of at least three samples from at least two drillholes was applied to increase the reliability of the estimates. The general definition of the interpolation strategy is presented in Table 14-6 below.

“Parent estimation technique” was 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. To do that, all lenses were separated vertically from each other by 1,000 m. The vertical search was 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. Change of support was honoured by discretizing to 5-points x 5-points x 5-points. These point estimates are simple averages of the block estimates. The general definition of the interpolation strategy is presented in Table 14-6.

Table 14-6: Interpolation parameters

Interpolation method	Ordinary Kriging				
Search radii	Less or equal to 1/3 of semi-variogram ranges	Less or equal to 2/3 of semi-variogram ranges	Less of equal to semi-variogram ranges	2 semi-variogram ranges	Greater than 2 semi-variogram ranges
Minimum number of points	4	4	4	1	1
Maximum number of points	16	16	16	16	16
Minimum number of drillhole	3	3	3	1	1

14.13. Bulk Density Values

Dry bulk density values were obtained during previous and recent exploration programs on the deposit. Direct measurements of 3,594 core samples were taken and processed by GAC. More information relating to bulk density measurement procedures, is provided in the previous sections of this Report (Section 11.4).

Bulk density values can be assigned to block model cells using the following methods:

Direct assignment of the values to block model cells

Calculation of values for each cell using regression formulas

Interpolation of values

Use of geological model to assign values into each model cell.

CSA Global used the first method, i.e., 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

No changes were made to the previously adopted 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 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 14-6.

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 semi-variogram ranges. It was decided that the exploration grid of at least 50 m x 50 m would support the Indicated Resource category if blocks were estimated from at least two drillholes by search ellipse not exceeding semi-variogram ranges. All the remaining model cells were classified as Inferred. No Measured Mineral Resource category was applied to the Dasa model.

The resource classification strategy is illustrated below (colours: green – Indicated, blue – Inferred) in Figure 14-12.

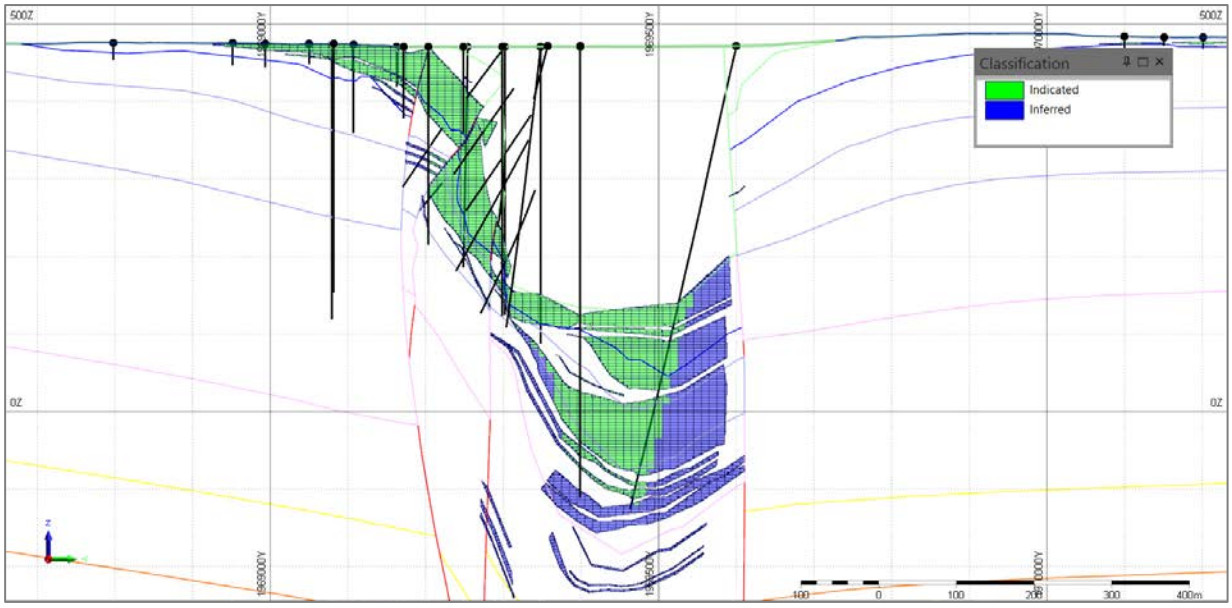


Figure 14-12: Resource classification strategy (Section 360050mE)

A summary diagram showing the distribution of Indicated and Inferred blocks is shown in Figure 14-13.

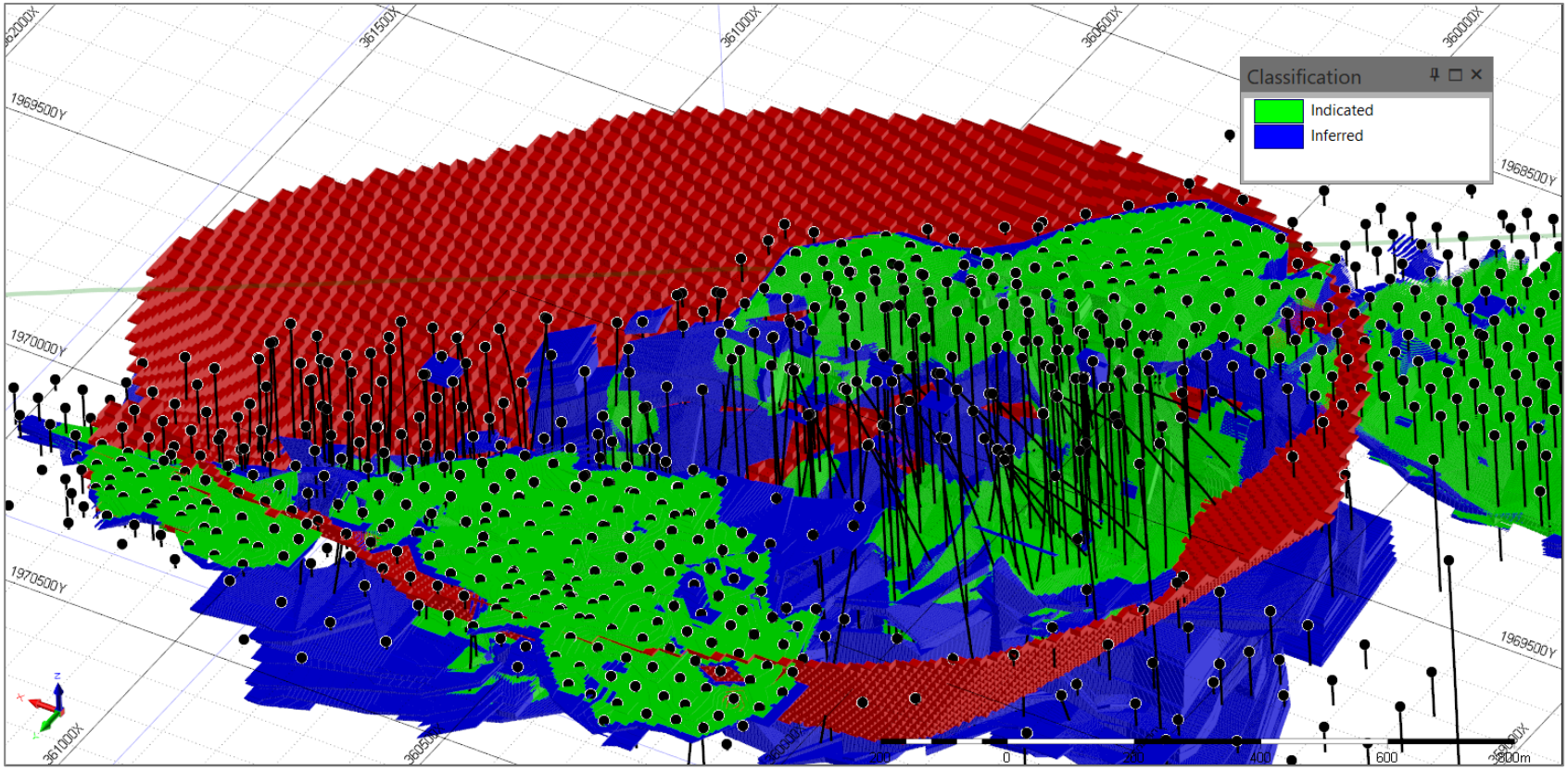


Figure 14-13: Summary diagram of indicated and inferred blocks in relation to the optimized open pit (looking southeast)

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
- Generation of swath plots to compare input and output grades in a semi-local sense, by easting, northing, and elevation.

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 14-14).

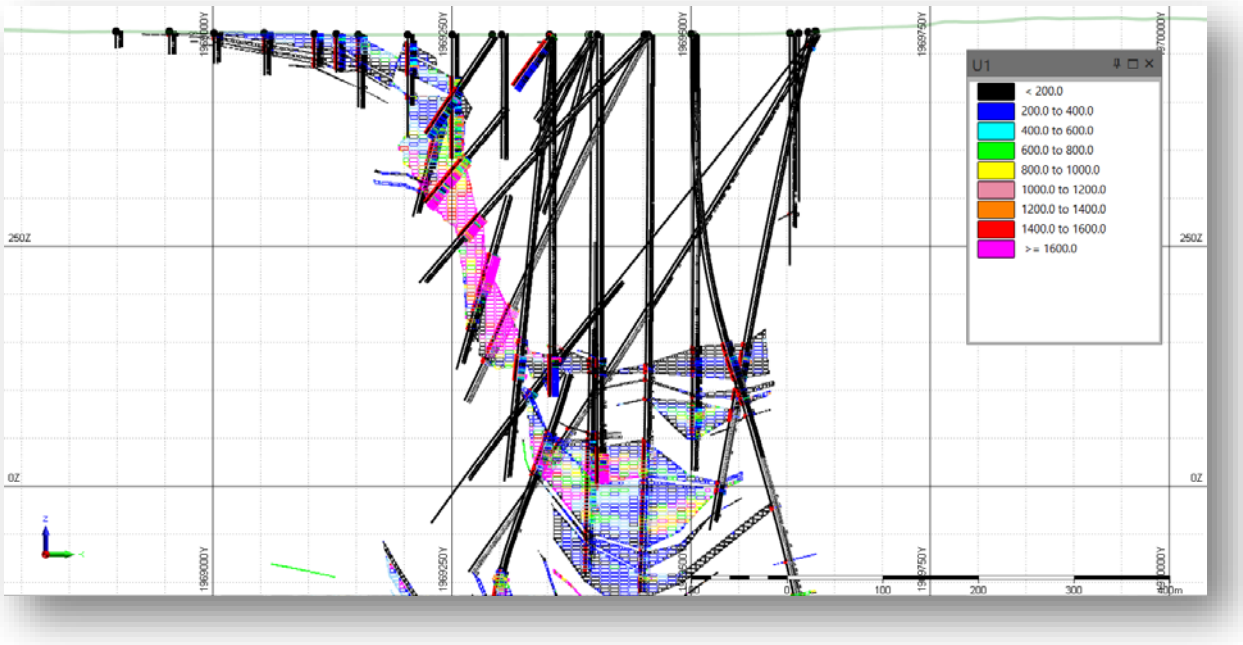


Figure 14-14: Visual comparison of eU3O8 grades in the model vs assays (Section 360,000 mE, looking west)

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 modelled grades were 17% relative lower than the grades in the composites (765 ppm eU_3O_8 in the composite file vs 631 ppm eU_3O_8 in the block model) for the combined categories. This difference is acceptable as it was expected due to the relative clustering of high-grade samples.

Comparison with Alternative Interpolation Methods

All grades were also interpolated using the IDW method with the power of two and three and then compared to the grades estimated by the OK method. A comparison of the grades and metal tonnage using OK vs IDW method at various cut-off grades is given in Table 14-7. OK returned generally more conservative grades and lower metal, but overall, the grades differ within acceptable limits.

Table 14-7: Comparison of global block model grades by OK and IDW methods

Cut-off (U_3O_8, ppm)	Kriged model		IDWx2 model		IDWx3 model		Difference (%)			
	U_3O_8 (ppm)	Metal (Mlb)	U_3O_8 (ppm)	Metal (Mlb)	U_3O_8 (ppm)	Metal (Mlb)	With IDWx2		With IDWx3	
							Grade	Metal	Grade	Metal
200	959	235	1,016	243	1,039	244	-5.9	-3.2	-8.3	-3.6
300	1,145	224	1,207	233	1,235	234	-5.4	-3.7	-7.8	-4.1
400	1,349	214	1,401	224	1,431	225	-3.9	-4.4	-6.1	-5.0
500	1,751	198	1,830	208	1,870	209	-4.5	-4.6	-6.8	-5.4
600	2,118	187	2,227	196	2,280	198	-5.2	-4.8	-7.7	-5.7
700	2,450	179	2,586	188	2,657	190	-5.5	-5.0	-8.4	-5.9
800	2,748	172	2,908	181	2,990	183	-5.8	-5.2	-8.8	-6.3
900	3,034	166	3,215	175	3,309	177	-6.0	-5.4	-9.1	-6.6
1,000	3,311	161	3,502	170	3,606	172	-5.8	-5.7	-8.9	-7.0
1,100	3,588	156	3,779	166	3,885	168	-5.3	-6.1	-8.3	-7.6
1,200	3,866	152	4,067	162	4,185	164	-5.2	-6.4	-8.3	-7.9
1,300	4,122	148	4,337	158	4,454	160	-5.2	-6.6	-8.1	-8.3
1,400	4,369	144	4,620	154	4,744	156	-5.8	-6.6	-8.6	-8.3
1,500	4,609	141	4,879	151	5,011	153	-5.8	-6.7	-8.7	-8.6

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 14-15 to Figure 14-17. The plots demonstrate close correlation between the modelled uranium grades and sample composites. It is apparent that the model has smoothed the composite grades, which is to be expected due to the volume variance effect. The Qualified Person was satisfied with the results of the model validation.

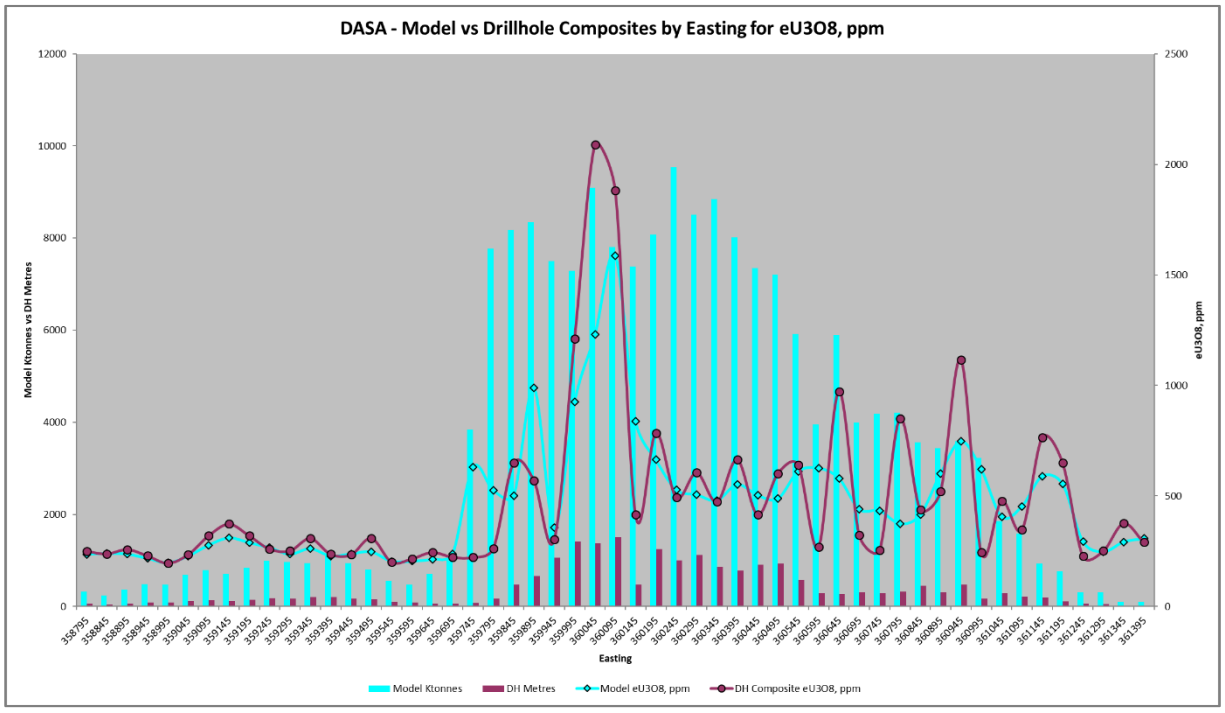


Figure 14-15: Swath plot for 50 m easting sections

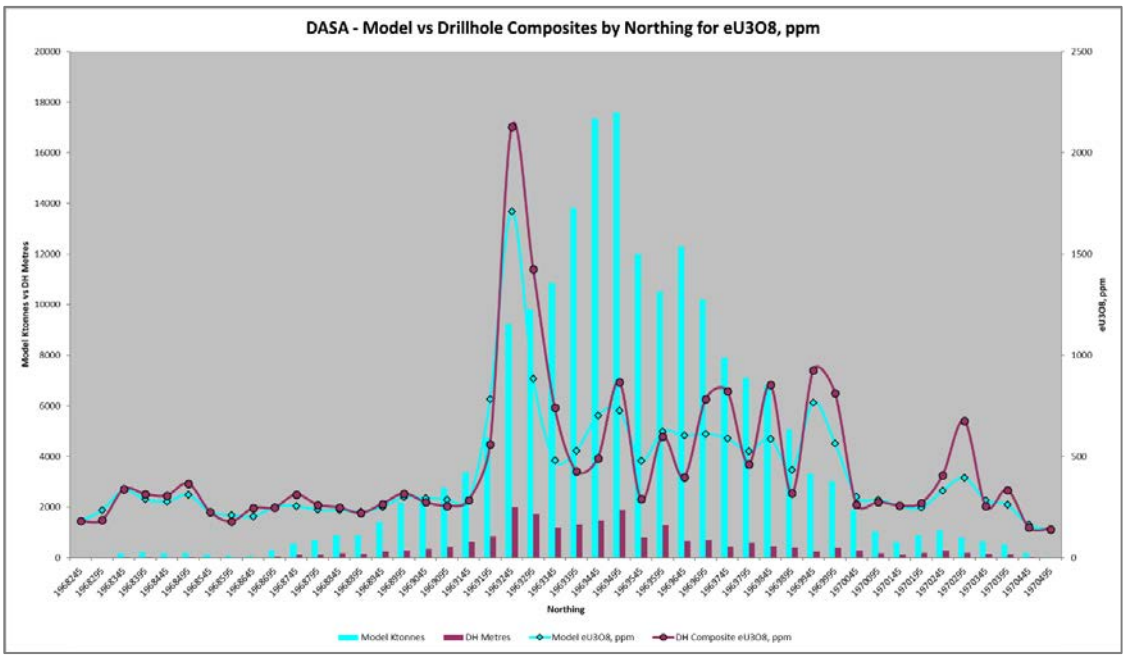


Figure 14-16: Swath plot for 50 m northing sections

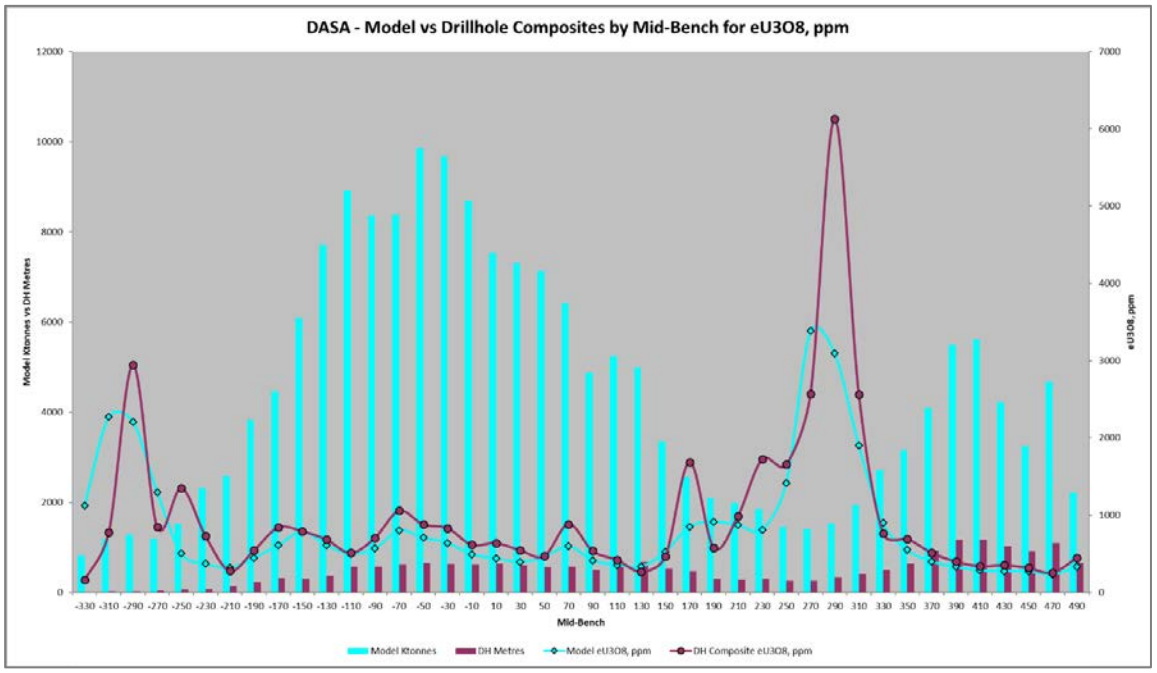


Figure 14-17:Swath plot for 20 m flitches

14.16. Reasonable Prospects of 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 taking into account possible extraction scenarios and processing recoveries.

To ensure that reported resources have a reasonable prospect of economic extraction, a conceptual pit shell was developed for the 2019 block model using the existing topographic surface provided by GAC. Estimated block values and economic parameters provided by GAC and deemed reasonable by QP were used to generate an ultimate undiscounted pit shell. The results from the pit shell analysis were used solely for the purpose of reporting mineral resources that have reasonable prospects for economic extraction.

Open Pit Input Parameters

The conceptual pit shell study was based on the following information, provided by GAC, or generated by CSA Global:

Classified block model (generated by CSA Global)

Topographic surface (GAC)

Input economic parameters (provided by GAC and CSA Global).

The input parameters for the base case are shown in Table 14-8. The costs are considered reasonable estimates for a project of this type and scale but have not been informed by any kind of formal mining study of the Project. CSA Global elected to use a higher uranium price than was in place at that time to reflect the uncertainty over when the Project will be developed and the likely pricing scenarios that could be in play in the future. Considering the current Uranium market trend, the Qualified Person believes selection of a higher price scenario is warranted.

Table 14-8: Pit optimization parameters

Parameters	Units	Values
1. Mining		
Mineralized material mining cost	$/t	2.5
Waste mining cost	$/t	2.5
Mining losses	%	5
Mining dilution	%	5

Parameters	Units	Values
2. Processing		
Processing cost	$/t	24
General and administration costs	$/t	5
Processing recovery: U_3O_8	%	95
3. Pricing		
Element price for U_3O_8	$/lb	45
Selling cost	$/unit	0
4. Other		
Bulk density parameter	t/m3	2.36
General pit slope	°	45

Open Pit Optimization Process

The pit optimization was carried out using the Mining module of the Micromine version 18.0 software application using the Lerch-Grossman algorithm.

The Lerch-Grossman algorithm is an industry-standard optimization technique used in mining and exploration. It is based on graph theory and is one of the widely used methods that guarantees the detection of the true optimum pit.

In the Lerch-Grossmann algorithm, directed arcs indicate which blocks need to be removed before a block can either be mined and processed, or be dumped as waste. Each block in the model is assigned a revenue value based on the grade of that block and metal price, and then all associated costs are subtracted from the revenue, so that all blocks are assigned a positive or negative dollar value. If the dollar value is positive, that block could potentially be mined profitably providing that all the blocks above do not make a loss if mined. The model pit slopes are specified in terms of the blocks that must be removed to provide access to each block within the block model.

Pit optimization requires that a fixed cost/value be associated with each block. The value of a waste block usually defines the cost of mining and disposal (dumping, reclaiming, etc.). A negative value indicates a loss.

The value of a mineralized block is usually defined by the profit from the mineral sale, minus the costs associated with mining and processing. A block will have a negative value if the costs are greater than the profit. It makes sense to consider it a mineralized block if the loss is less than it would be if it was treated as a waste block. In general, the pit optimization process treats negative blocks as waste, and positive blocks as mineralized feed.

The pit optimization process determines the “ultimate pit”. The ultimate pit is a pit that gives the highest possible undiscounted surplus between revenue and summary of operating costs but does not consider some scheduling constraints and discounting. Capital costs are also not considered for the ultimate undiscounted pit.

The ultimate discounted pit shell for the base case scenario was 1,900 m-long x 670 m-wide and approximately 670 m-deep. It demonstrates that there may be viable open pit option but also there are significant resources outside of the open pit limits that may be suitable for underground mining or possibly for in-situ leach operation (Figure 14-18).

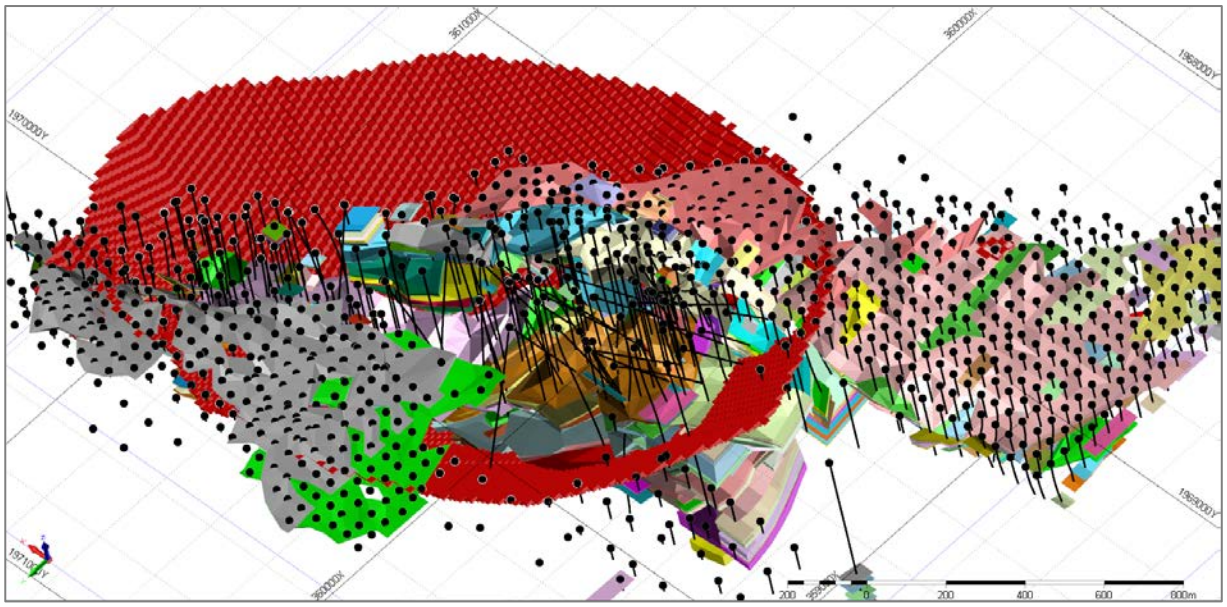


Figure 14-18: Ultimate pit shell and mineralization wireframes (looking southeast)

Underground Resources

Mineral Resources for underground mining were estimated outside of the limits of the pit shell and above the cut-off grade 1,200 ppm U_3O_8. The cut-off grade was calculated using the economic parameters in Table 14-9 and a U_3O_8 price of 45 $/lb.

Table 14-9: Input parameters for underground mining

Parameters	Units	Values
1. Mining		
Mining cost	$/t	71
Mining losses	%	10
Mining dilution	%	15
2. Processing		
Processing cost	$/t	24
General and administration costs	$/t	5
Processing recovery: U_3O_8	%	95
3. Pricing		
Elements price for U_3O_8	$/lb	45
Selling cost	$/unit	0
4. Other		
Bulk density parameters	t/m^3	2.36

14.17. Summary

The pit optimization in this study was carried out to support the case for eventual economic extraction of deposit via open pit mining, and also to separate the Mineral Resources into those that are suitable for open pit and for underground mining methods.

The author and Qualified Person deem that there are reasonable prospects for eventual economic extraction on the following basis:

There is substantial mineralization that is close to surface and therefore amenable to either lower cost open pit mining or potentially for in-situ leaching technology

The cut-off grades adopted for reporting (320 ppm for open cut mining and 1,200 ppm for underground method mining) are considered reasonable, given the Mineral Resource is likely to be exploited by combined open cut and underground mining methods and processed using leaching techniques.

14.18. Mineral Resource Statement

Mineral Resources for the Dasa deposit were estimated assuming that parts would be exploited by open cut mining and the deeper, higher-grade parts via underground mining.

Open pit Mineral Resources were estimated within the limits of conceptual ultimate pit shell and above the cut-off grade of 320 ppm U_3O_8. The cut-off was chosen based on the economic parameters in the Table 14-8 and U_3O_8 price of $45/lb.

Mineral Resources for underground mining were estimated outside of the limits of pit shell and above the cut-off grade of 1,200 ppm U_3O_8. The cut-off grade was calculated using the economic parameters in the Table 14-9 and U_3O_8 price of $45/lb.

The Mineral Resources for open pit and underground mining are shown in Table 14-10. The distribution of the Mineral Resources is displayed in Figure 14-19. It shows the material that occurs within the conceptual optimized pit and the distribution of potential underground resources outside of the constraining pit shell above the corresponding cut-offs (320 and 1,200 ppm U_3O_8).

Table 14-10: Dasa Mineral Resources for open pit and underground mining with an Effective Date of 1 June 2019

Mineral Resource 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 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, (CSA Global).

The Effective Date of the MRE is 1 June 2019.

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

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.

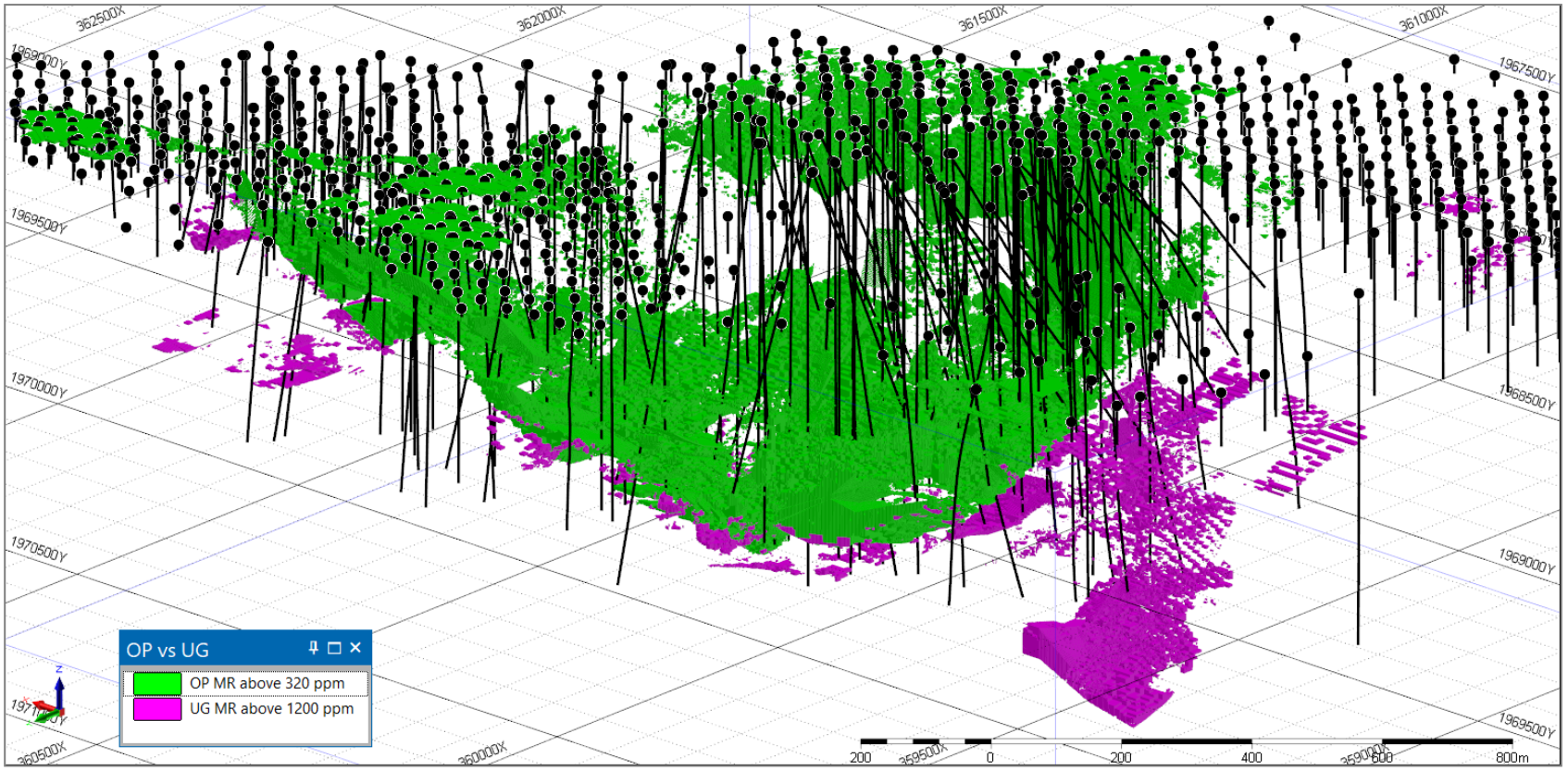


Figure 14-19: Distribution of open cut and underground mineral resources above the selected cut-offs (looking southeast)

14.19. 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 studies and 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, grade model methodology).

The REF 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 for uranium grades supports the reliable calculation of eU_3O_8 grades, that were the basis of the modelled grades, as well as the estimation of the uranium equilibrium 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 lithologic contacts, mineralization domains, modelling of internal waste domains, etc.).

Changes to design parameter assumptions that pertain to the resource constraining conceptual pit shell.

Changes to process recovery estimates if the metallurgical recovery in certain domains is less or greater than currently assumed.

14.20. Grade-Tonnage Sensitivity Report

The grade-tonnage sensitivity report for the Dasa deposit is given in Table 14-11 at a range of cut-off grades between 200 ppm eU_3O_8 and 15,000 ppm eU_3O_8 and subdivided by adopted resource classification. The cut-off grades were applied to the eU_3O_8 values in the block model.

The grade-tonnage sensitivity curves for Indicated and Inferred Resource categories separately for eU_3O_8 grades and contained metal are shown in Figure 14-20 to Figure 14-23.

Table 14-11: Dasa grade-tonnage report

Cut-off (eU_3O_8, ppm)	Category	Volume (Mm3)	Tonnes (Mt)	Bulk density (t/m^3)	eU_3O_8 (ppm)	Contained eU_3O_8 (Mlb)
200	Indicated	22.2	52.3	2.36	1,035	119.3
	Inferred	25.0	59.0	2.36	893	116.2
250	Indicated	17.8	41.9	2.36	1,235	114.2
	Inferred	19.9	46.9	2.36	1,066	110.2
300	Indicated	14.6	34.4	2.36	1,446	109.6
	Inferred	15.9	37.6	2.36	1,260	104.6
320	Indicated	13.6	32.0	2.36	1,530	108.0
	Inferred	14.8	35.0	2.36	1,333	102.7
400	Indicated	10.6	24.9	2.36	1,865	102.4
	Inferred	11.2	26.5	2.36	1,644	96.1
500	Indicated	8.2	19.4	2.36	2,265	97.0
	Inferred	8.8	20.7	2.36	1,979	90.4
600	Indicated	6.8	16.0	2.36	2,631	92.9
	Inferred	7.3	17.1	2.36	2,281	86.0
700	Indicated	5.8	13.7	2.36	2,971	89.6
	Inferred	6.3	14.8	2.36	2,541	82.7
800	Indicated	5.1	12.0	2.36	3,291	86.8
	Inferred	5.5	12.9	2.36	2,796	79.6
900	Indicated	4.5	10.6	2.36	3,598	84.3
	Inferred	4.9	11.5	2.36	3,044	76.9

Cut-off (eU_3O_8, ppm)	Category	Volume (Mm3)	Tonnes (Mt)	Bulk density (t/m^3)	eU_3O_8 (ppm)	Contained eU_3O_8 (Mlb)
1,000	Indicated	4.1	9.6	2.36	3,885	82.1
	Inferred	4.3	10.2	2.36	3,308	74.2
1,100	Indicated	3.7	8.6	2.36	4,198	79.9
	Inferred	3.9	9.2	2.36	3,553	71.9
1,200	Indicated	3.3	7.9	2.36	4,483	78.0
	Inferred	3.6	8.4	2.36	3,783	69.9
1,300	Indicated	3.1	7.3	2.36	4,771	76.3
	Inferred	3.3	7.7	2.36	3,992	68.1
1,400	Indicated	2.8	6.7	2.36	5,053	74.6
	Inferred	3.0	7.2	2.36	4,195	66.5
1,500	Indicated	2.6	6.2	2.36	5,328	73.1
	Inferred	2.7	6.3	2.36	4,563	63.7
2,000	Indicated	1.9	4.6	2.36	6,624	66.8
	Inferred	1.9	4.5	2.36	5,713	56.8
2,500	Indicated	1.5	3.6	2.36	7,849	61.9
	Inferred	1.4	3.4	2.36	6,838	51.4
3,000	Indicated	1.2	2.9	2.36	8,973	58.0
	Inferred	1.2	2.7	2.36	7,833	47.4
3,500	Indicated	1.0	2.5	2.36	10,036	54.7
	Inferred	1.0	2.4	2.36	8,490	45.0

Cut-off (eU_3O_8, ppm)	Category	Volume (Mm3)	Tonnes (Mt)	Bulk density (t/m^3)	eU_3O_8 (ppm)	Contained eU_3O_8 (Mlb)
5,000	Indicated	0.7	1.6	2.36	13,186	46.8
	Inferred	0.7	1.6	2.36	10,805	37.2
10,000	Indicated	0.2	0.6	2.36	24,401	31.1
	Inferred	0.3	0.8	2.36	14,598	25.3
15,000	Indicated	0.1	0.3	2.36	34,236	24.3
	Inferred	0.0	0.1	2.36	21,493	4.0

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

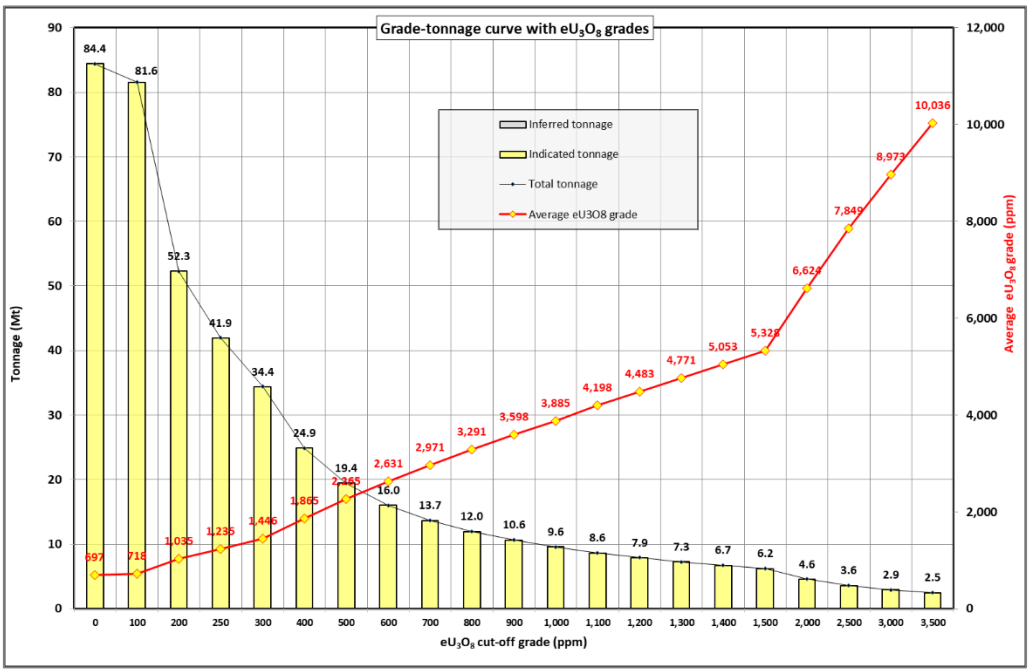


Figure 14-20: Grade-tonnage curve for Dasa showing eU_3O_8 grades – Indicated category

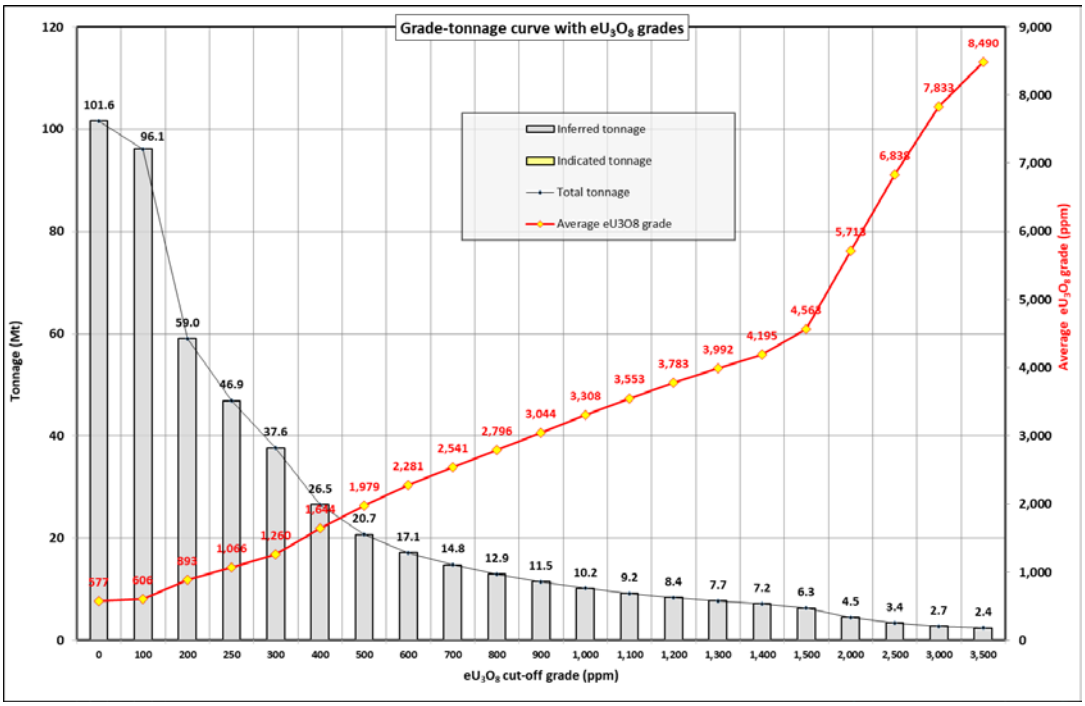


Figure 14-21: Grade-tonnage curve for Dasa showing eU_3O_8 grades – Inferred category

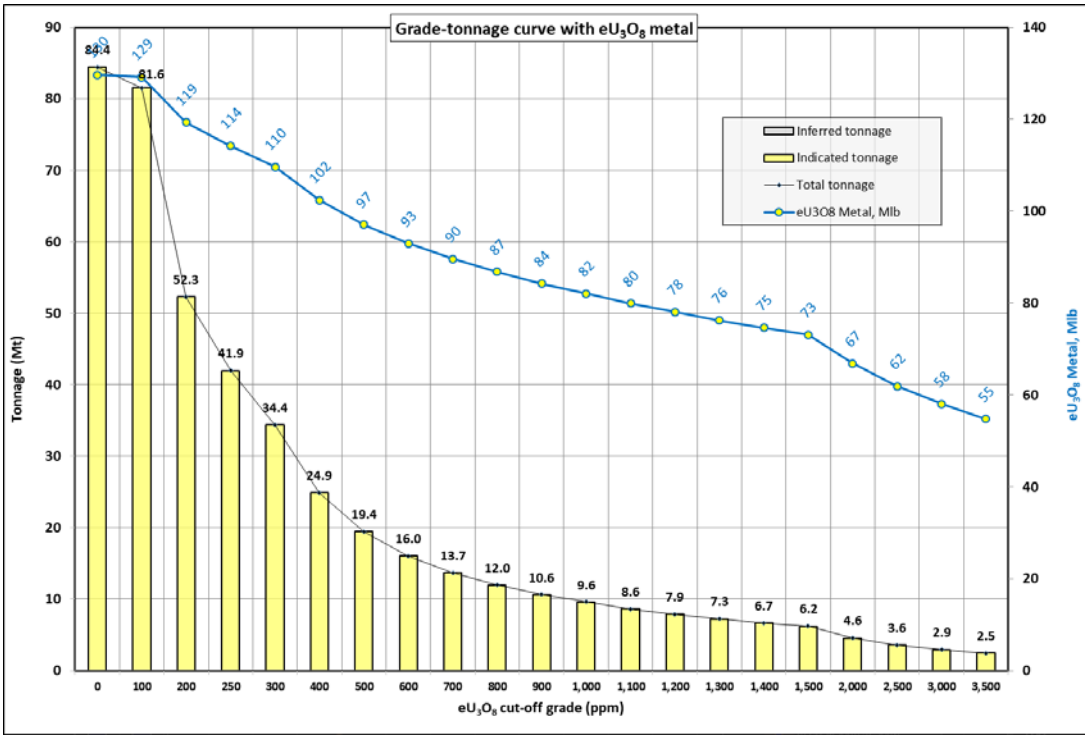


Figure 14-22: Grade-tonnage curve for Dasa showing eU_3O_8 metal – Indicated category

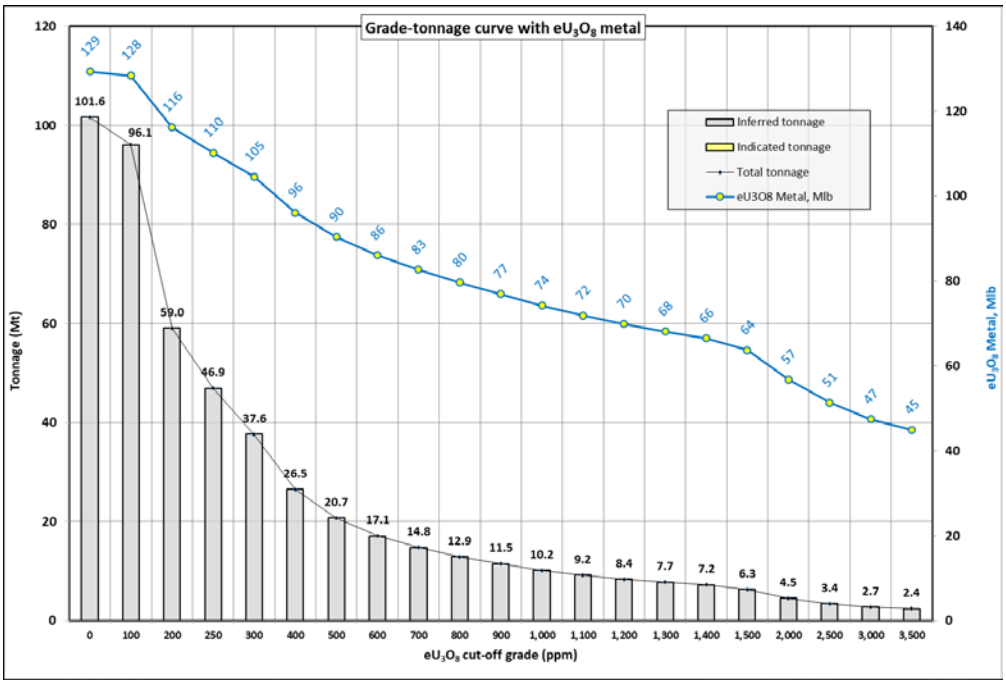


Figure 14-23: Grade-tonnage curve for Dasa showing eU_3O_8 metal – Inferred category

14.21. Conclusions

The Mineral Resource reported herein is an update of the previous estimate based on additional drilling results and further development of the geological model.

The Author/QP and CSA Global believes this Mineral Resource is a reliable estimate of the mineralization present at the DASA, 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. Additionally, mineralization potential exists within the project along strike to the north and south, as well as within the graben providing significant upside potential. As such, the Author/QP and CSA Global concludes that additional exploration work is warranted at the Project to enlarge the resource and upgrade current Mineral Resource to a higher classification particularly in areas suitable for the underground mining method.

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, the sub-sections below summarise this work undertaken in order 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 longhole 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 22.5m vertical intervals with a footwall drive developed along strike 20m from the stopes. Stope access crosscuts will be developed at 16.5m 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			
From host rock (waste)	%	2.1	At average grade below cut-off
From backfill	%	2.3	Zero grade
Ore recovery (from mining)	%	95	
Cut-off grade	ppm	2074	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 minimised at all times.

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 5 457 ppm U_3O_8. Thus, annual production (U_3O_8 equivalent) is approximately 4 134 000 lb/a

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 gives a summary of the capital costs while Table 15-4 gives a summary of the operating costs.

Table 15-3: Capital Costs

($millions)	Initial	Sustaining	Total
Mining	55	114	169
Processing	90	6	96
Infrastructure	18		18
Total Direct Capital Costs	**163**	**120**	**283**
Indirect & Owner's Costs	27		27
Total Direct & Indirect Capital Costs	**190**	**120**	**310**
Contingency	18	17	35
Total Capital Costs	**208**	**137**	**345**

Table 15-4: Operating Costs

Cost Item	$/lb U_3O_8 Recovered	$/tonne of Feed
Mining cost	6.07	65
Processing cost	5.85	62
Overhead cost	3.72	40
Cash costs before royalties	**15.64**	**167**
Royalties	3.27	35
Total cash costs	**18.91**	**202**
Sustaining capital	3.02	32
AISC	**21.93**	**234**

Metal Prices

The metal price used in the DFS evaluation of the Dasa Project is $35 per lb of U_3O_8.

Payable Mineral Resource Included in Mine Layout

In order 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_3O_8 Price ($/lb)	**35**
lb/kg	2.205
U_3O_8 Price ($ /kg)	**77.16**
Operating cost ($/t milled)	
Mining	45
Processing	54
Overheads (G&A)	35
Operating cost ($/t milled)	**134**
Breakeven recovered grade (kg/t)	1.74
Metallurgical recovery	94%
Breakeven RoM grade (ppm) before royalty	1852
Royalties (%)	12%
Breakeven after royalty	**2074**
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	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 (deg)	60	
Unplanned dilution - HW (m)	0.5	End walls only
Unplanned dilution - FW (m)	0.5	End walls only
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=300m
Stope direction (azi °)	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 ore. For a stope to be included in the mine design it needed to contain a minimum of 1000 tonnes of indicated resources. Figure below shows the stope shapes produced by DeswikSO®.

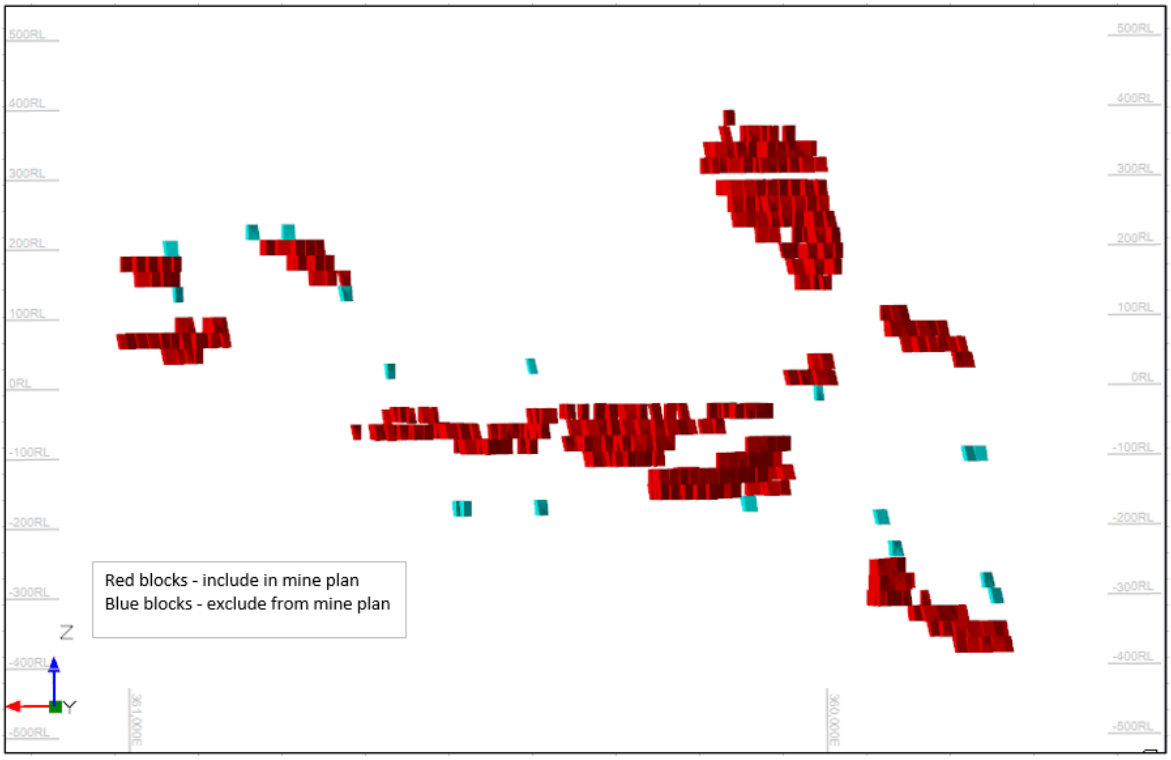


Figure 15-1: Stope shapes produced by DeswikSO®

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.

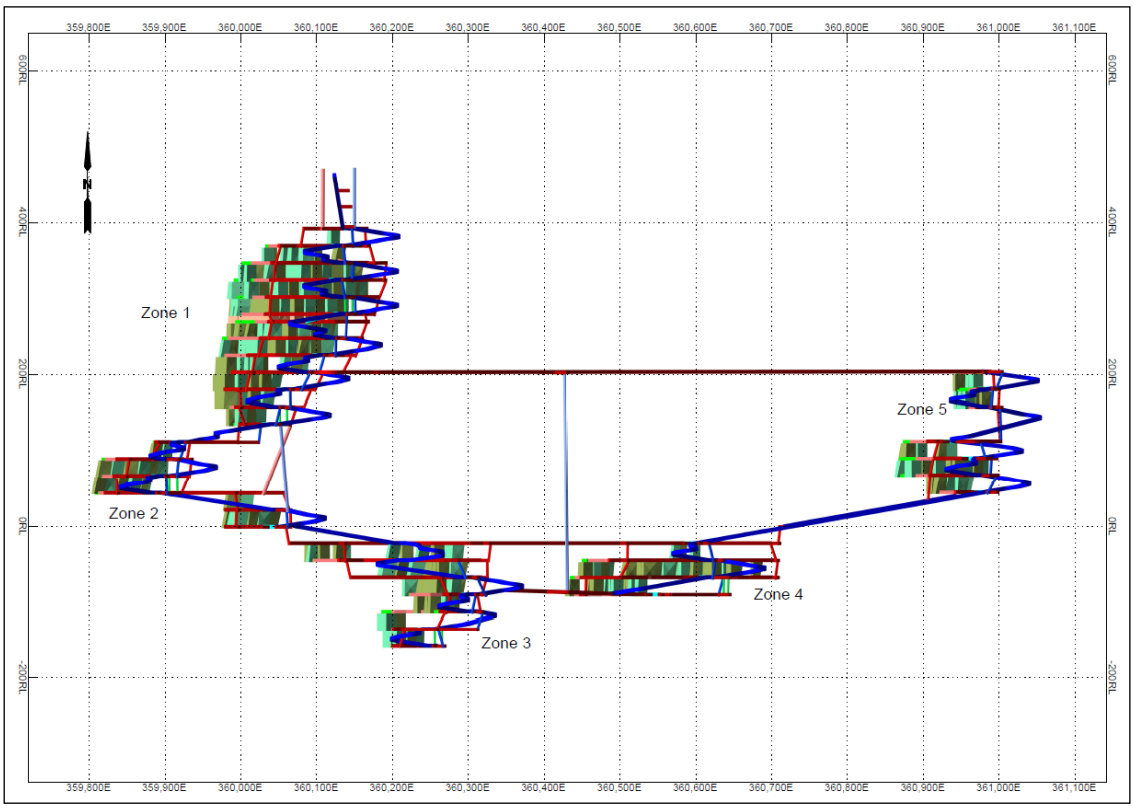


Figure 15-2: View of underground mine looking North

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	m/month	75
Level Access		60
Footwall drive		60
Stope crosscut – waste		60
Stope crosscut – ore		50
Drop raising		24

The activities were then scheduled taking into account the productivities detailed in the above table, as well as the availability of resources (mining equipment) allocated to each activity.

The key aspects in the mining schedule are:

Excavation of the boxcut commences in January 2022.

Blasting of the decline face from the boxcut commences in April 2022.The Upcast ventilation system is established by the completion of ventilation raise in January 2023.

Level development commences on the 5367 Level in November 2022.

First ore is produced from development in May 2023.

Stoping commences in February 2024 on the 5300 Level.

Steady state production of 30,000 tonnes per month of ore is achieved in March 2024.

The Phase 1 mine has a life of just over 12 years.

A graphic production profile for the full mining schedule for the underground project is presented in Figure 15-3 below.

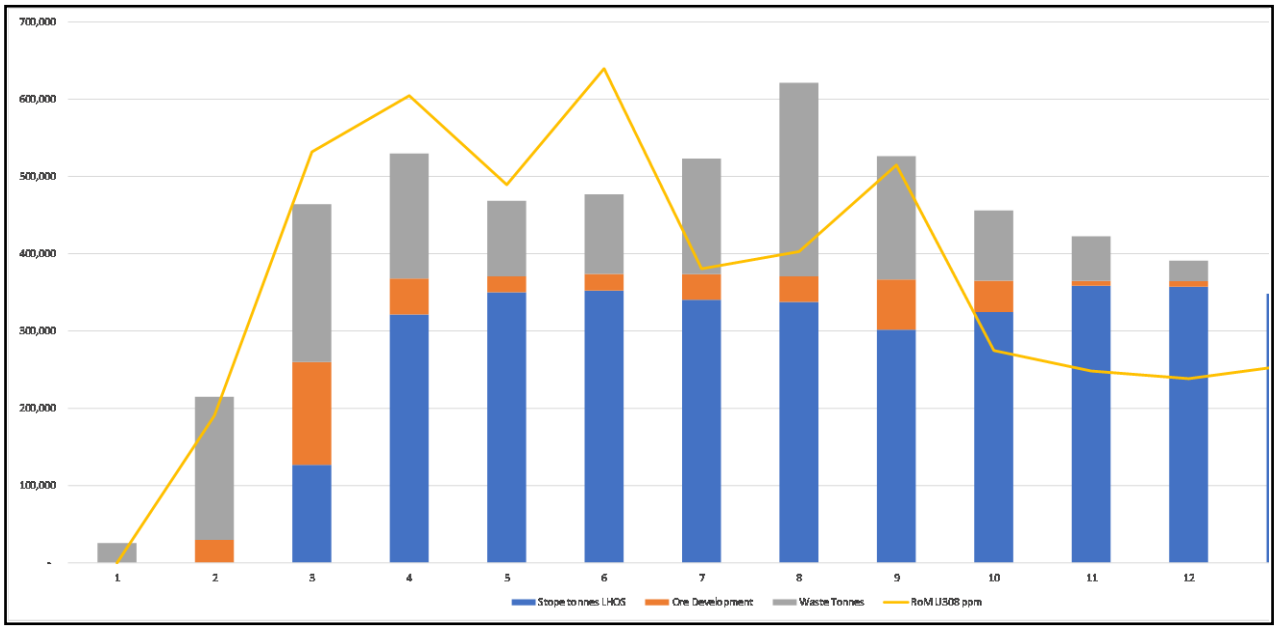


Figure 15-3: Dasa Phase 1underground production profile

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 4.4% of the RoM tonnes and 2.8% of the contained U_3O_8, the occurrence of the inferred resources in the mining schedule over time is illustrated in Figure 15-4 below.

Table 15-8: Mining Inventory by Mineral Resource Class

	RoM tonnes	U_3O_8 ppm	U_3O_8 (t)	U_3O_8 (Million lbs)
Measured	-	-	-	
Indicated	4,066,390	5,267	21,417	47.217
Inferred	187,236	3,375	632	1.393
Total Mining inventory	4,253,626	5,184	22,050	48.611

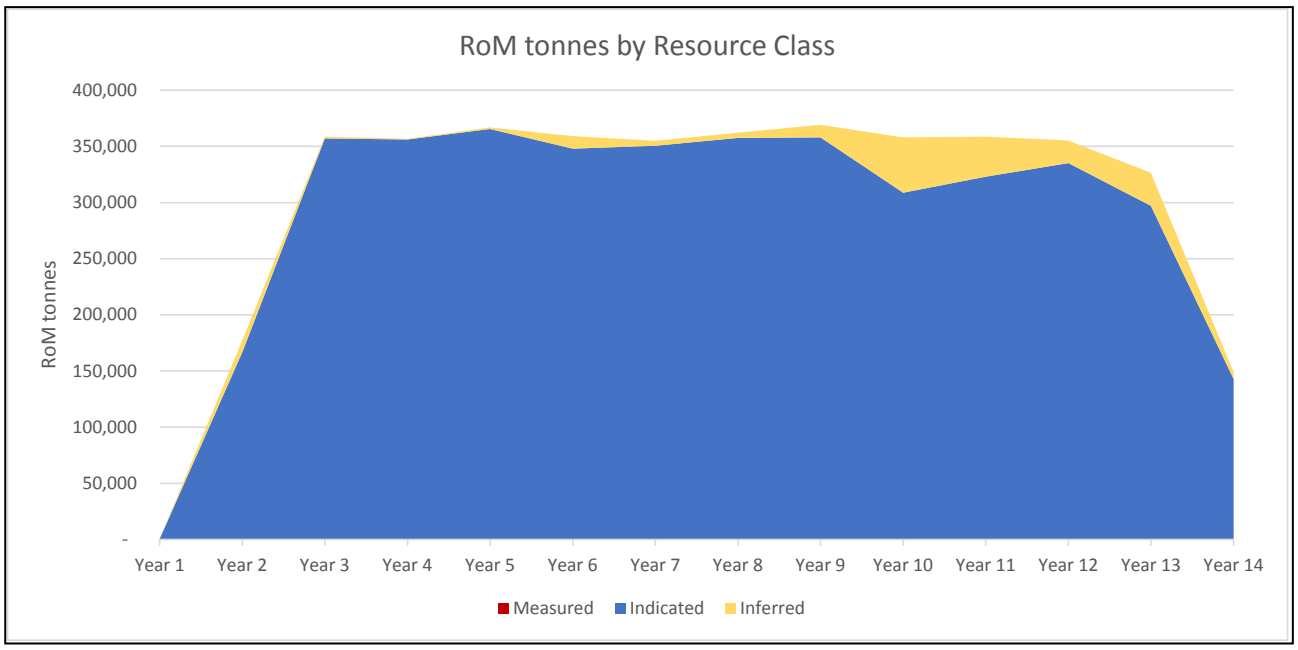


Figure 15-4: Mining Schedule showing RoM tonnes by Mineral Resource Class

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 15th November 2021. 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 (15th November 2021)

Mineral Reserve Category	RoM (Mt)	U_3O_8 (ppm)	U_3O_8 (kt)	U_3O_8 (Million lbs)
Proven Mineral Reserve	-	-	-	
Probable Mineral Reserve	4.1	5,267	21.5	47.2
Total Mineral Reserve	4.1	5,267	21.5	47.2

Notes on Mineral Reserves

Mineral Reserves are reported with an effective date of 15th November 2021

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.

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 20m 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.

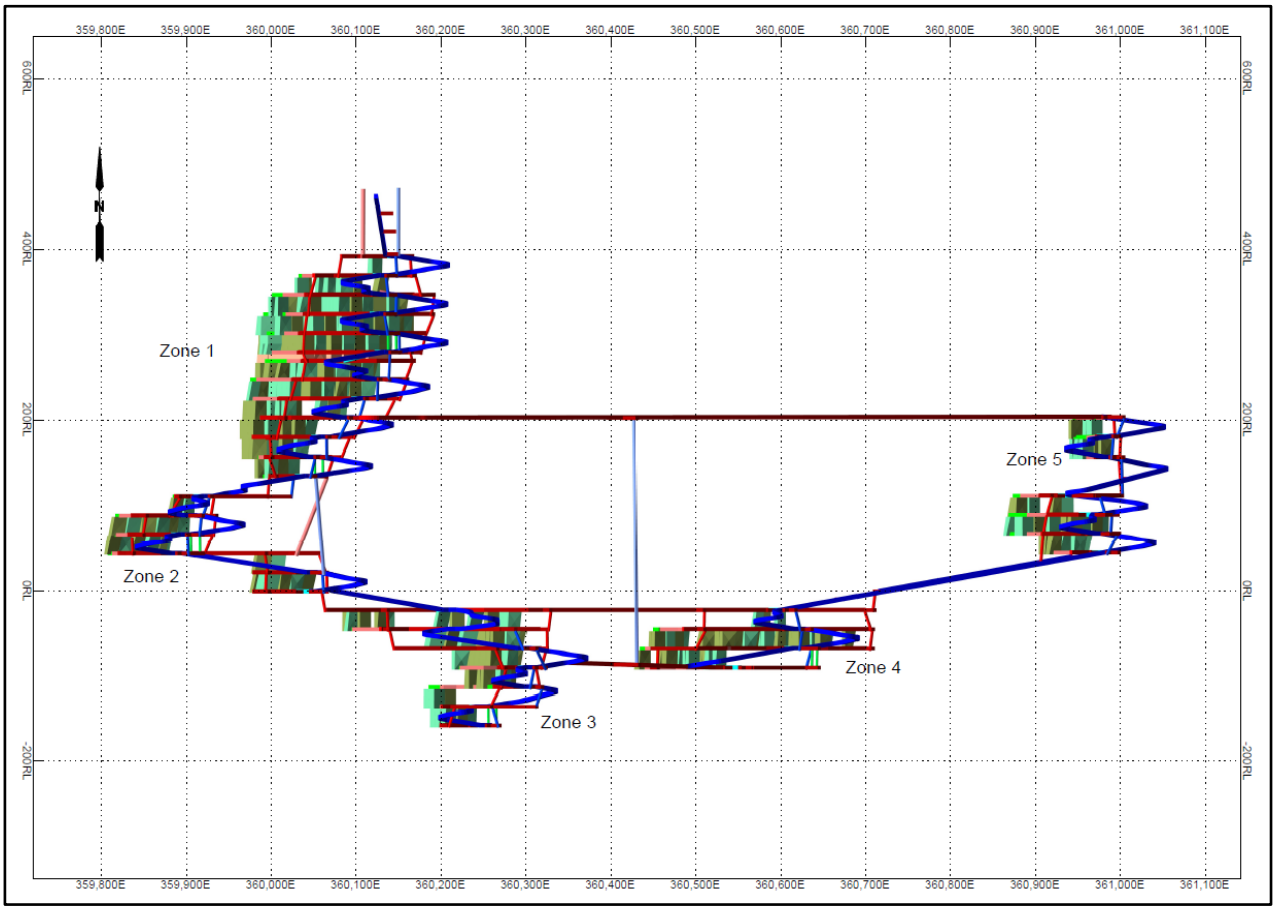


Figure 16-1: View of underground mine looking North

Mining will commence above the sill pillar on the 5275.5m 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 5200 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 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.

Figures 16-2 and 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.

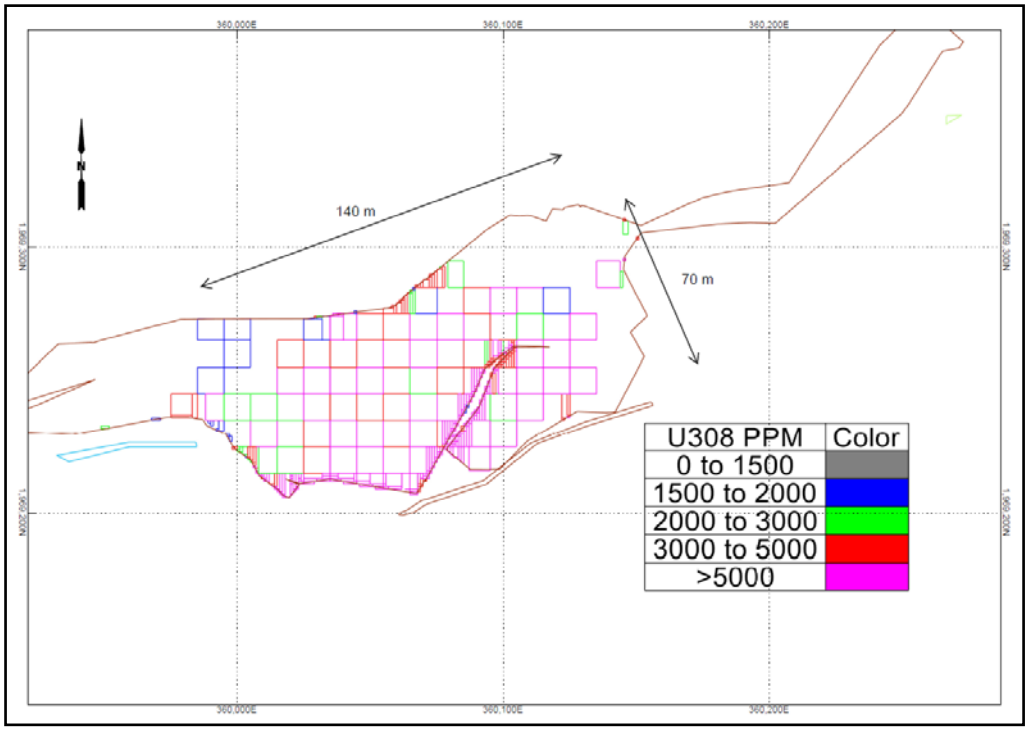


Figure 16-2: Plan View on 260 RL

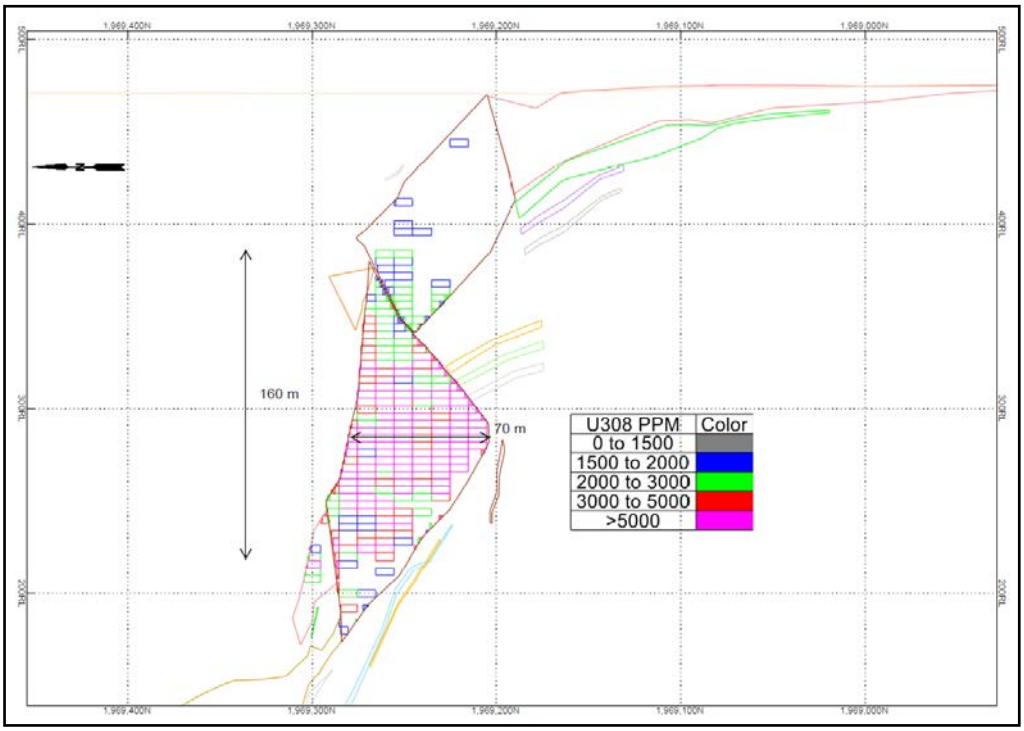


Figure 16-3: Cross Section Through Flank Zone

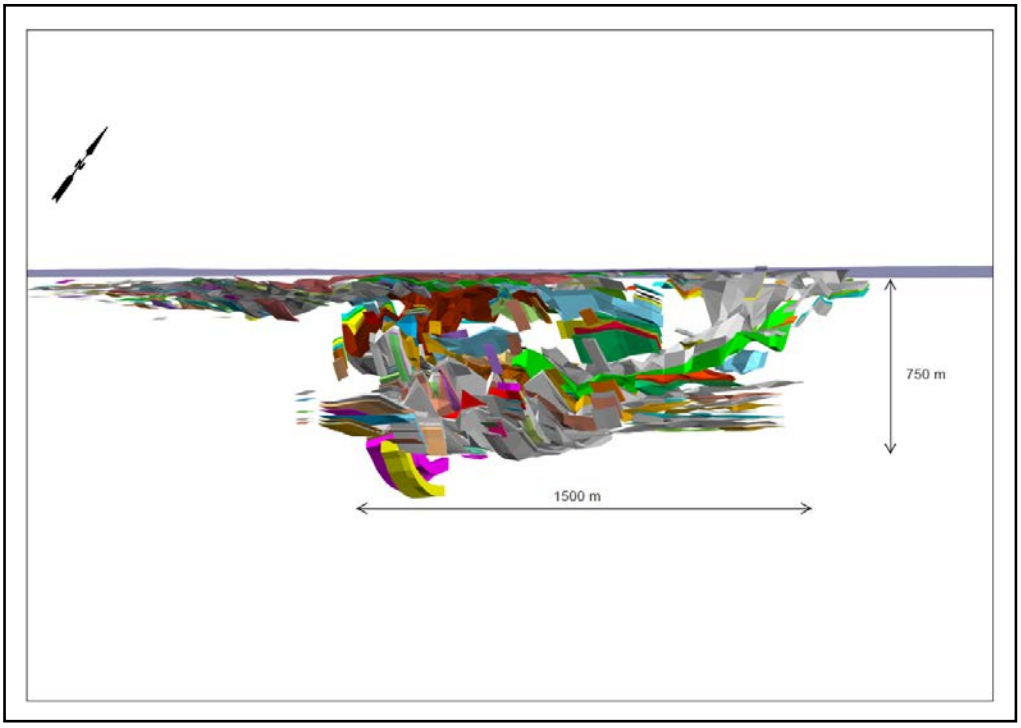


Figure 16-4: 3-D View of Orebody wireframes (Looking North-West)

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_2019.dm. A handover form was provided by CSA, who generated the model on behalf of GAC. The block model has a parent cell size of 10m x 10m x 4m but contains sub cells of 1m x 1m x 1m. 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.

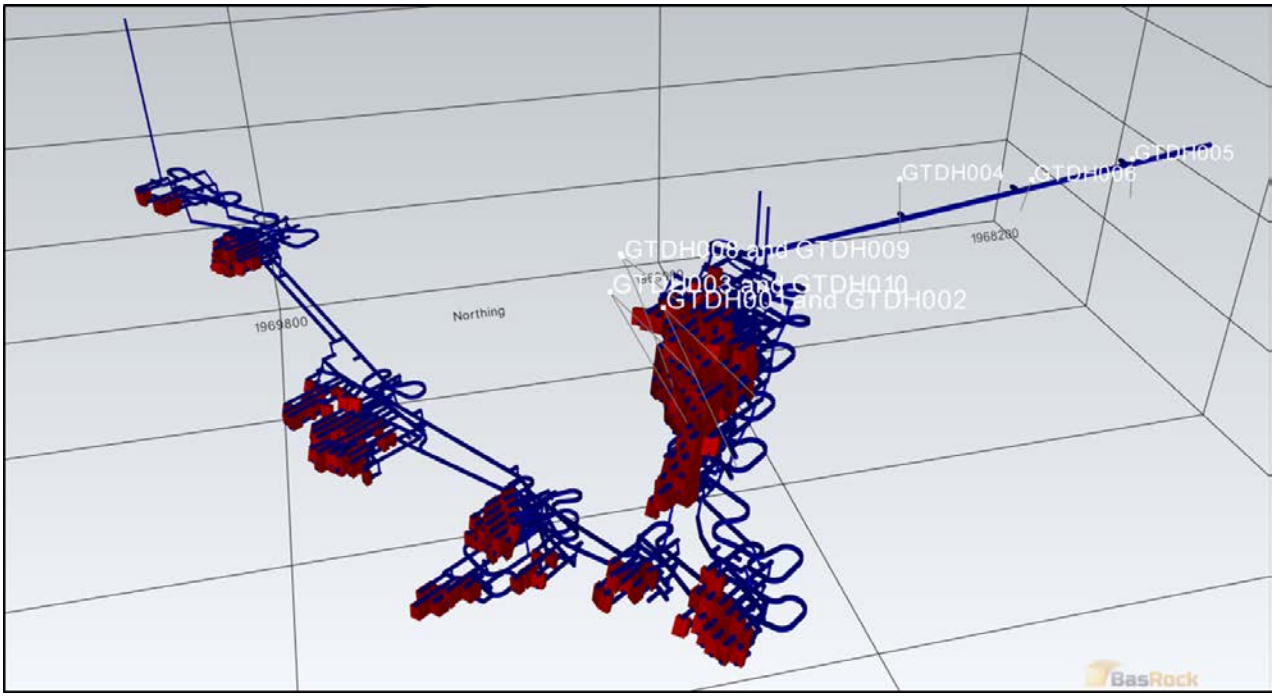


Figure 16-5: Locations of Borehole used in Geotechnical Evaluation and Design Work

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
25th %	47	57	1	5	20	60	59	3	8	27	69	50	2	11	24
50th %	53	59	2	8	22	65	62	5	15	31	75	53	3	17	27
75th %	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
25th %	82	68	4	9	29	86	71	5	14	41	86	63	2	13	36
50th %	86	72	7	15	35	90	75	11	25	50	90	67	5	25	43
75th %	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
25th %	77	68	4	6	32	82	72	7	15	44	82	63	4	16	38
50th %	86	73	9	14	37	90	77	13	26	51	90	68	6	28	44
75th %	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.

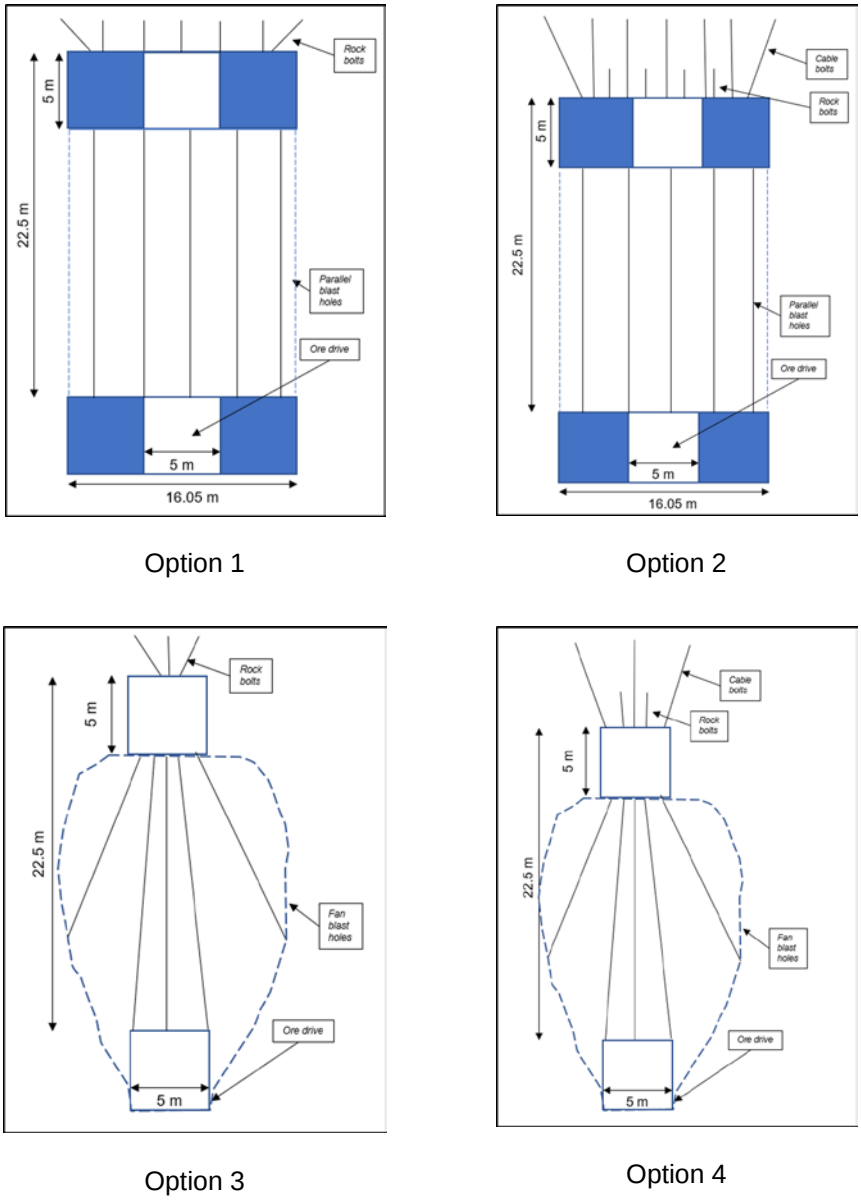


Figure 16-6: Four Options Considered for Open Stope Design

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

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. Table 16-7 below illustrated the constructed model modelled in RS2.

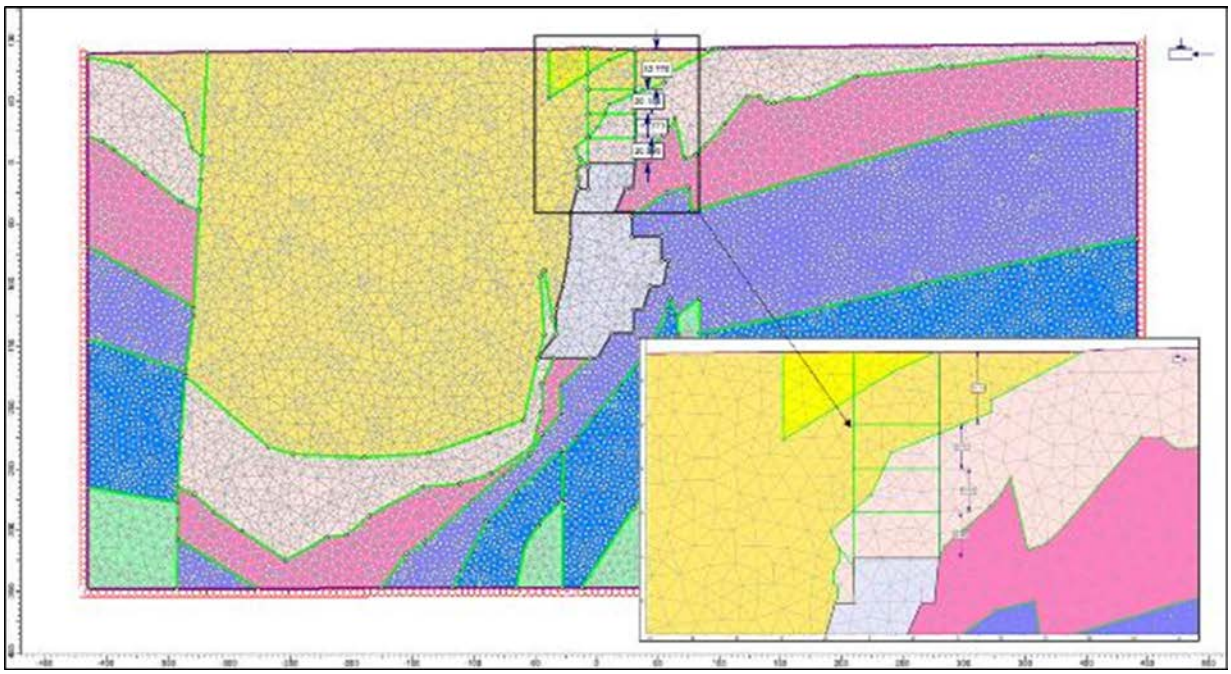


Figure 16-7: Modelling of Crown Pillar Requirements

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).

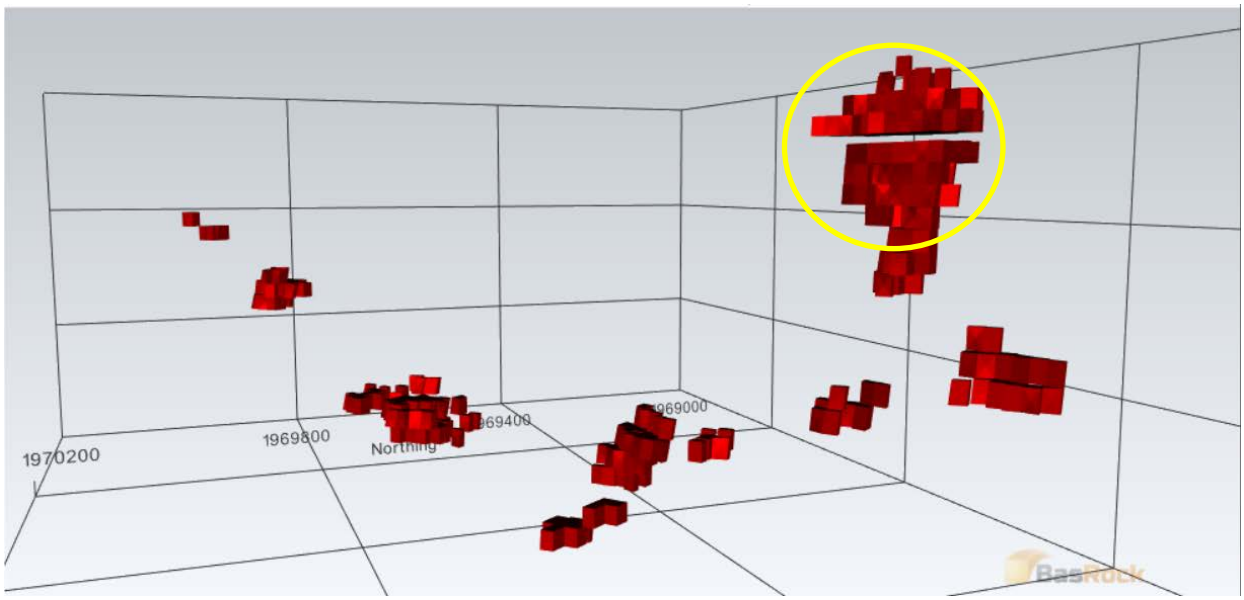


Figure 16-8: Sill Pillar Requirements

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.

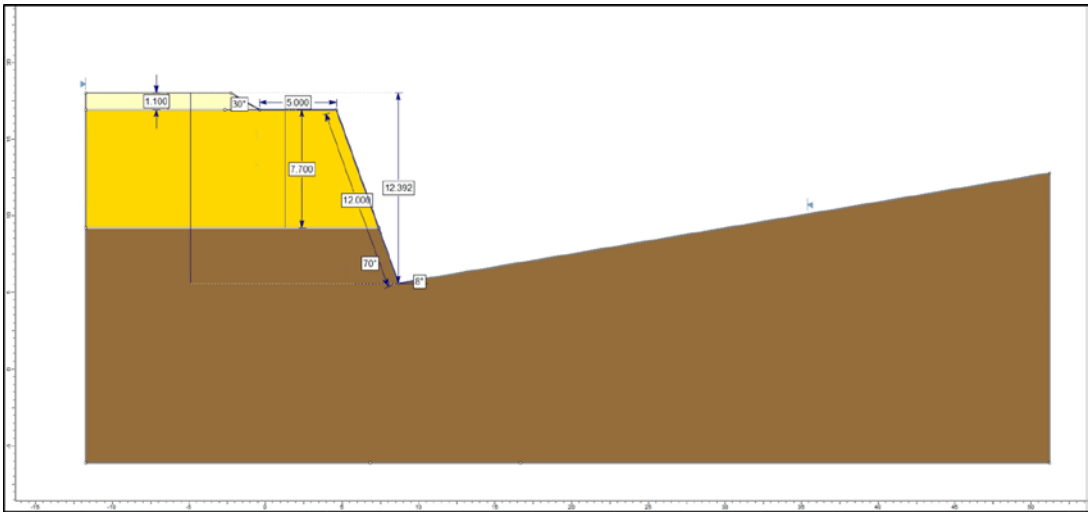


Figure 16-9: Long Section Through Boxcut

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.

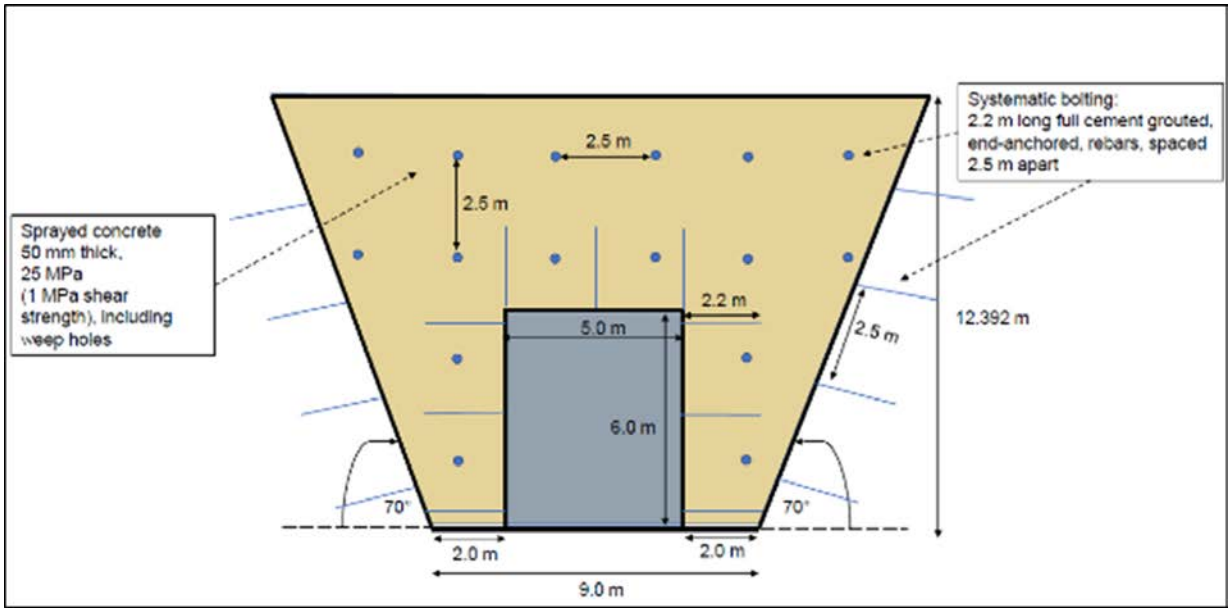


Figure 16-10: Support Requirements for Boxcut Highwall

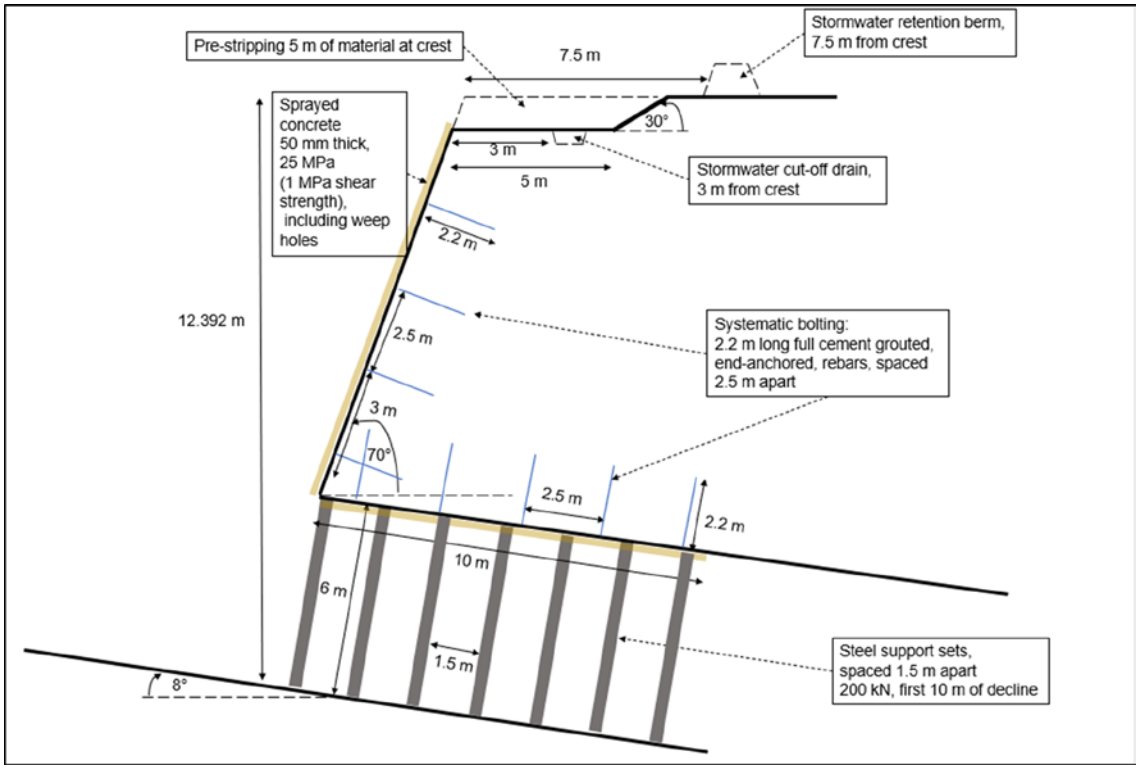


Figure 16-11: Section Through Boxcut Highwall Showing Highwall Support and Portal Area Support Requirements

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.

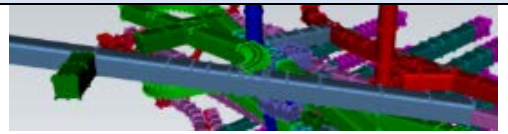
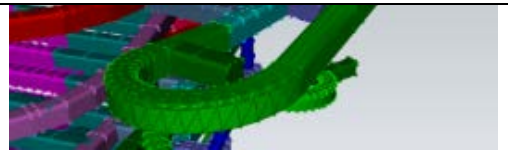
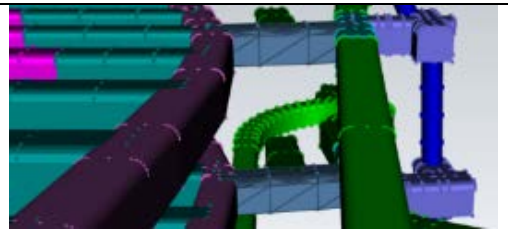
The support system for the large excavations was designed using a combination of empirical methods and design guidelines provided in Hutchinson et al (1996) such as:

- tunnel support guidelines based on Q-Index case studies (Figure 8-38) 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 16-7 and 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 (m²)	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

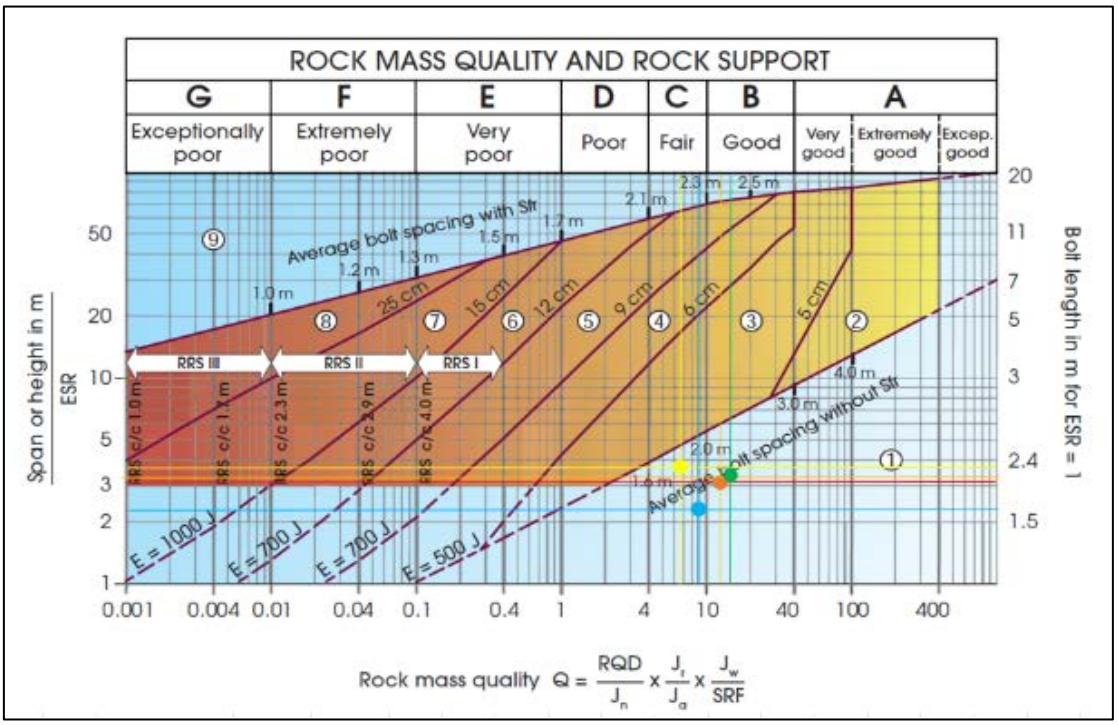


Figure 16-12: Support Guideline chart

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. Raisebore 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

Raisebore 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 Figures 16-13 and 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 raisebore in order to undertake a full ventilation shaft analysis.

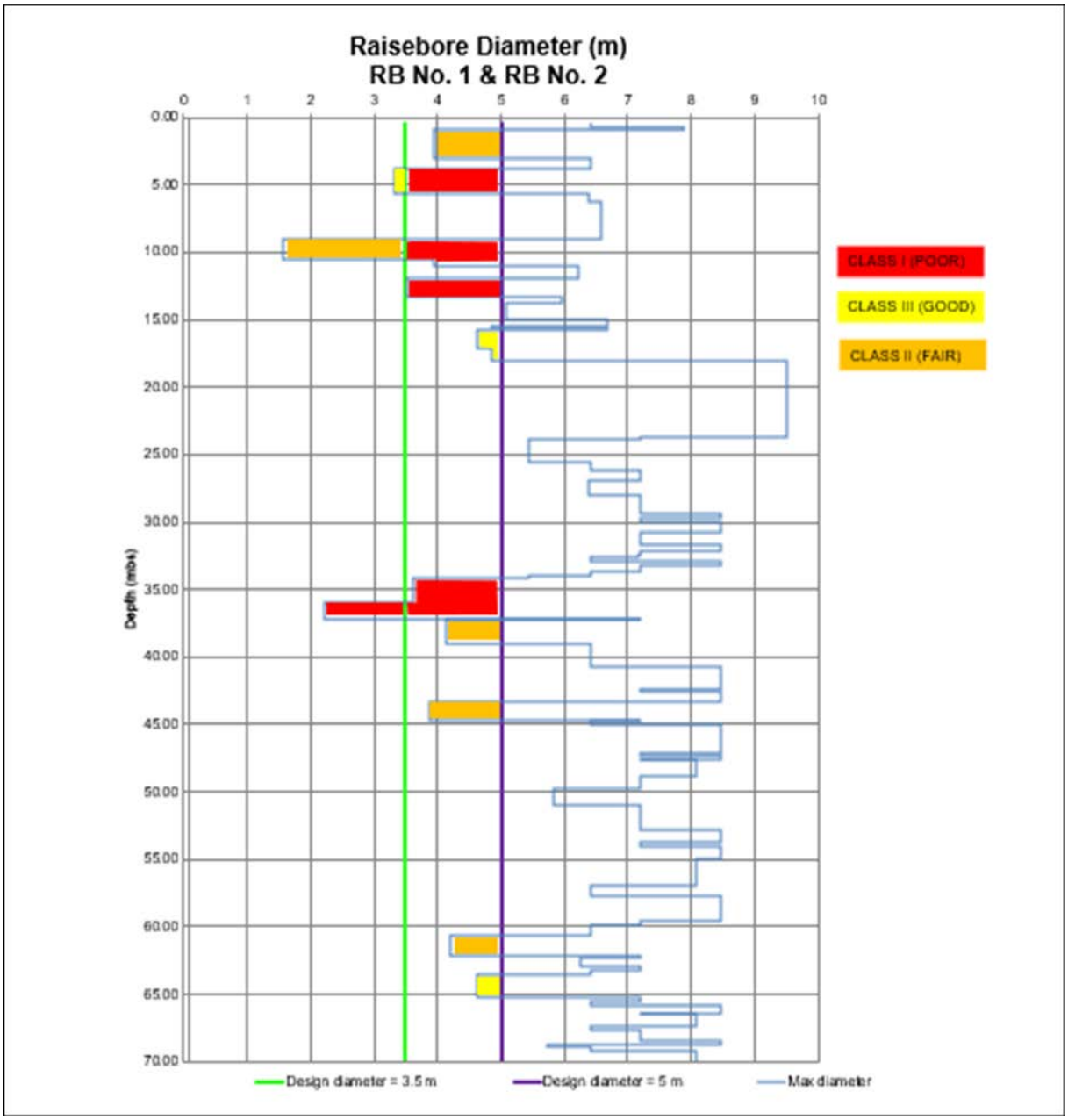


Figure 16-13: Variation of Maximum Diameter with Depth for RB1 and RB2

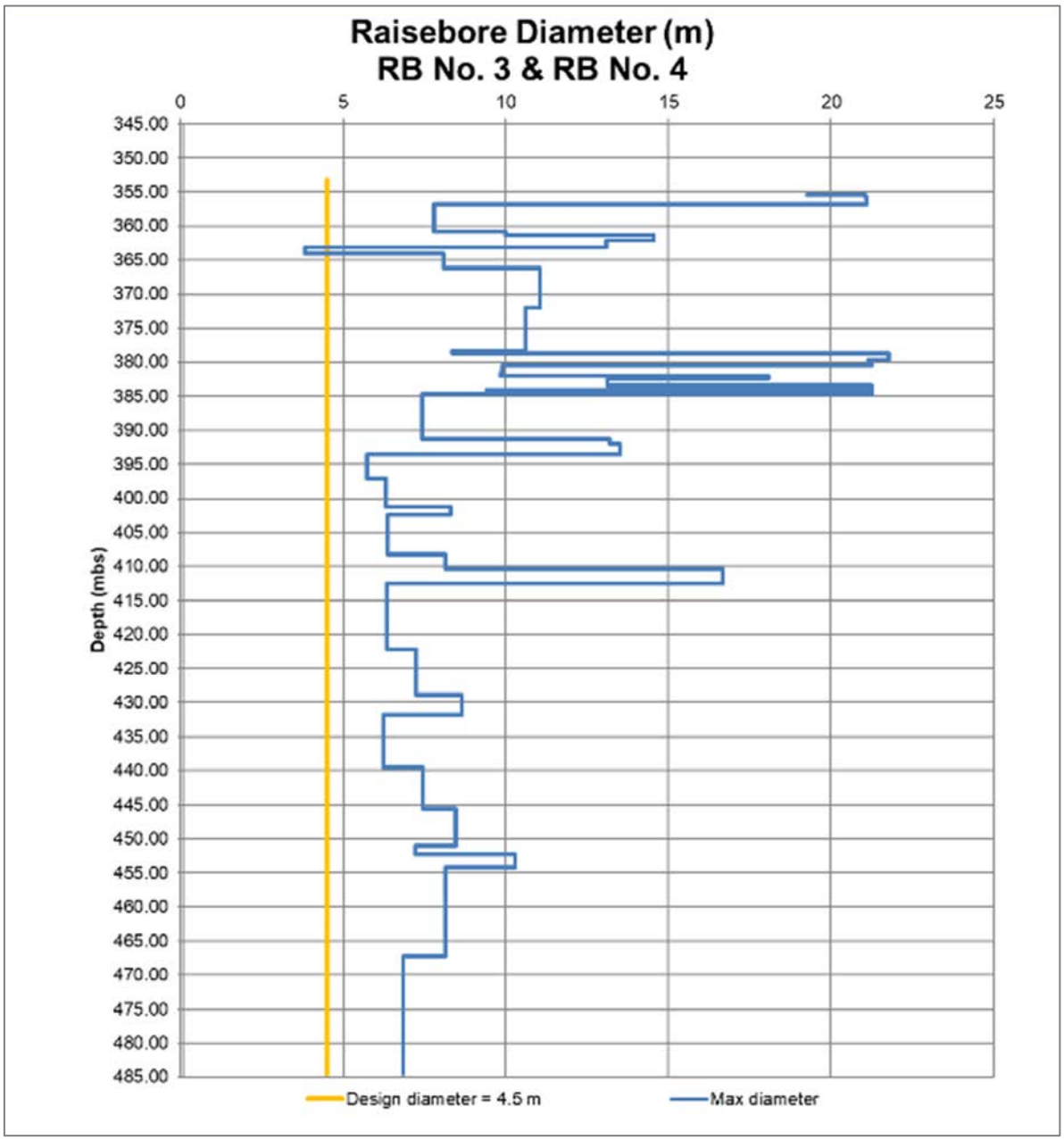


Figure 16-14: Variation of Maximum Diameter with Depth for RB3 and RB4

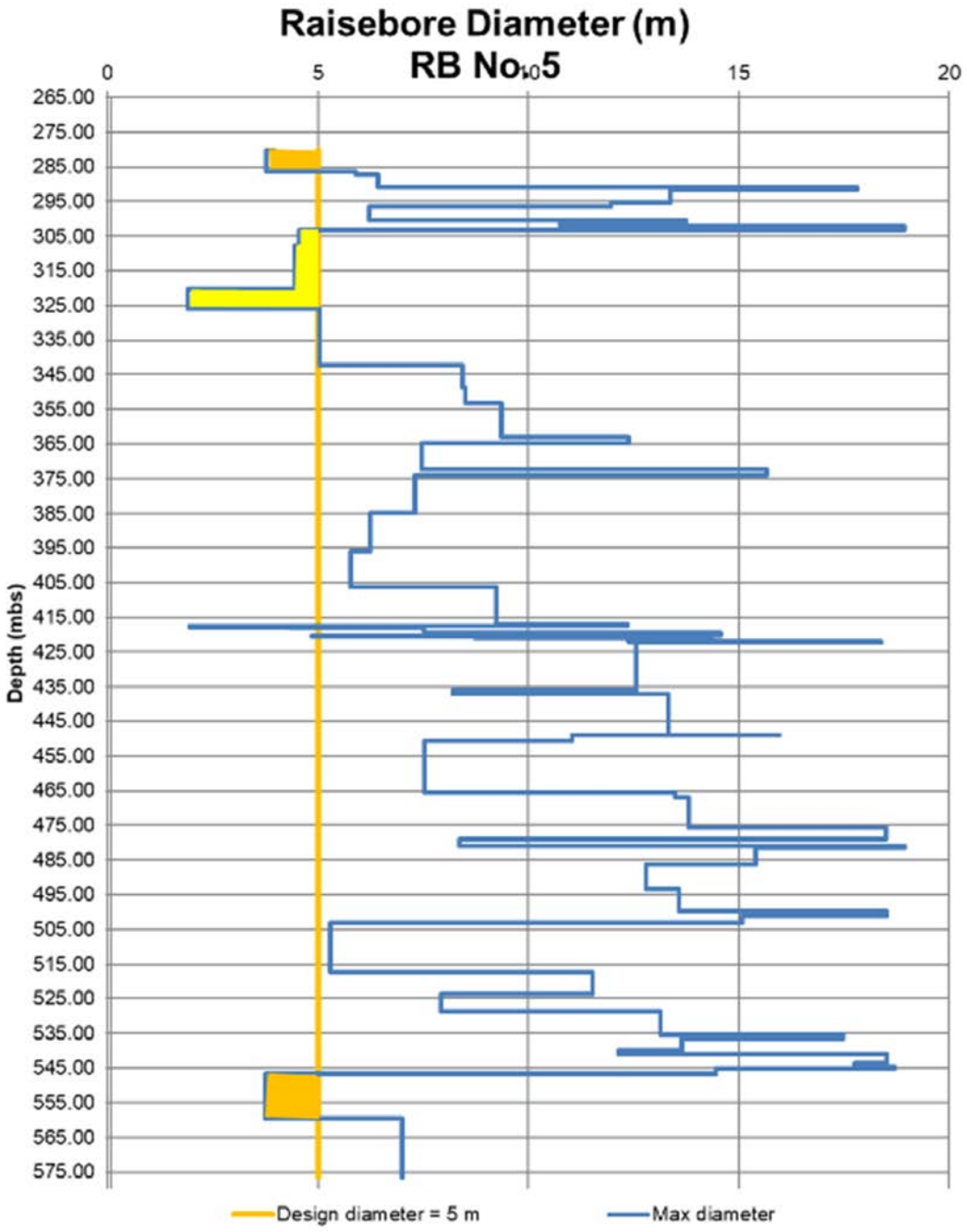


Figure 16-15: Variation of Maximum Diameter with Depth for RB5

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 as a result 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 16-14 below illustrates the layout and naming of the individual stoping zones considered during the modelling process.

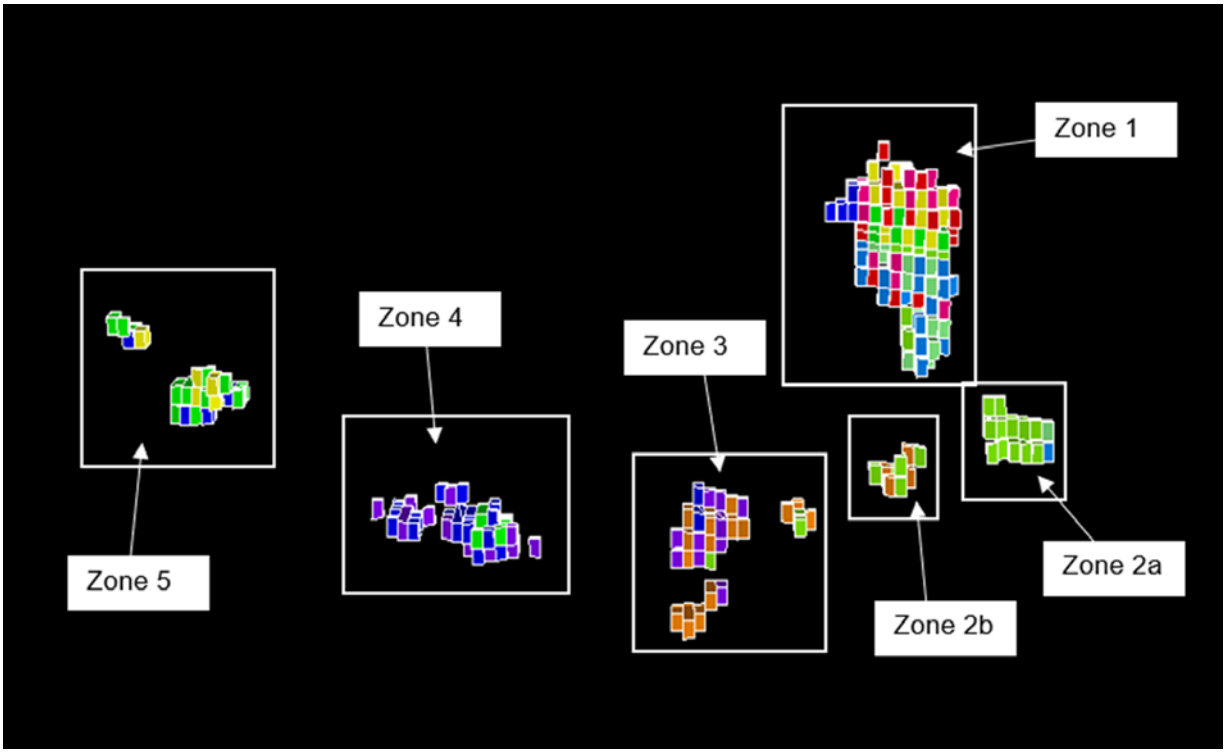


Figure 16-16: Mine Layout and Stoping Zones Considered in Modelling Process

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 x 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 cross cuts, 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 cross cuts. 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 cross cuts. 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

In reviewing the PEA Bara noted that the selected mining method for the PEA is Sublevel blasthole retreat with cemented backfill in a bottom-up configuration (LHOS with backfill). The PEA also notes that there are a number of other possible methods that could be employed specifically mentioning the sublevel caving and cut and fill methods, however no formal selection process has been undertaken at the PEA level of study for the mining method.

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 in addition to the Sublevel blasthole retreat with cemented backfill, assumed in the PEA.

Apart from the LHOS with backfill method selected for the PEA study, 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 tailing 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 are 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 cross cuts. 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 10m thick and is located between the elevations of 277.5 and 267.5 m 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 5277.5m 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 above. The design criteria are summarised below.

Table 16-11: Mine Design Criteria

Main Access and Transport	Trackless decline, trucking, rubber tyred 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	4.5% 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)	10m 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	21Ml 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	0.3m
Muck bay frequency	75m
Decline angle	-8° (1 in 7.9)
Level Development	
Access crosscut	4.6 m (w) x 5.1 m (h)

Strike footwall drive	4.6 m (w) x 5.1m (h)
Stope crosscut	4.5 m (w) x 4.5m (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	Ventilation study (Assume 400 m^3/s).
In-take airways	Decline (24m^2)
	Downcast ventilation shaft (5.0 m diameter)
Exhaust shaft/s	Upcast ventilation shaft (5.0 m diameter)

Mining Modifying factors

The table below shows the mining modifying factors proposed for use in the feasibility study mine 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			
From host rock (waste)	%	2.1	At average grade below cut-off
From backfill	%	2.3	Zero grade
Ore recovery (from mining)	%	95	

Cut-off grade	ppm	2074	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 will facilitate truck hauling of rock, transport of personnel and material in and out of the mine as well as supplement the intake ventilation capacity.

In order to establish a safe mining face in hard rock to start the decline development a boxcut is excavated. The boxcut will be 13m deep and 31m 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 will be 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 will have 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. Figures 16-18 and 16-19 show a plan and a long section view of the boxcut and portal excavations.

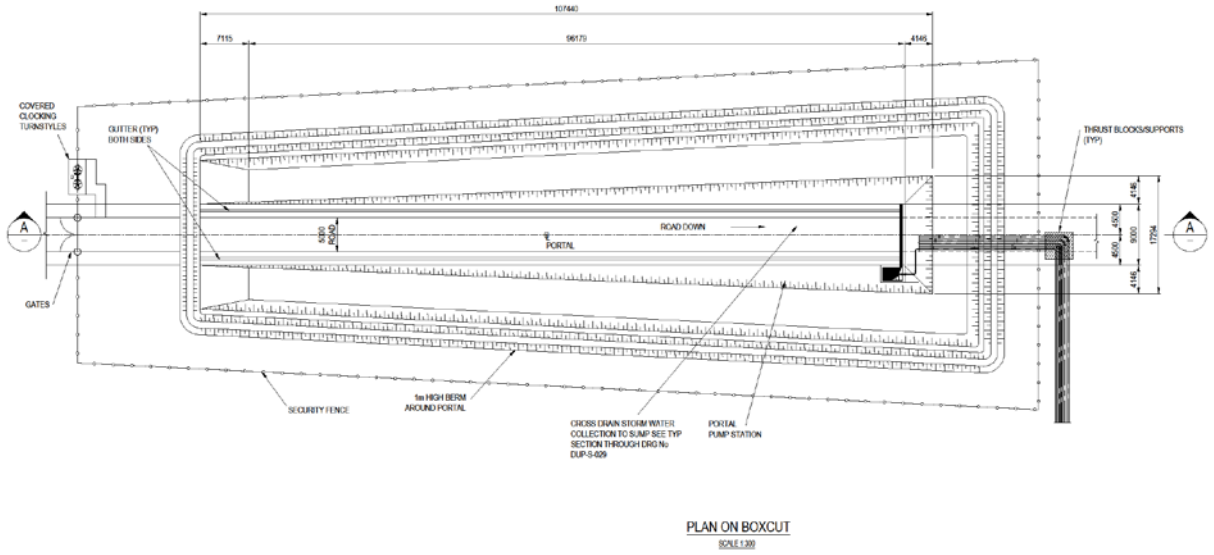


Figure 16-18: Plan View of the Boxcut

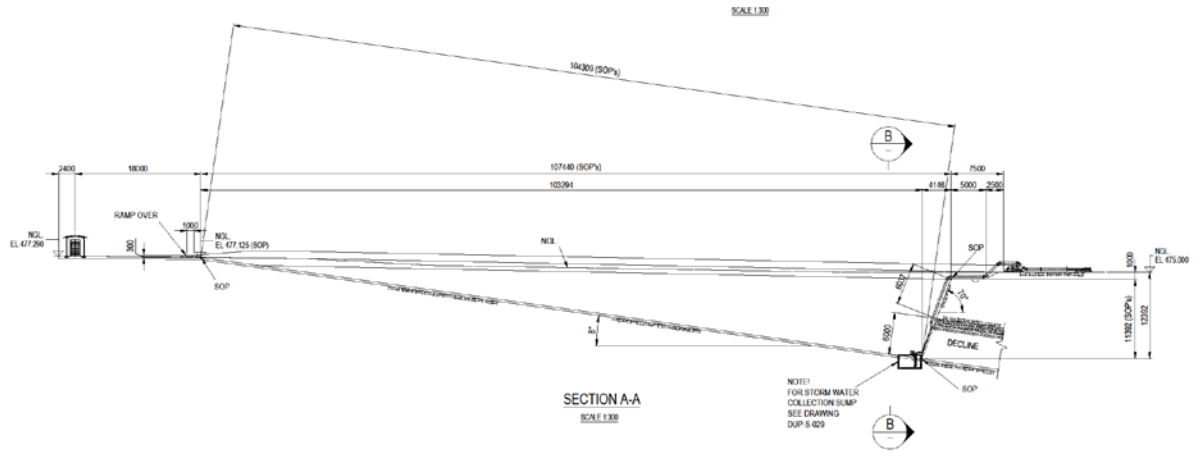


Figure 16-19: Cross Section of the Boxcut and Portal Excavation

The portal area and decline will be sized to accommodate the trucks selected for ore and waste haulage, the selection of which is discussed in 16.15. The finished dimensions of the portal and decline will be 4.6m high x 5.1 m wide as illustrated in Figure 16-20. To allow for a road base 30 cm thick, the excavation will be blasted at 4.6m wide x 5.4 m high. The decline will be developed at the same inclination as the boxcut roadway, an inclination of minus 8 degrees or 1 in 7.1 (14%).

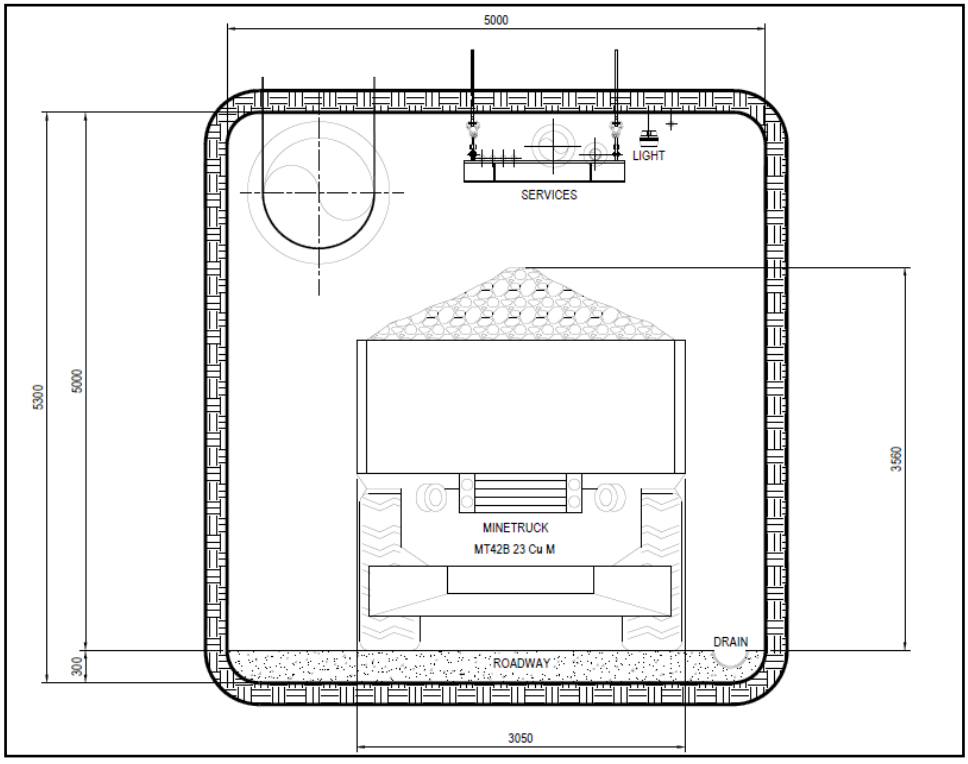


Figure 16-20: Cross section through Decline

The portal entrance hangingwall will be supported by fabricated steel sets, together with the recommended support requirement as specified by the geotechnical engineer. The undercut will be 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 to be installed on a chain suspended steel bracket arrangement, pinned to the highwall. A steel reinforced concrete support arrangement will be positioned at the top of the highwall and designed to carry the loads induced by the services. The piping and electrical reticulation services will also be support from the underside of the steel sets and carried into the decline, on a typical hanger bracket system.

The complete boxcut area is enclosed with medium security fencing with a gate and turnstile installation. The turnstile access point would interface with the mine human resources management system to control and monitor personnel and vehicle movement, in and out of the decline.

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 physical conditions that are encountered when construction takes place, may affect the final length of the arch set, and will depend on the rock engineer's inspection and recommendation.

The total length of the boxcut and decline system from surface to the lowest level of the mine is 4495 m. The figure below shows a plan view and vertical projection of the boxcut and decline system from the mine design software used to layout the mine.

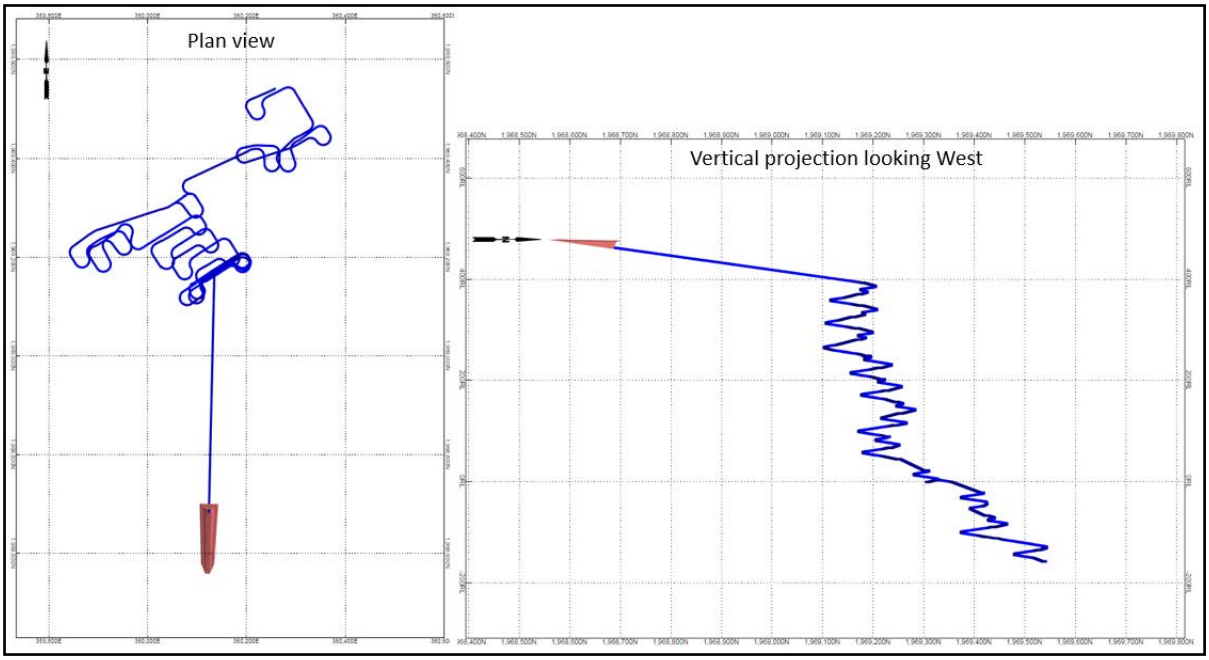


Figure 16-21: Plan view and vertical projection of decline

16.14. Development

The decline will be 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 x 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 raises 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 16.5m 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 hangingwall 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.

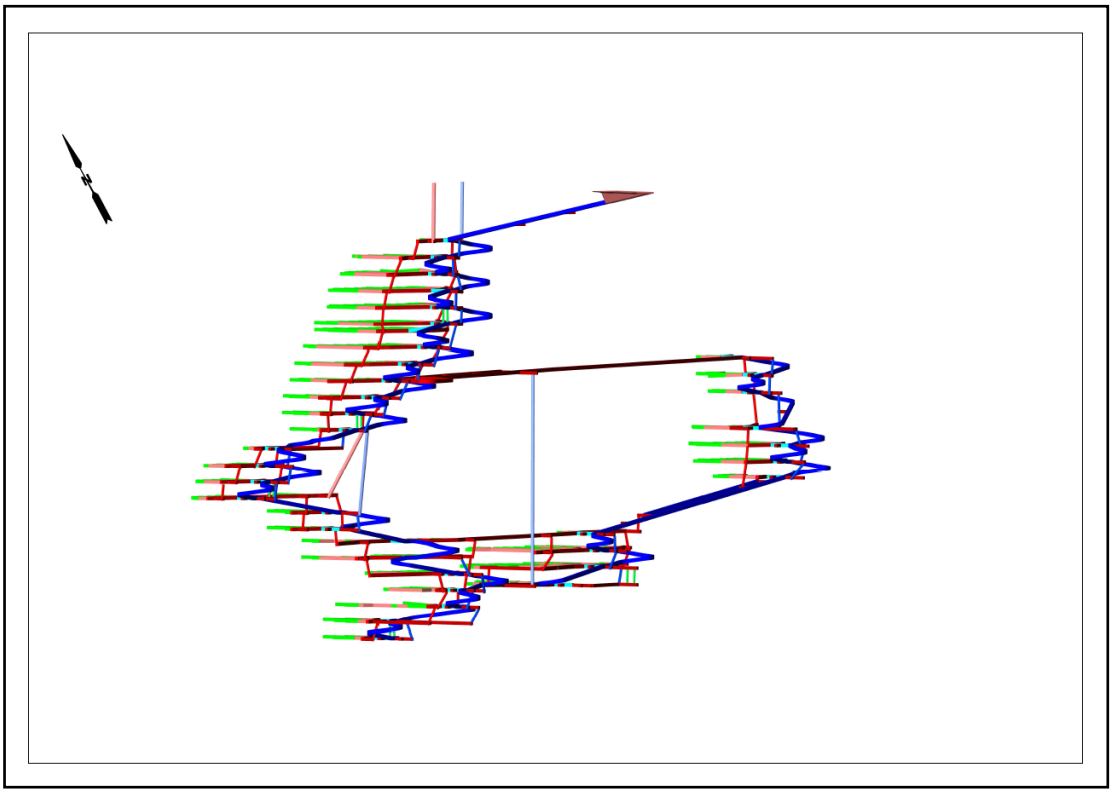


Figure 16-22: Isometric view of development layout looking North-East

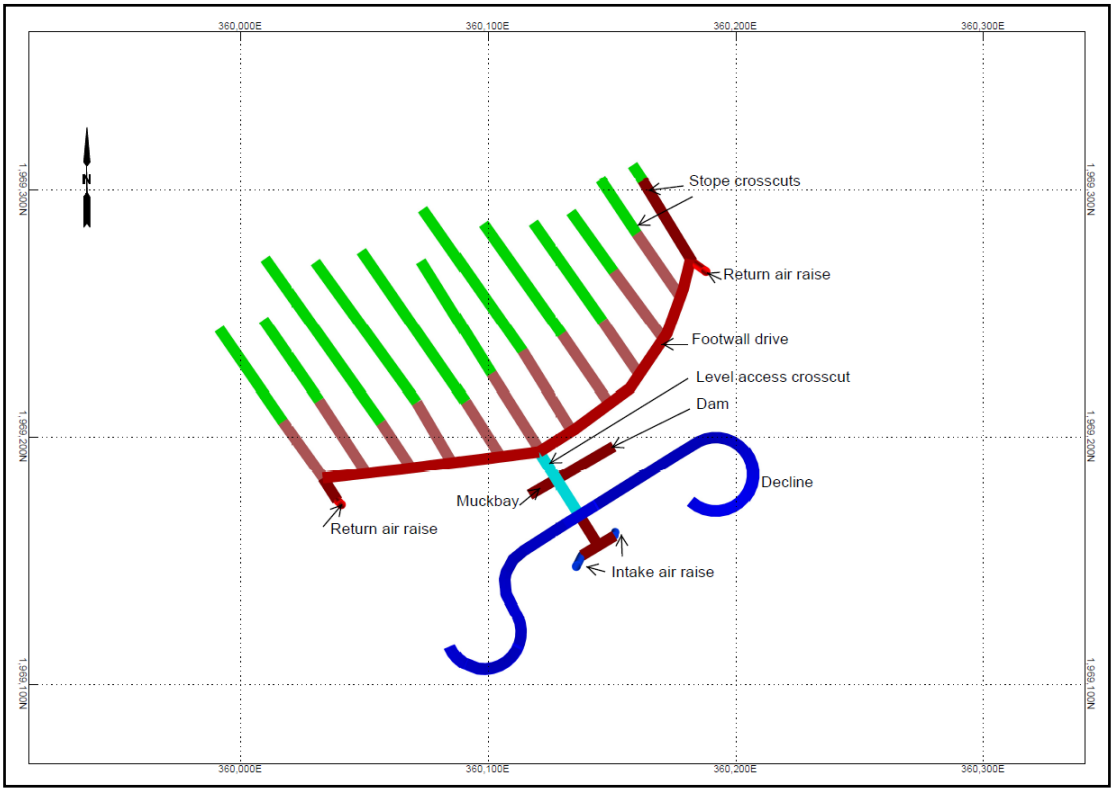


Figure 16-23: Plan view of typical production level

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 wate 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 (Raisebore)	4.4		15.2	2.36	35.9
Return air raise	3.5	3.5	12.3	2.36	28.9
Return air raise (Raisebore)	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 x 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 4.5m x 4.5m.

Figure 16-24 shows a cross section of a stope crosscut (4.5 m x 4.5 m finished) with the largest equipment expected to operate in these ends, a 14-tonne capacity loader (LHD).

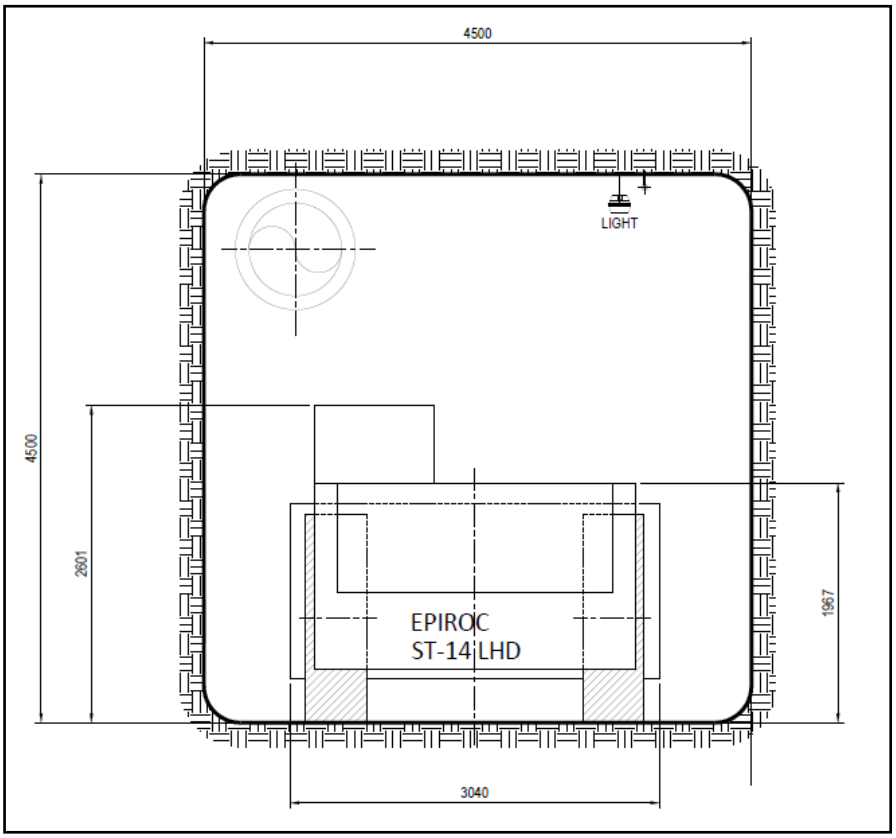


Figure 16-24: Ore drive showing 14-tonne loader

All development drilling will be done by a twin boom electrohydraulic drill rig (Epiroc Boomer or equivalent). 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

In order 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. The table 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 ($/lb)	**35**
lb/kg	2.205
U_3O_8 Price ($ /kg)	**77.16**
Operating cost	
Mining	45
Processing	54
Overheads (G&A)	35
Operating cost ($/ t milled)	**134**
Breakeven recovered grade (kg/t)	1.74
Metallurgical recovery	94%
Breakeven RoM grade (ppm) before royalty	1852
Royalties (%)	12%
Breakeven after royalty	**2074**
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	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 (deg)	60	
Unplanned dilution - HW (m)	0.5	End walls only
Unplanned dilution - FW (m)	0.5	End walls only
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=300m
Stope direction (azi °)	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 ore. 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®.

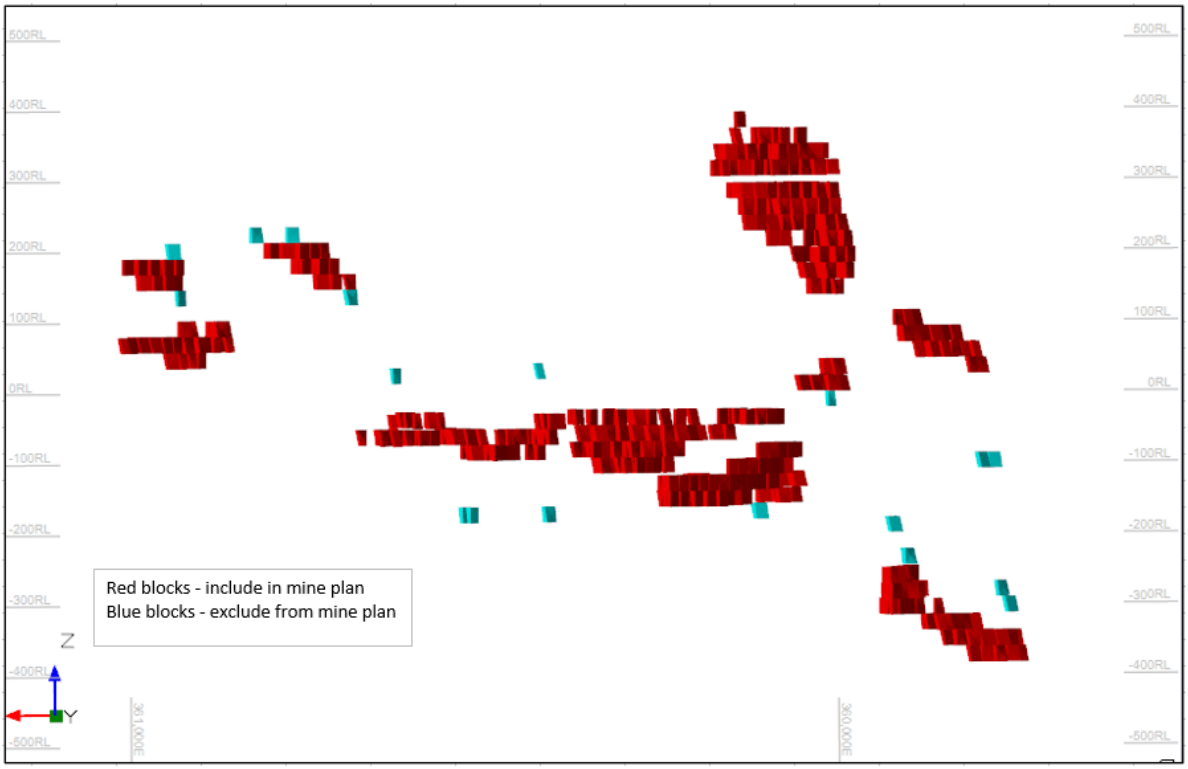


Figure 16-25: Stope shapes produced by DeswikSO®

The stopes coloured red in the figure 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 2074 PPM U_3O_8 ($35/lb)

	Unit	Tonnes	U_3O_8 PPM	U_3O_8 (t)
In situ		4,112,837	5,237	21,539
Measured	Tonne	-	-	-
Indicated	Tonne	3,498,811	5,968	20,880
Inferred	Tonne	170,421	3,869	659
Tonnes unclassified (dilution)	Tonne	443,605		

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 cross cuts 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 x 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 a number of 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 3D 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 below.

Table 16-17: Mine Scheduling Parameters

Item	Unit	Value
Stope productivity		
Long hole drilling	m/month	5700
Stope Loading (LHD)	t/day	900

Item	Unit	Value
Backfilling	m^3/day	400 (minimum)
Backfill curing time	days	14
Development advance		
Ramp	m/month	75
Level Access		60
Footwall drive		60
Stope crosscut – waste		60
Stope crosscut – ore		50
Drop raising		24

The activities were then scheduled, taking into account, 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:

Excavation of the boxcut commences in January 2022.

Blasting of the decline face from the boxcut commences in April 2022.

The Upcast ventilation system is established by the completion of ventilation raise in January 2023.

Level development commences on the 5367 Level in November 2022.

First ore is produced from development in May 2023.

Stoping commences in February 2024 on the 5300 Level.

Steady state production of 30,000 tonnes per month of ore is achieved in March 2024.

The mine has a life of mine of 14 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

Name	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2035	Total
Ramp Development Metres	410	1,040	778	886	480	526	560	852	720	480	481	123	-	-	-	7,337
Cubby Metres	34	341	365	633	320	124	317	806	416	206	160	142	-	-	-	3,864
FW Drive Metres	-	619	899	538	431	522	884	1,390	957	350	136	66	-	-	-	6,792
Access Xcut Metres	-	129	165	151	90	78	90	183	30	90	45	42	-	-	-	1,094
FAR Metres (Drop Raise)	-	218	206	301	142	213	432	268	153	59	94	22	-	-	-	2,106
RAR Metres (Drope Raise)	-	218	206	301	142	213	432	268	153	59	94	22	-	-	-	2,106
FAR Metres (Raisebore)	-	80	-	-	8	112	-	-	213	76	-	-	-	-	-	488
RAR Metres (Raisebore)	-	81	-	-	45	58	-	-	-	-	-	-	-	-	-	184
Stope Xcut (W) Metres	-	1,068	1,565	656	377	378	762	1,334	658	498	162	112	-	-	-	7,569
Stope Xcut (O) Metres	-	619	2,788	972	429	455	689	693	1,358	852	134	146	-	-	-	9,136
Stope tonnes LHOS	-	-	126,712	321,334	350,148	352,178	340,470	337,533	301,794	324,283	358,332	357,509	348,216	286,512	11,997	3,817,019
Stope grade	-	-	5,970	8,277	6,557	8,651	5,128	5,559	7,571	3,757	3,216	3,097	3,282	3,506	3,673	5,328
Stope U308 (t)	-	-	756	2,660	2,296	3,047	1,746	1,876	2,285	1,218	1,152	1,107	1,143	1,004	44	20,335
Ore Development	-	29,564	133,252	46,467	20,483	21,767	32,945	33,123	64,881	40,740	6,399	6,988	-	-	-	436,608
Development ore grade	-	2,451	7,663	4,268	1,736	1,254	2,455	1,295	2,191	1,751	1,908	1,412	-	-	-	3,926
Development ore U308 (t)	-	72	1,021	198	36	27	81	43	142	71	12	10	-	-	-	1,714
Total Ore Tonnes	-	29,564	259,964	367,801	370,631	373,945	373,415	370,656	366,675	365,023	364,731	364,497	348,216	286,512	11,997	4,253,626
RoM U308 ppm	-	2,451	6,838	7,771	6,291	8,221	4,892	5,178	6,619	3,533	3,193	3,065	3,282	3,506	3,673	5,184
RoM U308 (t)	-	72	1,778	2,858	2,332	3,074	1,827	1,919	2,427	1,290	1,165	1,117	1,143	1,004	44	22,050
Waste Tonnes	25,701	185,317	204,106	161,988	97,810	103,019	149,822	250,301	159,607	90,969	57,803	26,378	-	-	-	1,512,821
Hauled Tonnes	25,701	214,880	464,070	529,788	468,441	476,964	523,237	620,957	526,283	455,992	422,535	390,874	348,216	286,512	11,997	5,766,447
Tonne.km	13,209	265,164	711,415	790,352	727,541	760,325	874,970	1,164,827	1,164,123	1,500,104	1,724,891	1,671,128	1,558,073	1,240,520	51,479	14,218,120

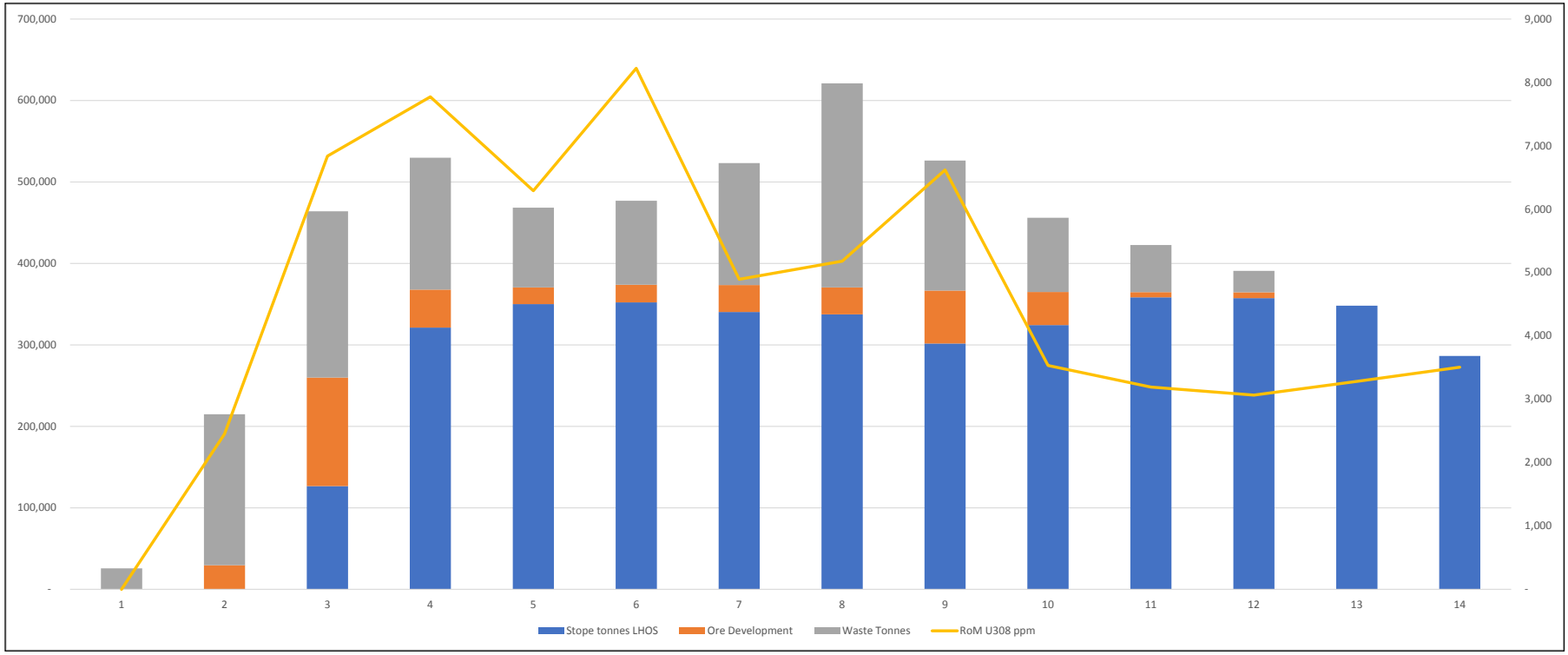


Figure 16-26: Dasa Phase 1underground production profile

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 the Tables 16.19 and 16.20 below.

Table 16-19: Mining Inventory by Mine Zone

Zone	In-situ Tonnes	U_3O_8	RoM Tonnes	RoM U_3O_8	RoM U_3O_8
Unit	Tonnes	PPM	Tonnes	PPM	Tonnes
1	2,464,615	6,980	2,316,047	6,887	15,950
2	264,339	3,621	256,078	3,574	915
3	656,114	3,093	633,541	3,056	1,936
4	604,673	3,003	584,616	2,966	1,734
5	478,916	3,312	463,345	3,269	1,515
Total	**4,468,657**	**5,279**	**4,253,626**	**5,184**	**22,050**

Table 16-20: Mining Inventory by Resource Class

	RoM tonnes	U_3O_8 ppm	U_3O_8 (t)	U_3O_8 (Million lbs)
Measured	-	-	-	
Indicated	4,066,390	5,267	21,417	47.217
Inferred	187,236	3,375	632	1.393
Total Mining inventory	4,253,626	5,184	22,050	48.611

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 4.4% of the RoM tonnes and 2.8% of the contained U_3O_8.

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 have placed orders for a number of 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 14t 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 (10t)
Option 4	MT2200 (22t)	ST7 (7t)

In order 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 0.75m 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 Figures 16-27 and 16-28.

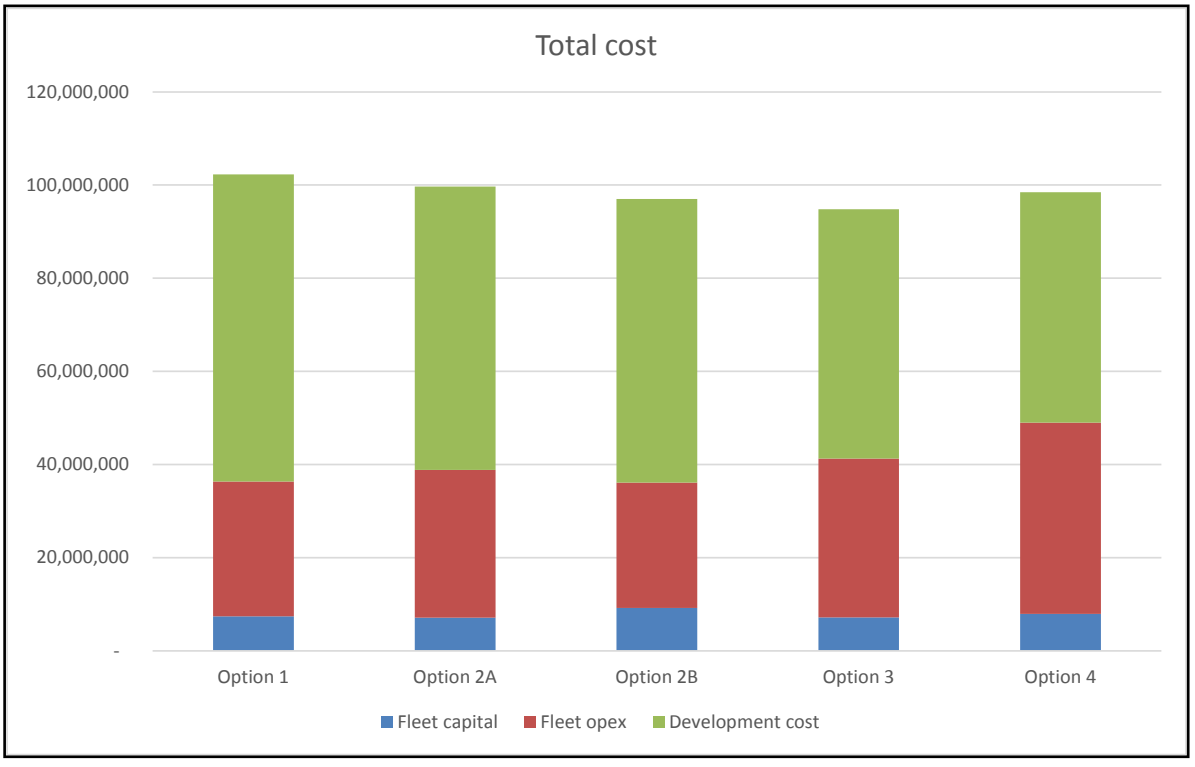


Figure 16-27: Comparison of total costs

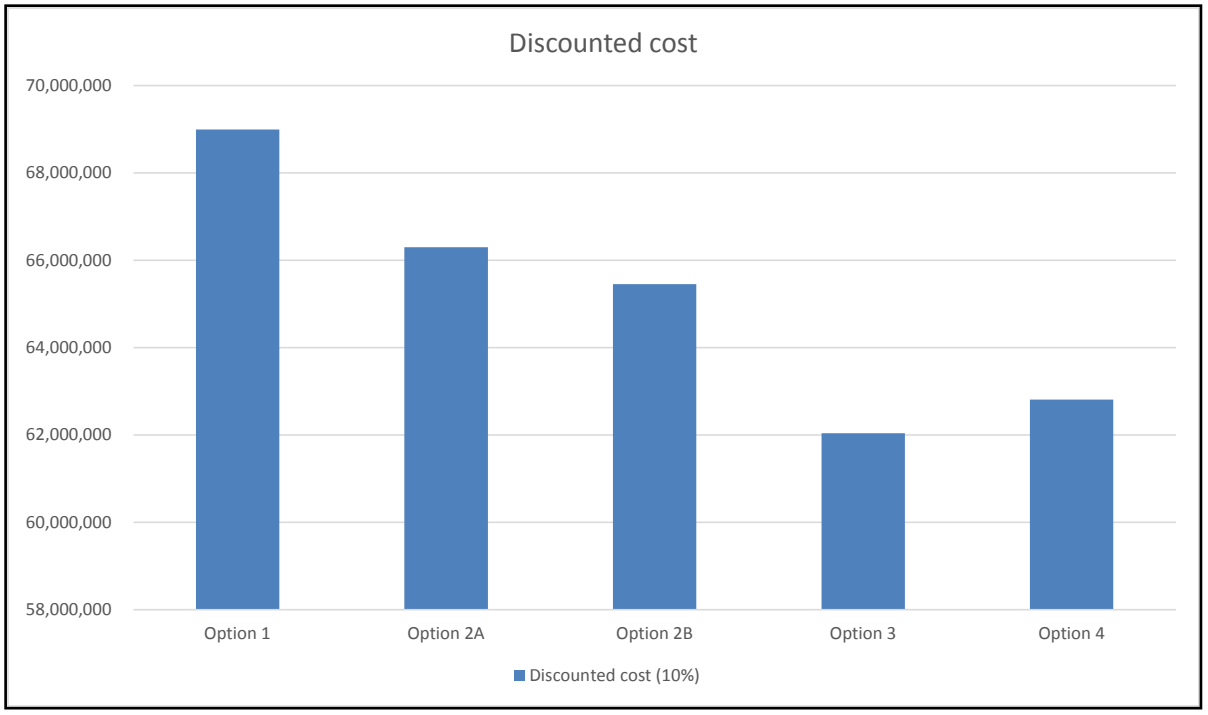


Figure 16-28: Comparison of net discounted cost

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 40t 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	1
LHDs (14t)	4

Description	Qty
ADTs (42 t)	5
Grader	1
Utility vehicles	3
Transmixer	1
Shotcreter	1
Telehandler	2
Pick-ups	2
Troop carrier	2

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 minimised at all times.

Ventilation Design Criteria

The ventilation design criteria used conforms to established international best practices to provide a safe and healthy working environment. Tables 16-23 to 16-27 below list the principal criteria used in the design of the ventilation systems for Dasa.

Table 16-23: Environmental Criteria

Design intake air temperature [wet bulb/dry bulb]	22.0/32.0°C
Design relative humidity	40%
Design reject air temperature [wet bulb/dry bulb]	32.0/37.0°C

“Withdraw from working place” wet bulb temperature	32.5°C
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]	5.0m 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/100m
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.

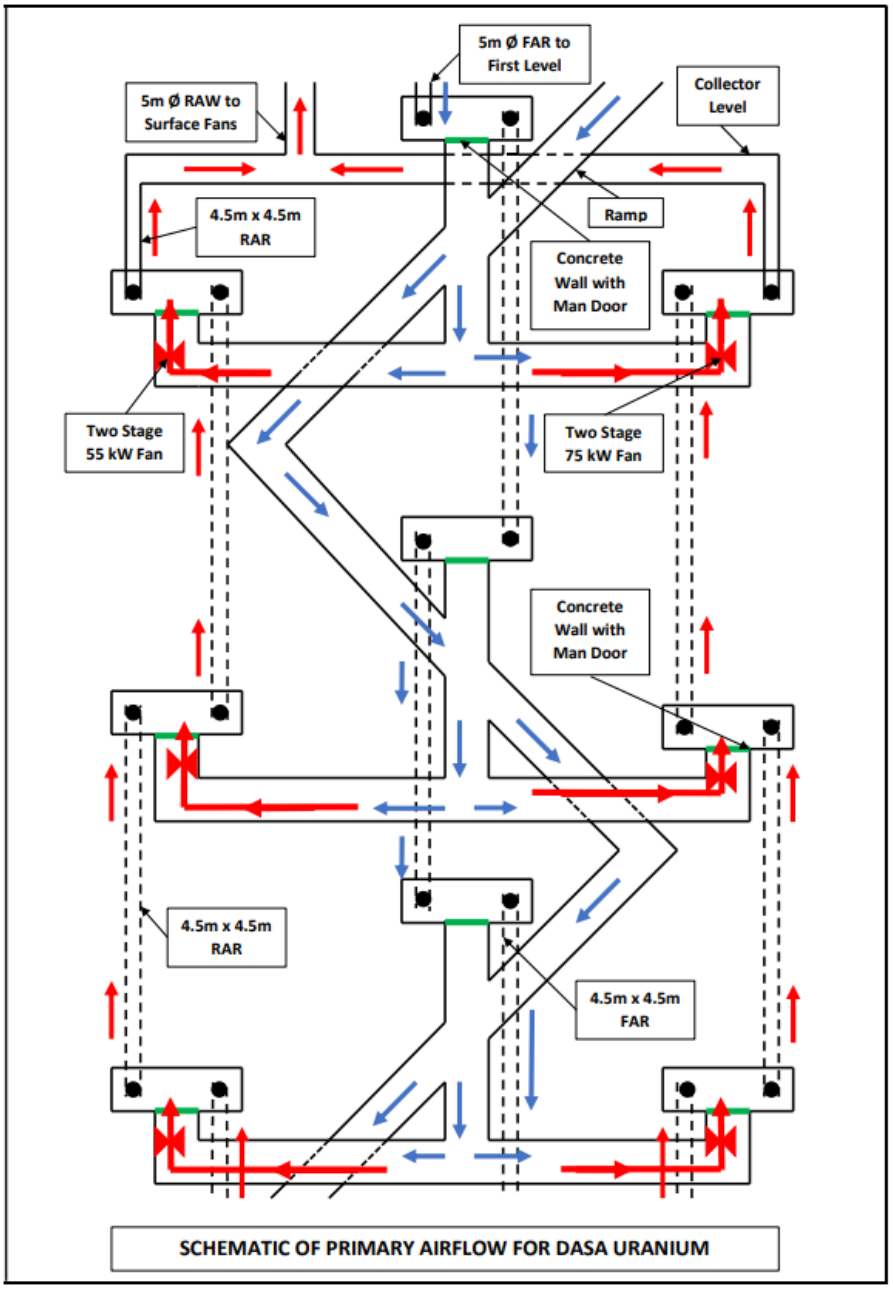


Figure 16-29: Schematic of Primary Airflow at dasa

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 1200mm 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 1200mm 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.

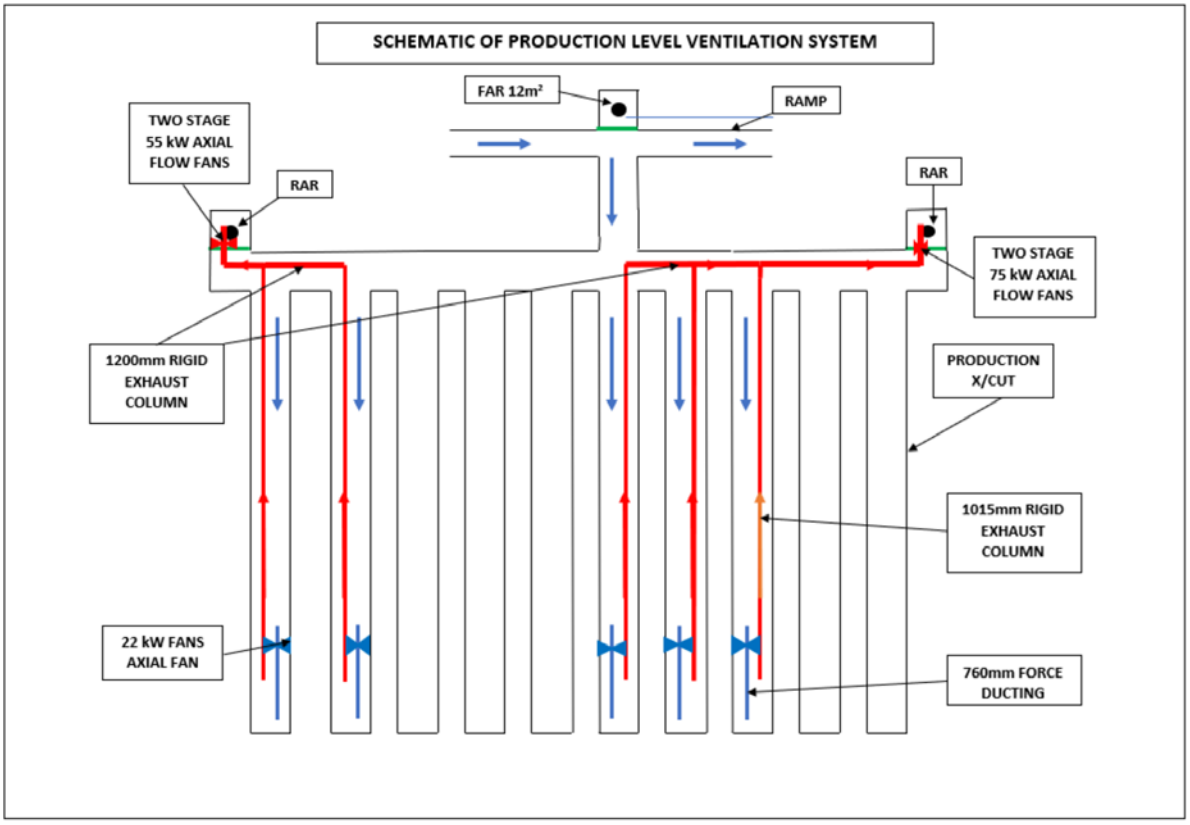


Figure 16-30: Schematic of Production Level Ventilation System

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 1200mm 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.

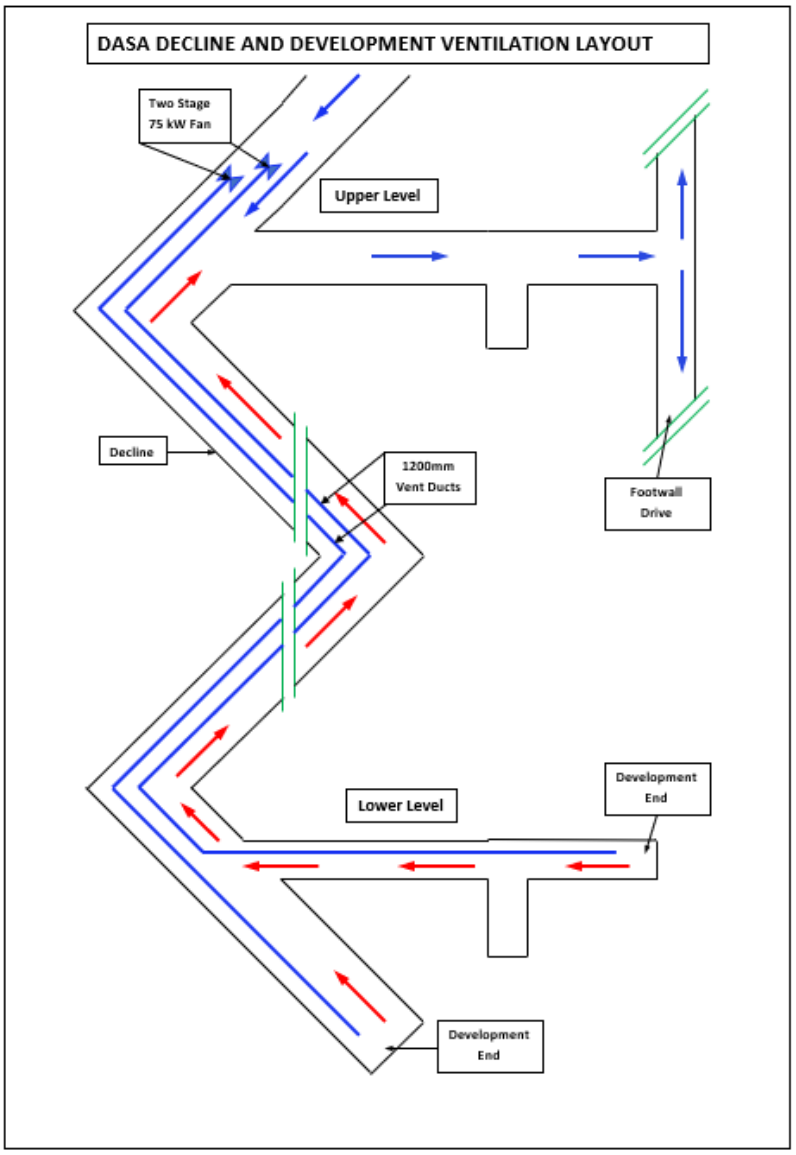


Figure 16-31: Decline and Development Ventilation Layout

Main 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-tyred 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 tyred 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 work place where the temperature exceeds wet bulb of 27.5°C or dry bulb of 32.5°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.

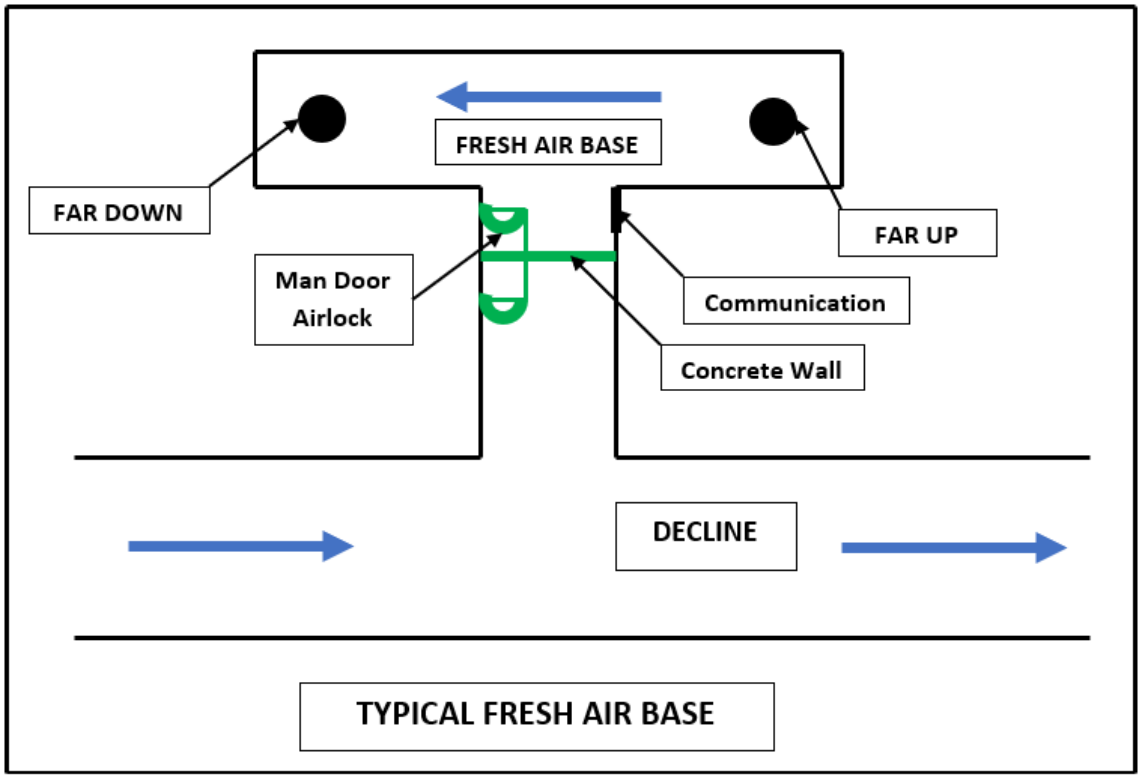


Figure 16-32: Typical Fresh Air Base

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, the 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, therefore it is important to have a Radiation Management Plan to ensure that dose rates are kept as low as possible. It is recommended that the mine should consult a Radiation Protection Expert to provide competent advice on the compilation of the Radiation Management Plan, purchase 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 2022, by that time a Radiation Plan must be in place and Personal Radiation Dosimetry and site Radiation Measurements with specialist instruments be available. The mine can decide to contract out the site measurements on advice from the Radiation Specialist but will have to contract out the Personal Dosimetry as to set up an onsite Laboratory is excessively expensive and complicated.

Outlines of a 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 an 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)

Auditing and review of the effectiveness of the RMP.

16.19. Human Resources

Mine Development Phase – Mining Contractor

The initial two years of underground development will be undertaken by a mining contractor. The contractor will be responsible for all the underground operations during the period of the contract. The selected contractor has provided a manpower complement which totals 178 people which is illustrated in their organogram, presented in Figure 16-33 below.

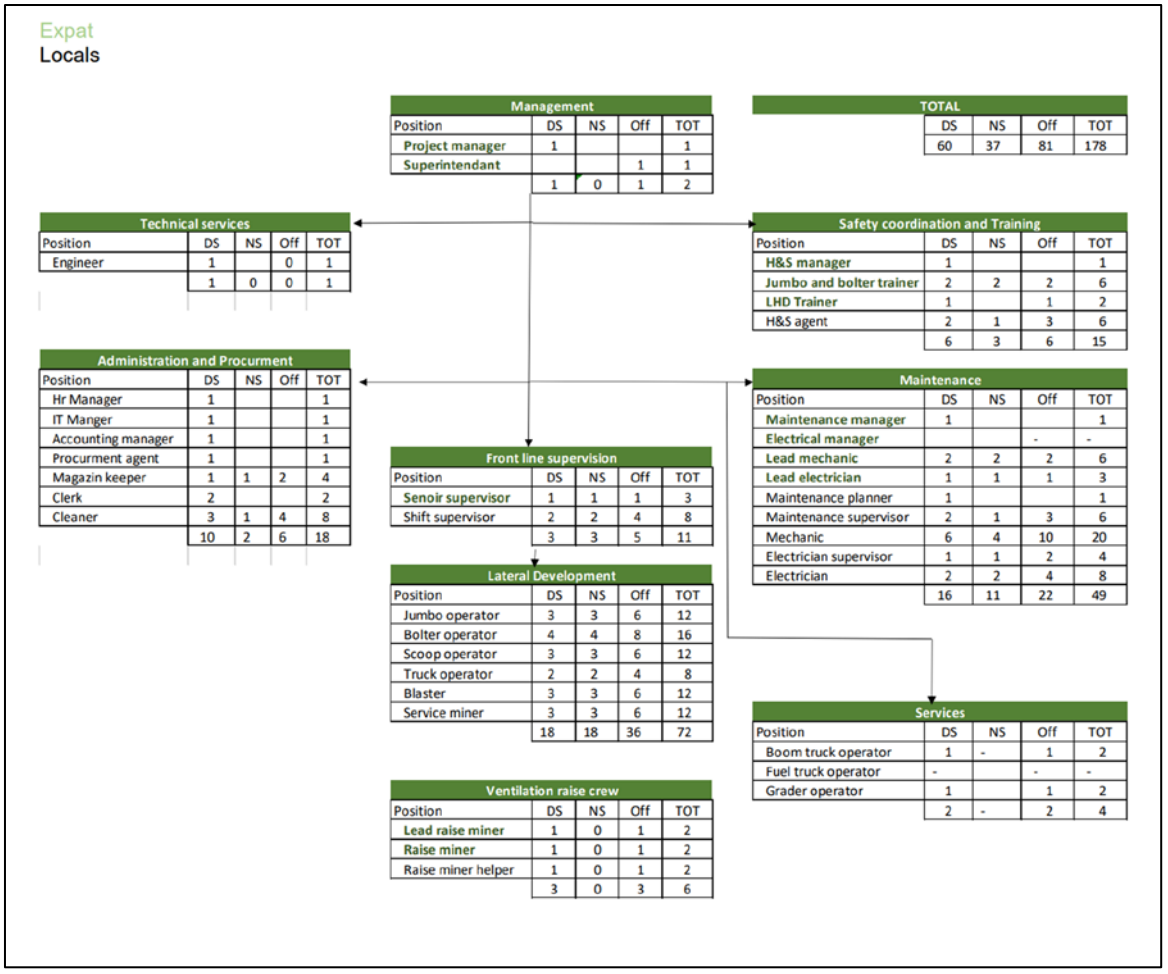


Figure 16-33: Mining Contractor Manpower Plan

After the initial 2 years of mine development, Global Atomic Corp plan to assume operational control of the mine. It is likely that when this occurs the workforce employed by the contractor will largely become mine employees. At this point it is planned to structure the mine labour as discussed below.

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 90km 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). Figure 16-34 below indicates the proposed shift cycle; these shifts will be 10 hours in duration.

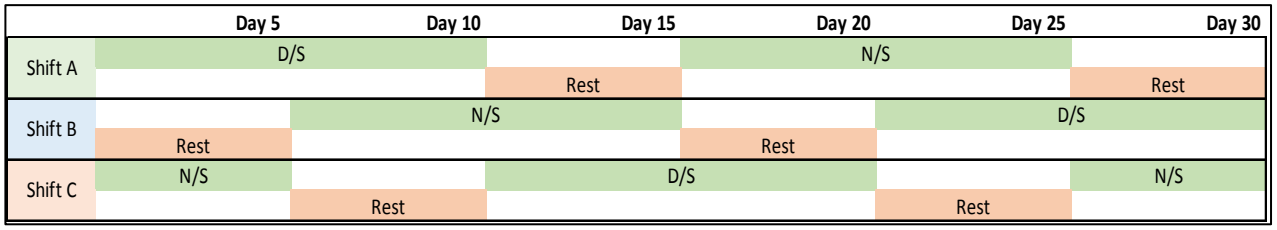


Figure 16-34: 10/5 Shift Cycle

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 16-35 below indicates the proposed shift cycle; these shifts will be 8 hours in duration.

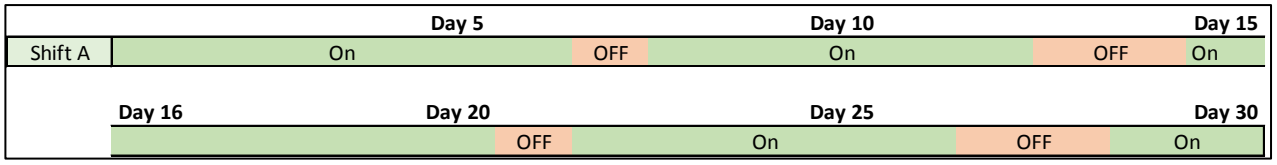


Figure 16-35: 11/14 Shift Cycle

Organogram and Manpower Complement

A high-level organogram which is shown in Figure 16-36, 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 16-37. The posts highlighted in green indicate the mining and engineering positions, which will be on the 10/5 shift rotation.

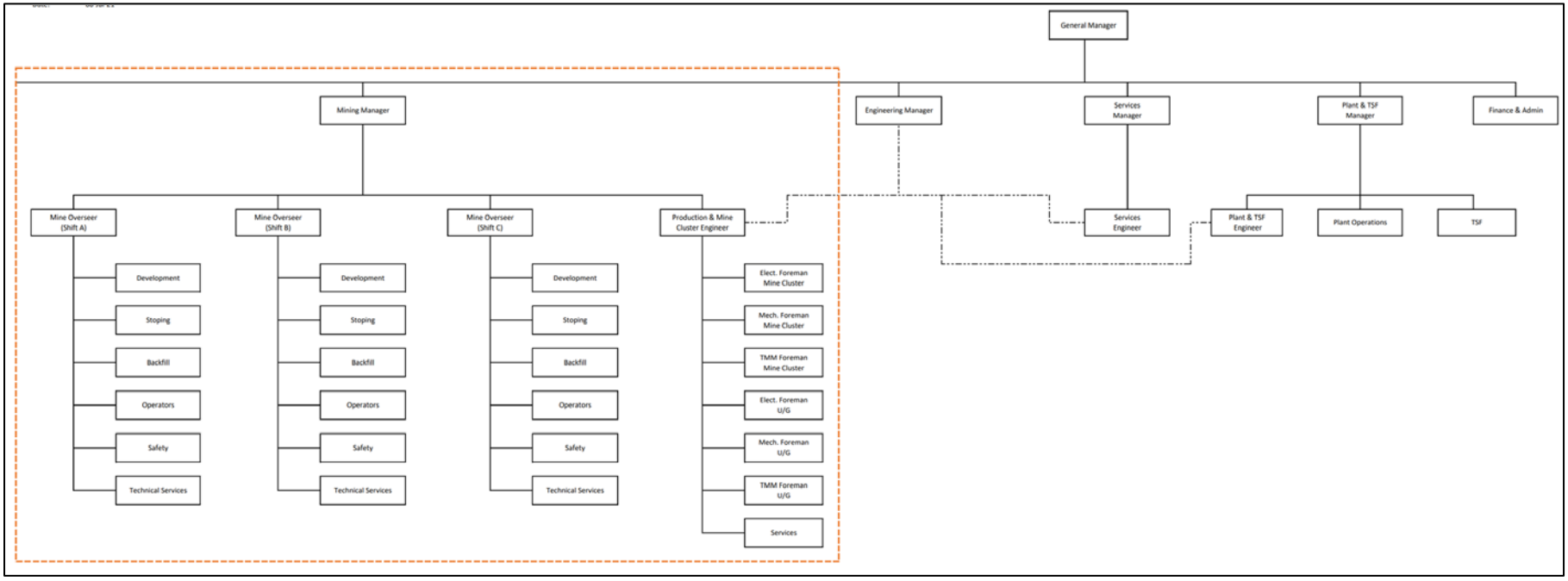


Figure 16-36: High Level Mine Wide Organogram

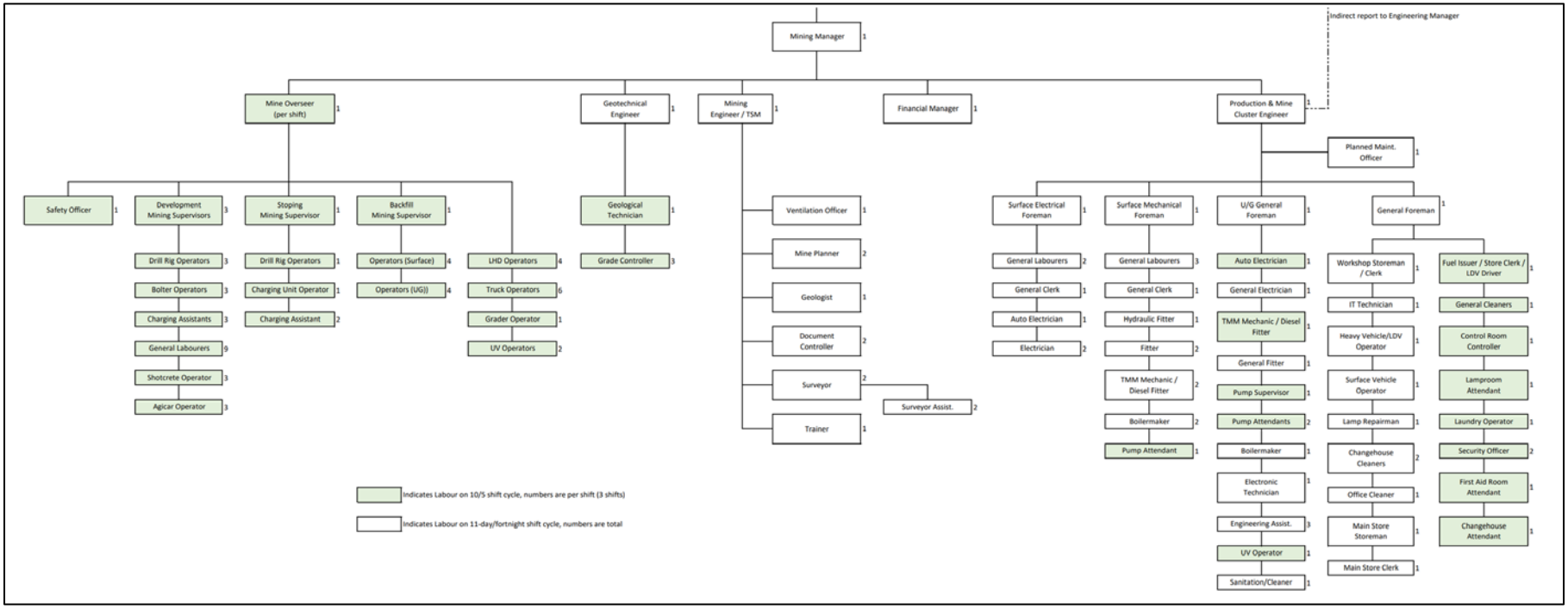


Figure 16-37: Mining Labour Focused Organogram

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

Manpower Area	Complement
Backfill operators (UG)	12
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

Manpower Area	Complement
TMM Mechanic / Diesel Fitter	2
Fitter	2
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

Manpower Area	Complement
Office Cleaner	1
Main Store Storeman	1
Main Store Clerk	1
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

	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 16-38 shows the labour build-up per month over the life of mine.

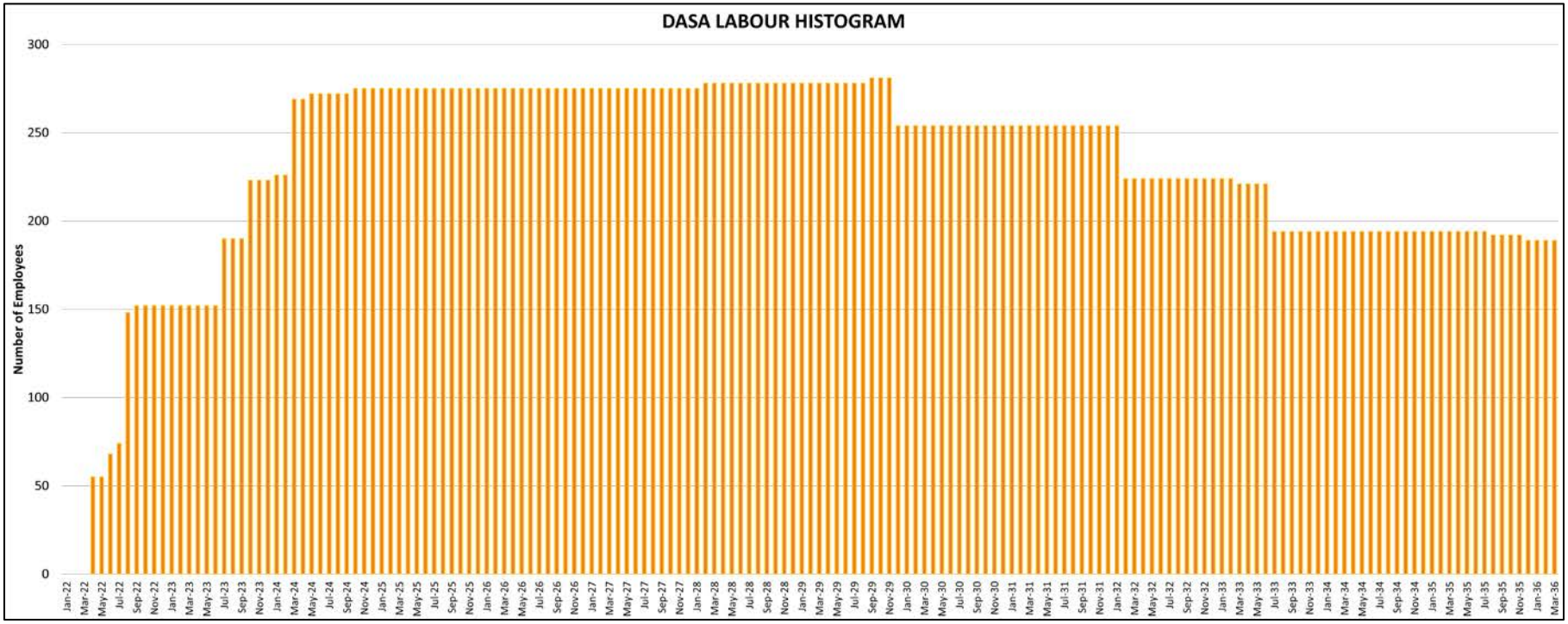


Figure 16-38: Dasa labour Histogram

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

In order to optimise the plant design, trade off studies were completed covering dry milling, final product precipitation reagents and leach tailings disposal options.

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*.

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
c. 1 = worst score in the criteria

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 x S	(0 - 5)	W x S	(0 - 5)	W x 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**

Table 17-1: Weight scoring and ranking methodology

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:

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.

Table 17-2: Precipitation products quality from test work with various reagents

	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 the Table 17-3. 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 reagents.

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 the following assumptions:

1. A dried yellow cake product.
2. Final product to be transported from Niger to converter plants in France, Canada or the United States.
3. Packaging to be done in 210L steel drums.
4. 20ft containers to be used for shipping.
5. Each container to accommodate 40 drums.

Table 17-4: Final product packaging and transport costs

	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 ~ 1000m
- 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.

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 (1.2m spacing)	52 595
5	20° Carry Idlers	126
6	Training Idlers	5 075
7	Return Idlers (3m 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 ~ 1 000m
- Tailing tonnage ~ 500 tonnes/day
- Required tailings solids content~ 50 % m/m solids
- Distance from plant to water makeup borehole ~ 2 000m
- 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.

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 3X1.5X11 Pumps - 5.5 kW Each	15 743
5	KSB UPAC 150-030/19EE - 5.5 kW Each	23 732

Item	Description	Cost ($)
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&Gs 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 ~ 1 000m
- 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 ($)
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	Coster annum ($)
1	Fuel and Services	58 619
2	Spares parts	11 244
	Total Operating Cost	**69 863**


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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.

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 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	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	mm	250
SAG mill feed size, F_{80}	mm	80
SAG mill Product, P_{95}	mm	0.6
Pug Leaching		
Pug leach	% m/m solids	85
Pug Drum Residence Time	mins	10

Description	Units	Value
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 12 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 500m3 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.

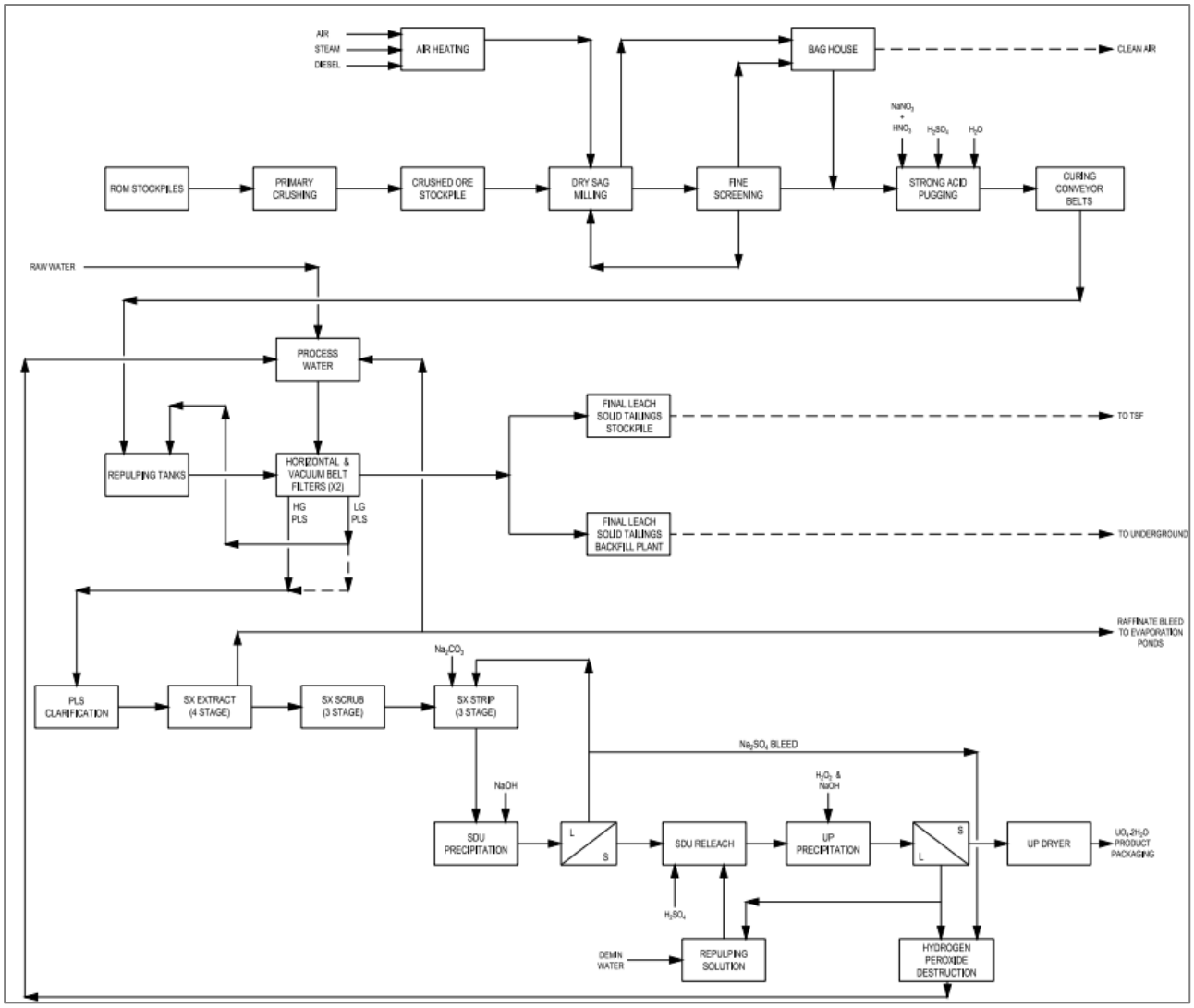


Figure 17-1: Process block flow diagram

Process Flow Diagrams

The block flow diagram was developed into a set of process flow diagrams (PFDs) which was used to develop the mass balance and mechanical equipment list (MEL). The PFDs were used to develop a preliminary equipment list and P&IDs.

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 Table 17-15.

Table 17-15: Summary of process plant major equipment

Item	Equipment Name	Sizing/Description	Number Required
1	Feed Bin	Volume 40m^3	1
2	Jaw Crusher	69 t/h design throughput, ROM size F100 ~ 500mm, P100 ~ 250mm, P80 ~ 80mm	1
3	Crushed Ore Stockpile	8h live capacity	1
4	SAG Mill	50 t/h design throughput, 4.88mD x 1.83mL, F_{80}~ 80mm	1
5	Screens	Dry high frequency sizing screens, Final Product P_{95} ~ 600µm	2
6	Pug Drum	60 t/h throughput, F_{95} ~ 600µm, 10 mins residence time	1
7	Curing Conveyors	2 x 135m long conveyors, 2.4m wide, 0.025 m/s velocity	2
8	Nitric Acid Regeneration Plant	1.25 t/h, 40 % m/m concentration	1
9	Repulp Tanks	20m^3 live capacity tanks, 50% m/m solids, 21 minutes residence time, mechanically agitated, + 1 x 30m^3 live capacity filter feed tank	4 + 1
10	Belt Filter	75m^2 belt filters, 3 wash stages, 50% m/m solids feed slurry	2
11	Pin Bed Clarifier	3.5m diameter	1
12	SX Mixer-Settler	15.7m^3 effective volume mixer tank, 79.1m^2 effective settler area, 10 x mixer settler tanks	1
13	SDU Precipitation	10m^3 live capacity tanks, mechanically agitated	5
14	UP Precipitation	6m^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&IDs). P&IDs 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 & 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

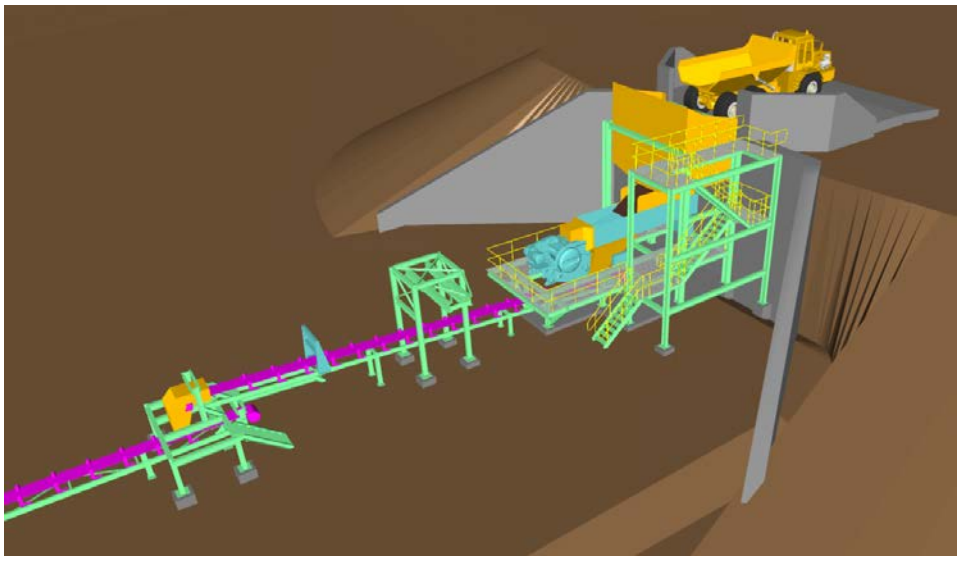


Figure 17-2: Crushing circuit

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

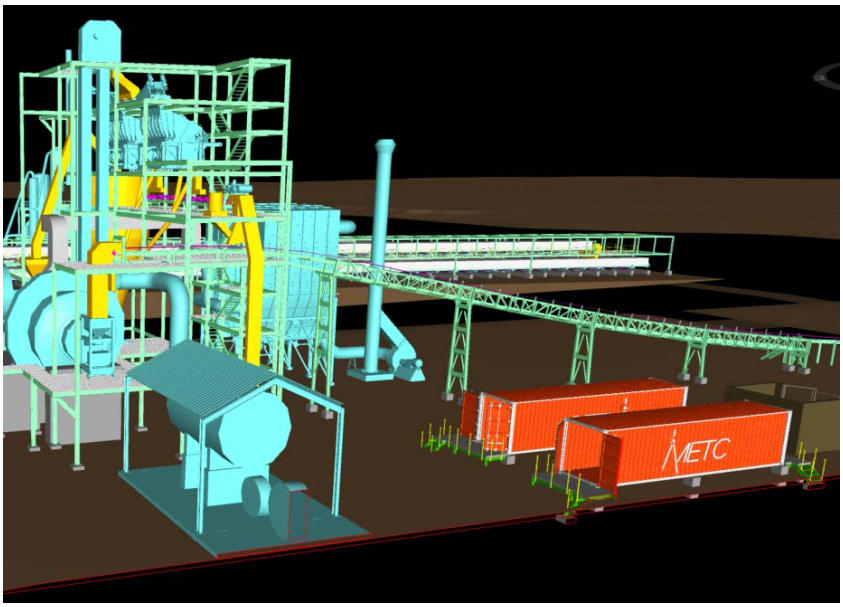


Figure 17-3: Grinding and Classification Circuit

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

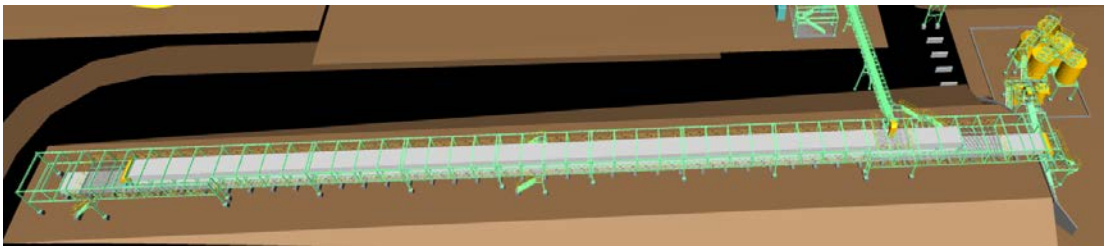


Figure 17-4: Curing and repulping Circuit

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 2.4m wide conveyor belts with a total length of 270m and a belt speed of 0.025m/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

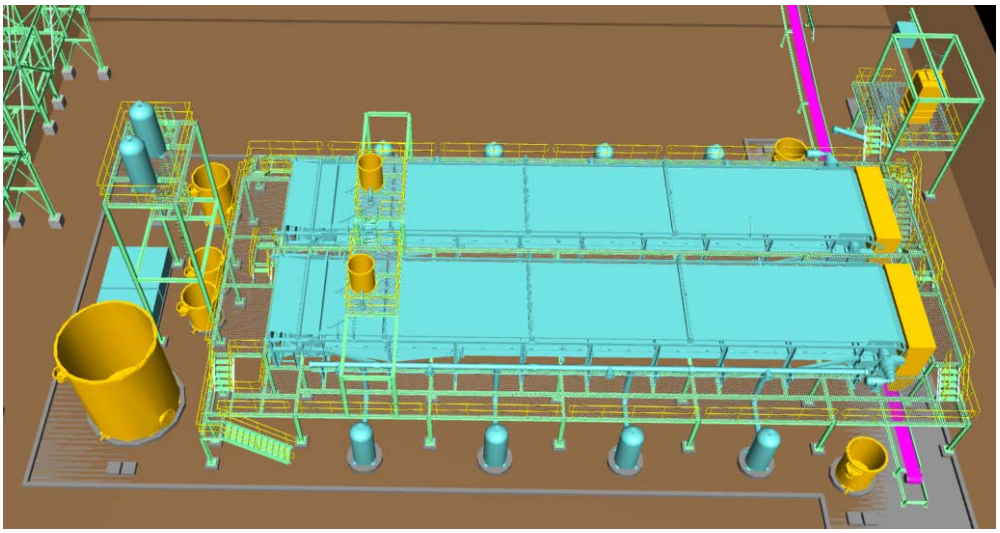


Figure 17-5: Solid liquid separation Circuit

Solid-liquid separation is carried out using two 75m^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 (500m^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 24h 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

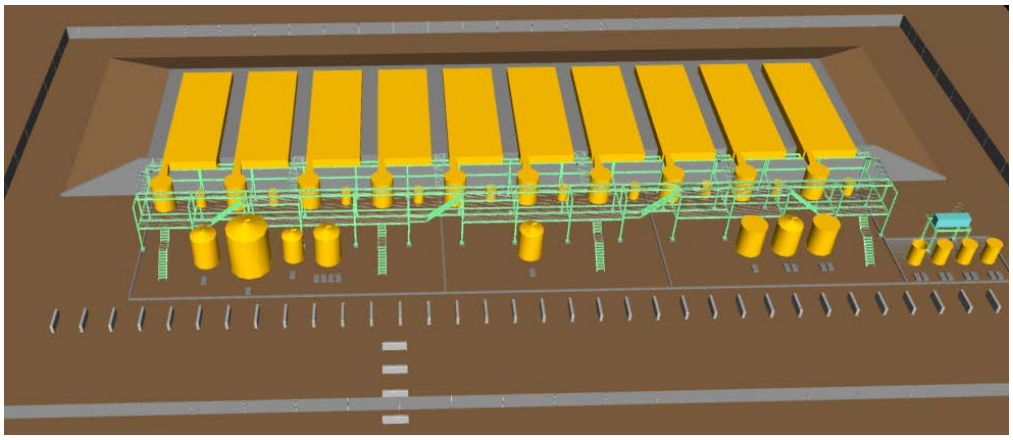


Figure 17-6: Solvent Extraction Circuit

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

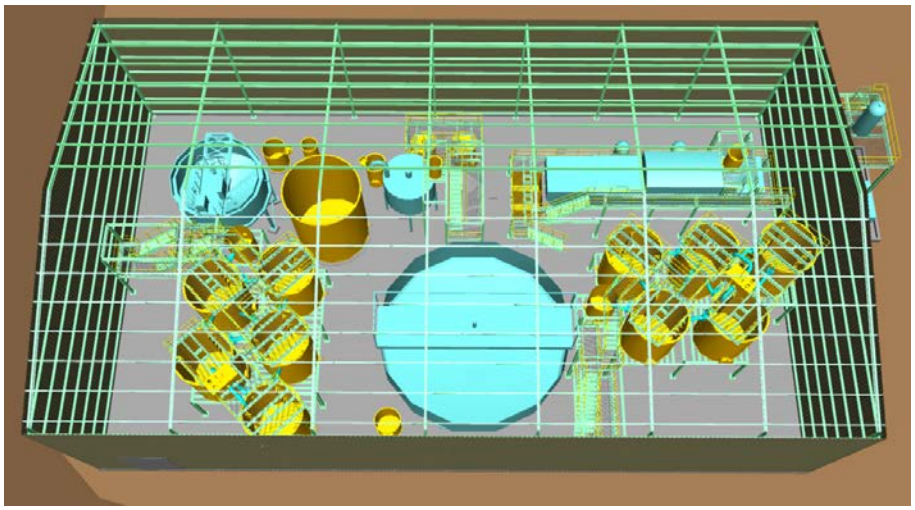


Figure 17-7: SDU Precipitation, UP and Dewatering Circuits

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

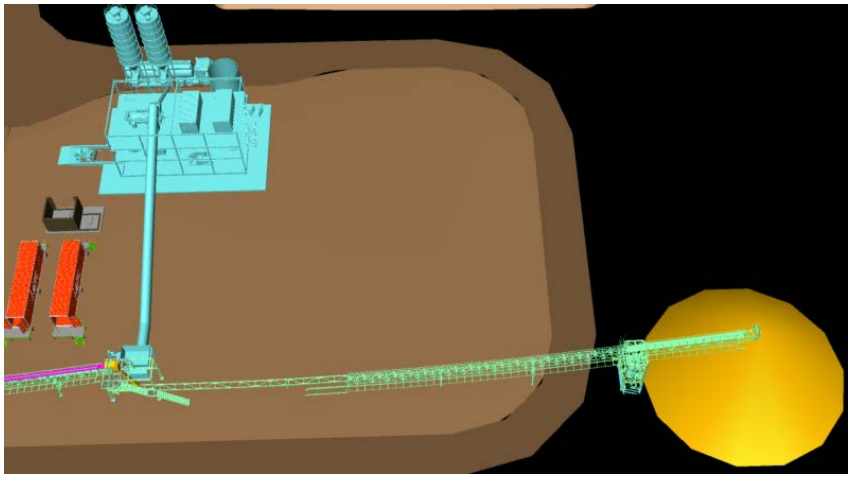


Figure 17-8: Tailings Disposal and Backfill paste plant

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 500m^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 1970s. The Dasa Project test work has proven this as a viable process but has identified unfavourable settling and filtration characteristics. This portion of the plant is sized conservatively to allow for further test work during project execution that will allow for optimized equipment sizing. The main problems seem to relate to the production of excessive ultrafine particles during the precipitation process. Further investigation has

identified opportunities to increase precipitant particle sizes by using re-seeding and elevated temperatures to form larger crystals while the use of flocculants will increase settling rates. Areva (now Orano) in conjunction with Adelaide Control Engineering (ACE) have identified a process to improve solids collection which could be implemented should the future test work not demonstrate improvements. This provides confidence to the owners that the plant to be built will deliver filterable final product.

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 95km 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.

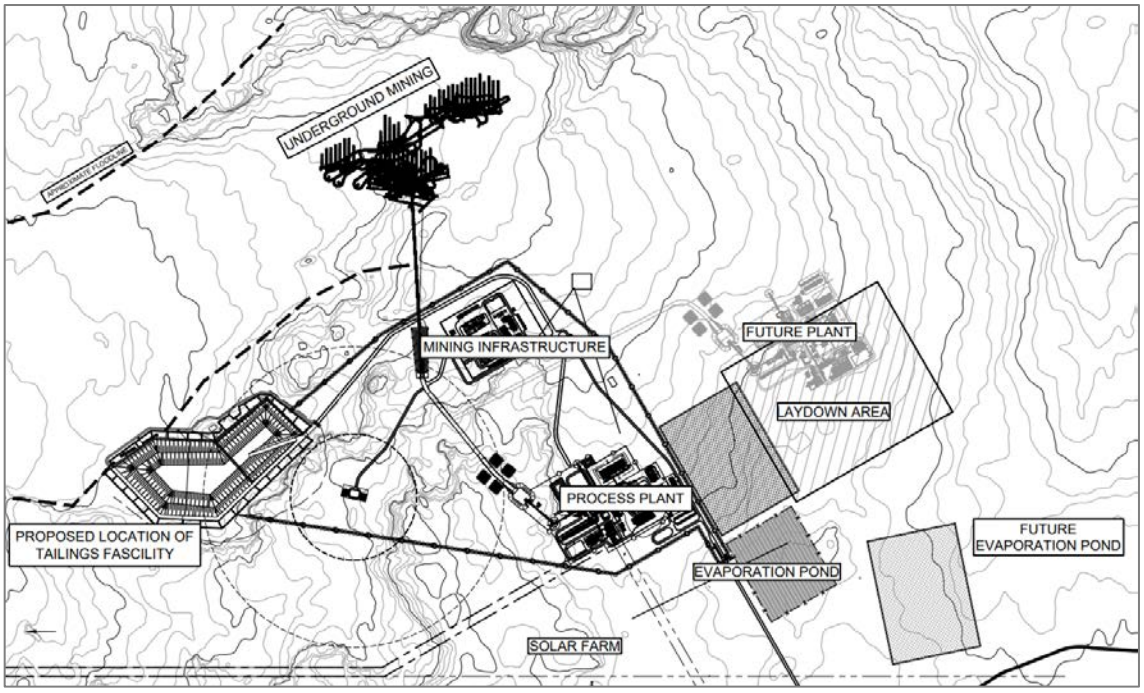


Figure 18-1: Proposed infrastructure and spatial placement of key infrastructure at Dasa Uranium Project

18.1. Process Plant and Services

Power Supply

The production and distribution of electricity is provided by the Nigerien Electric Company (NIGELEC). This energy comes mainly from operating units located in large and medium-sized cities. However, Niger operates the coal bed of Anu-Araren through the Nigerien Coal Company (Sonichar) 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 x 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 at the Sonichar plant that could be used 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

Power availability and reliability from the Nigerien Coal Company (Sonichar) has not been confirmed. Diesel power generation was considered during the study but based on costing it was decided to use the utility power. The following additional power has been allowed for:

4.6 MWp of Solar PV plant has been allowed to supplement the utility power supply during daylight hours.

Two 1MW diesel generators located at the process plant to supply selected process equipment that needs to be energised during a power outage.

1.6 MW (2 x 800 kW duty, 1 x 800 kVA standby) diesel generators located at the accommodation camp to supply the accommodation camp load during power outages.

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 a prefabricated e-house.

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 (MCCs) will be housed in dedicated low voltage steel substations that are converted 12m general purpose “high-cube” shipping containers. The containers will be insulated, air conditioned and fitted with a tropical roof for added cooling. The substations are elevated for cable access.

Transformers

Electrical loads are allocated to MCCs, 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 100kVA transformers.

A transformer loading schedule has been completed for each transformer.

Motor Control Centres (MCC)

The MCCs will be of steel 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, VSD's will be mounted within the MCC, while large VSDs will be mounted external to the MCC as standalone cubicles.

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 109.8mm. 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 27m to 47m below ground level. The project capital cost estimate includes for 5 equipped boreholes within a 2km 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 600 m^3/hr. This excess water will be pumped to the surface where it will be contained in an evaporation pond 350 m X 350 m in size. To assist with evaporation a series of mechanical evaporators will be employed, the number ramping up over the period, to assist with the evaporation of the excess water. 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; andInternational Commission on Radiological Protection (ICRP) guidelines.

The total life of mine production of tailings will amount to 4.39 million tonnes over 12 years of which only 50% will report to the DSTSF, equating to 2.20 million tonnes over 12 years. The remainder of the tailings produced will report to backfilling of underground stopes. The average rate of filter belt tailings reporting to the DSTSF is 500 dry tonnes per day.

Item	Criteria	Value	Source
1	Ore type	Uranium	GAC/METC
2	Design Life of Facility	12 years	METC
3	Average Filter Belt Tailings Production Rate	1,000 tpd	METC
4	Tailings Production Split to Backfill Operations	50%	METC
5	Average Filter Belt Tailings Backfill Rate:	500 tpd	METC
6	Average Filter Belt Tailings Deposition Rate to DSTSF	500 tpd	METC
7	Total Filter Belt Tailings	4.39 Mt	METC
8	Total Filter Belt Tailings reporting to DSTSF	2.20 Mt	METC
9	Particle Specific Gravity	2.59	SGS
10	Average Particle Size Distribution	32% passing 75 µm	SGS
11	% solids to water ratio (by mass)	80	METC
12	Average Dry Density	1.66 t/m^3	SGS/Epoch
13	Delivery Method	Trucked	METC
14	Geochemistry	Potentially Acid Forming	The Moss Group

Table 18-1: Design Criteria for the Dasa Dry Stack TSF

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 a number of the Class C liner layers can be removed or adjusted as follows (from top to bottom):

Finger drain 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.

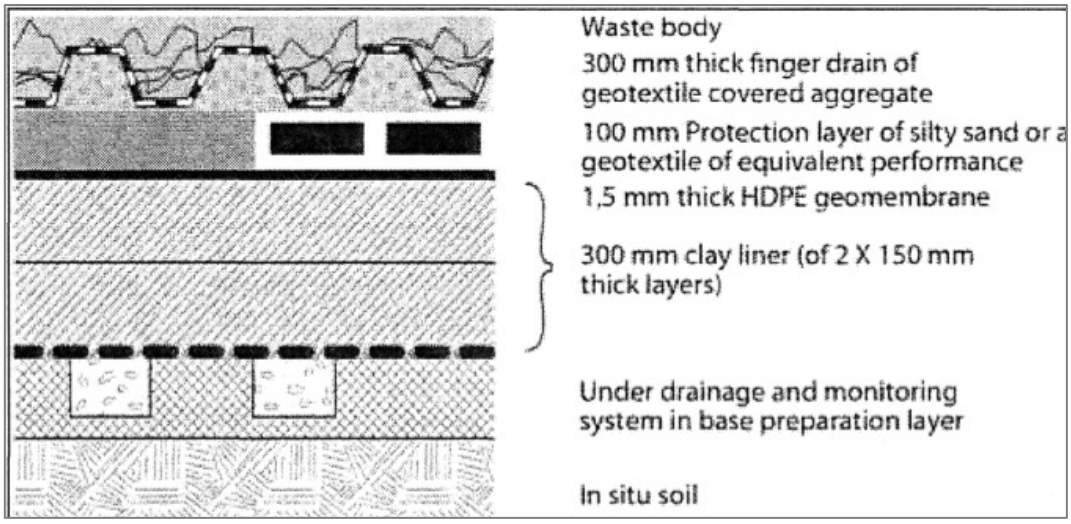


Figure 18-2: Engineering design requirements for a Class C Landfill (SA National Norms and Standards for the Disposal of Waste to Landfill

The preferred site for DSTSF development was revised following the Site Selection study completed by Epoch in 2020.The final site selected for the 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 fairly 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);

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 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.

TD 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	2.29	3.09	3.65	2.97
Cumulative years of tailings deposition	2.29	5.38	9.03	12
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

Table 18-2: Key Parameters Associated with the Dasa DSTSF

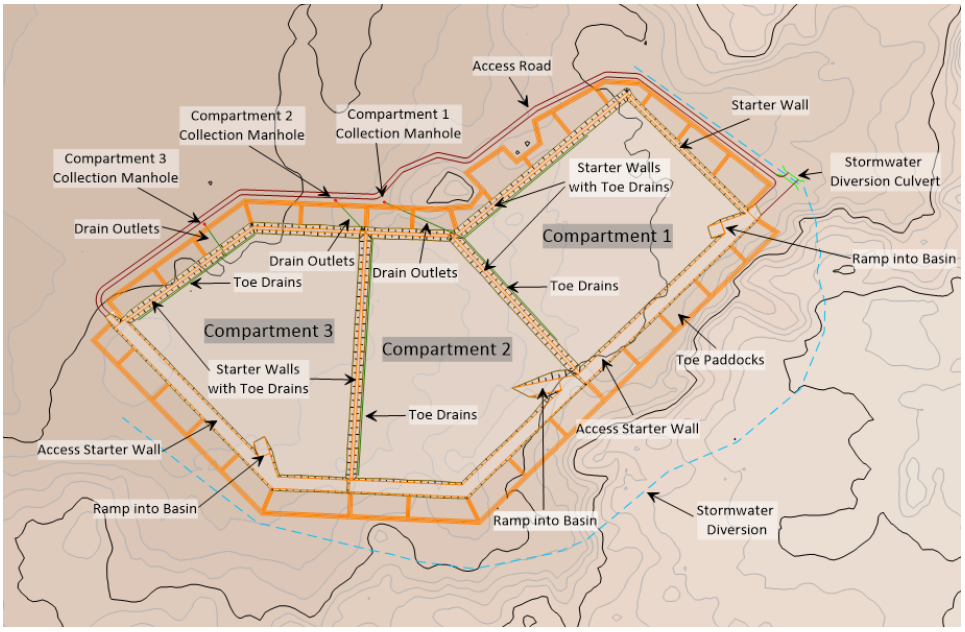
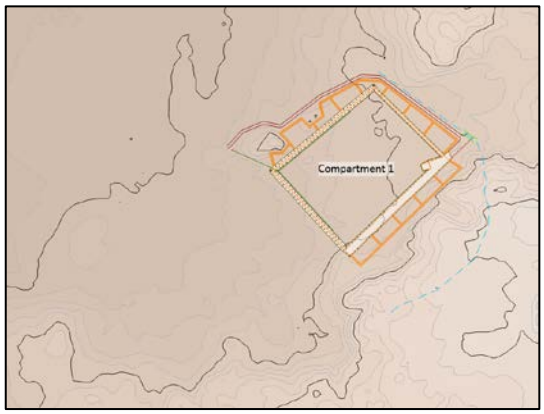
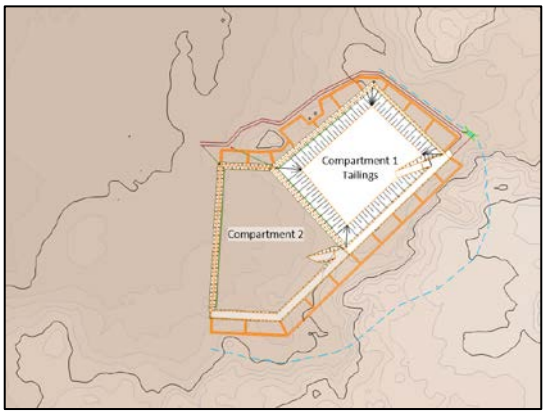


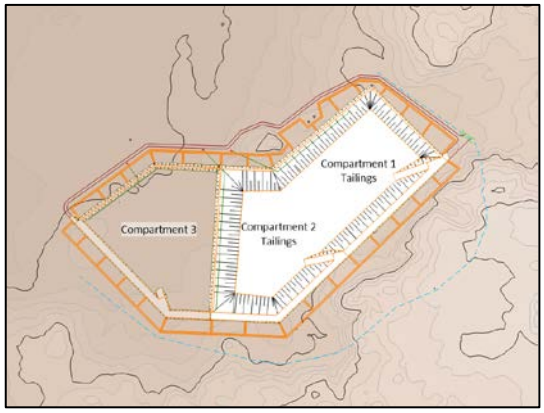
Figure 18-3: Infrastructure Layout of the Dasa DSTSF



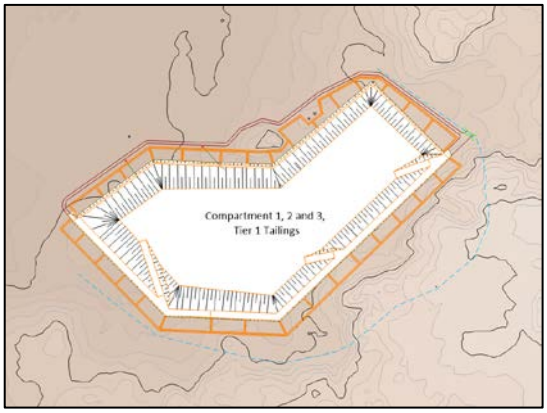
Compartment 1



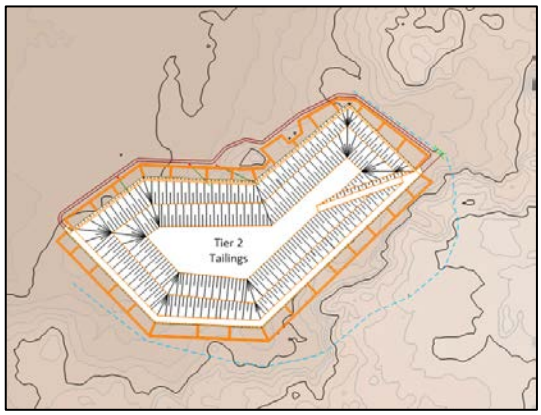
Compartment 2



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.

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 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.

Off-gases from the sulphuric acid plant will be treated by means of a caustic scrubber which will ensure that the effluent produced by the sulphuric acid plant conforms to the applicable environmental standards.

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. These goods will arrive via the port of Cotonou in Benin and be 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. 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 plantsite.

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 prefabricated in nature. 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. Prefabricated buildings or converted containers will be used for sections such as the kitchens, stores, and offices.

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.

The mosque will be stressed membrane structure and include the main prayer area and ablution facilities.

Accommodation Camp

An accommodation camp will be constructed approximately 7km from the mining operation and will consist of accommodation units with configurations to house 3 levels of seniority. There will be a single canteen with the requisite food storage, preparation, cooking, and housekeeping facilities servicing all staff. Recreation, security, access control, laundry and administration units will be provided at the camp. The camp will be constructed from prefabricated units, with a combination of standard structures and some specifically designed facilities and shall be able to house 450 people. The facility will likely be 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 AFFF (aqueous film-forming foam) 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 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.

Any temporary infrastructure that is required for the earthworks and preliminary construction work will be provided for by the appointed contractor.

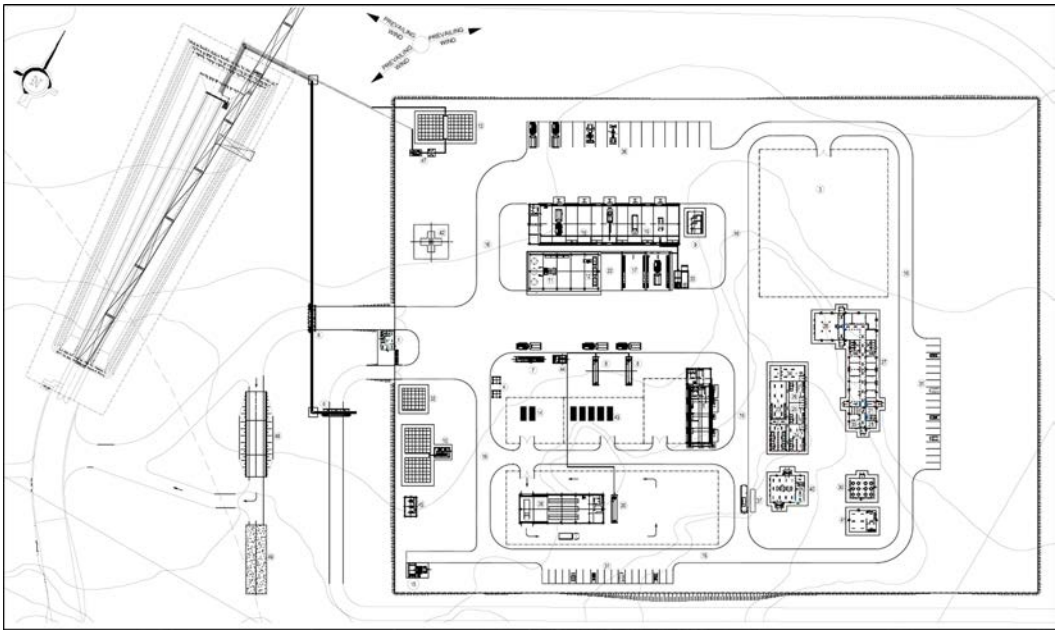


Figure 18-5: Proposed Layout of Surface Mining Infrastructure

Table 18-3: Surface Mining Infrastructure Provisions for Dasa

Mark No.	Description	Mark No.	Description
1	Main Gate Security Office	31	Parking - Light Vehicles
2	Control Room	32	Potable Water Storage Tank
3	General Laydown Area	33	Oil/Water Separation Sump
4	Emulsion Silos	34	Laundry
5	General Workshop	35	Mine Rescue Services Room
6	Pipe Gantries	36	Parking - Mining Vehicles
7	Lubrication Station	37	Personnel pick-up / drop off
8	Diesel Refuelling Stations	38	Mine Store
9	Compressor House - Surface	39	Lamp Room
10	Fire Water Storage Tanks	40	Muslim Prayer Room
11	Tyre Storage and Handling Bay	41	Helipad
12	Tyre Inflation Safety Cages	42	Generators
13	Service Water Storage Tanks	43	Diesel Offloading Transfer Pump
14	Sub-Station	44	Compressor House - Underground

Mark No.	Description	Mark No.	Description
15	Sewage Collection Sump	45	Decline Portal
16	Internal Roads	46	Settling Facility
17	Mining Vehicle Wash Bay	47	Brake Test Ramp
18	Mining Vehicle Workshops	48	Sand Pit
19	Support Vehicle Workshops		
20	Support Vehicle Refuelling		
21	First Aid Station		
22	Support Vehicle Wash Bay		
23	Change House – Workforce Men		
24	Change House – Workforce Ladies		
25	Change House – Management Men		
26	Change House – Management Ladies		
27	Main Office Block		
28	Open Plan Office		
29	Perimeter Fence		
30	Canteen / Restroom		

Access and Internal Roads

Provision has been made for unsealed surface roads with a 150mm 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 2.4m high diamond mesh fence. Provision has been made for a pre-fabricated 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 2.4m 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 150mm. Provision has been made for pole mounted, energy efficient LED type floodlights to illuminate the area.

Emulsion Silos

Bulk emulsion for underground blasting will be stored on site is iso-tainers provided by the explosives supplier. Provision has been made for a bunded concrete pad for placement of four iso-tainers 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 all the boilermaking, fitting and electrical 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 67,100L Containerised and self bunded tanks.

1 off Diesel offloading / Decanting facility.

2 off Diesel dispensing and metering pump units.

4 off 2,000L 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 of 50NB 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 and back-up emergency generator system. Power will be reticulated to the surface infrastructure, ventilation fans and underground areas using cables and overhead lines at 11kV. The single line diagram for the MV reticulation is shown in *Figure 18-6*.

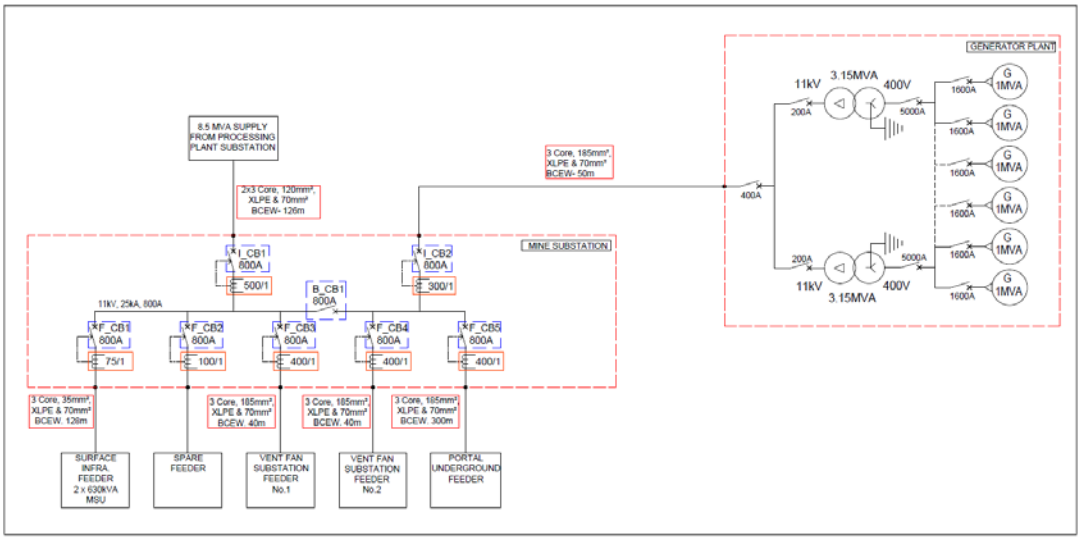


Figure 18-6: Dasa Mine MV Reticulation

The calculated maximum demand for the Dasa mining operation is 7.1MW, 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	4 000	3 220
Pumping	7 800	2 860
UG Infrastructure	410	220
Mining	2 600	430
Total	15 630	7 130
Peak Operating Load (MW)		**7.1**

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 is a prefabricated type building which will be assembled and installed on a reinforced concrete slab. Provision has been made for “air-lock” 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 is 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 a prefabricated type building which will be assembled and installed on a reinforced concrete slab. Provision has been made for “air-lock” 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.

Mine Rescue Services (MRS) room.

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 a prefabricated type building which will be assembled and installed on a reinforced concrete slab. Provision has been made for “air-lock” 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, prayer room, 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 330m^3, in a pressed steel type tank positioned on surface at the mining cluster. The storage tank will be installed on purpose designed, 5m 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 50NB 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 is 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 building is a prefabricated type building which will be assembled and installed on a reinforced concrete slab. Provision has been made for "air-lock" 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.

Prayer Room

The prayer room building is a prefabricated type building which will be assembled and installed on a reinforced concrete slab. Provision has been made for “air-lock” 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 building is equipped with benches and racks for the storage of shoes as well as a washing facility.

Generator Station

Diesel generators will provide emergency power in the event of a power outage at the mine. The calculated peak emergency power demand for the mining operation is 3.1MW (3.9MVA). Which includes critical surface infrastructure, 1 x main ventilation fan, de-watering pumping load and critical underground services. Based on the 3.9 MVA requirement – 5 off 1MVA generators have been provided for in a 4 running and 1 standby configuration. The generators will be housed in 20ft containers. Further provision has been made for transformers and switchgear, both also housed in 20ft containers. All these containers will be placed on level concrete pads. 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.

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 18-7.

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 80mm 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.

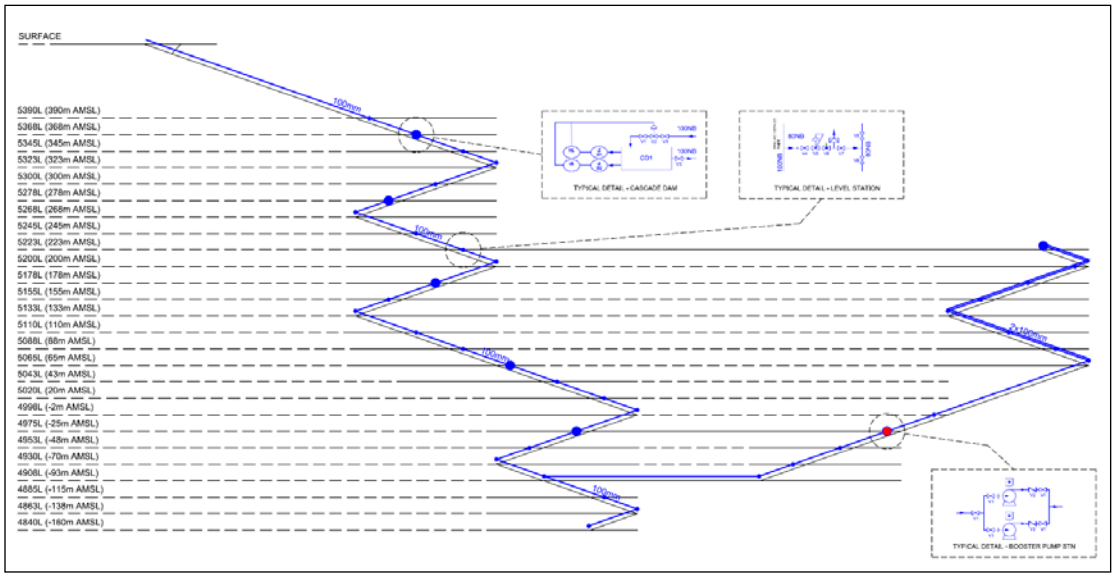


Figure 18-6: Mine Service Water Schematic

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 18-5 presents the dirty water handling requirements for 2030, which is the point in the mines life where the pumping requirement reaches the maximum.

Table 18-5: Underground Water Handing (Pumping) Requirement 2030

Description	Total	Units
Service Water		
Service Water per Day	0.73	ML/day
Peak Service Water Drilling Shift	16.8	l/s
Ground Water		
Ground Water Inflow	21.39	ML/day
Ground Water Inflow	247.60	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	22.12	ML/day
Pumping Duty	264.4	l/s
Water Balance		
Total Inflows	21.39	ML/day
Service Water (Recirculation)	-	ML/day
Ground Water	21.39	ML/day
Water In Backfill	0.32	ML/day
Total Outflows	1.23	ML/day
Footwall Loss	0.04	ML/day
Ventilation Loss	0.79	ML/day
Rock Hoisting Loss	0.25	ML/day
Backfill Loss	0.14	ML/day
Total Underground Makeup (-) / Excess (+)	**20.17**	**ML/day**
Total Underground Makeup (-) / Excess (+)	**840.21**	**m^3/hr**

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 lesser quantities of ground water are extracted from the production workings. As such a 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 includes 400 mm nominal diameter steel piping with 25 bar flanges for PS1 to PS3, 200 mm nominal diameter steel piping with 25 bar flanges for PS4 and 300 mm nominal diameter steel piping with 25 bar flanges for PS5. 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 reticulation will require bearer support such as the transition points from the vertical service raise to the horizontal development.

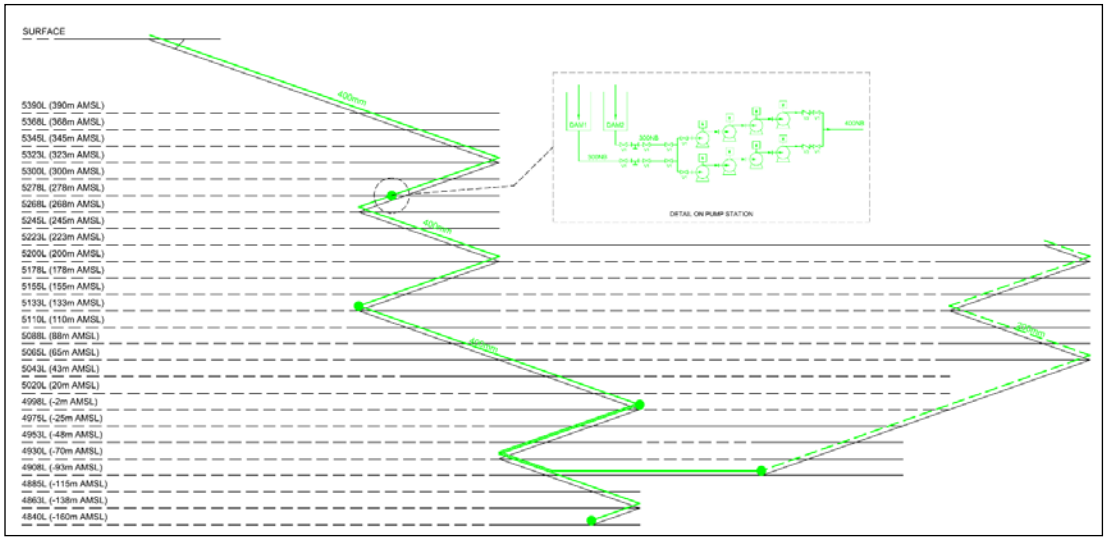


Figure 18-8: Mine water handling and dewatering system schematic

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.

Due to the expectation that the ground water inflows will predominately 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 100m drill cubbies are developed and a 200m 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 80mm 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 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.

Table 18-6 summarises the parameters that were used for calculation of the backfill supply requirements for the reticulation design.

Table 18-6: 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 system where variable conditions will occur.

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 in the control of flow.

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 18-9 shows a schematic arrangement of the piping distribution underground. The system will consist of a ‘backbone’ piping 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.

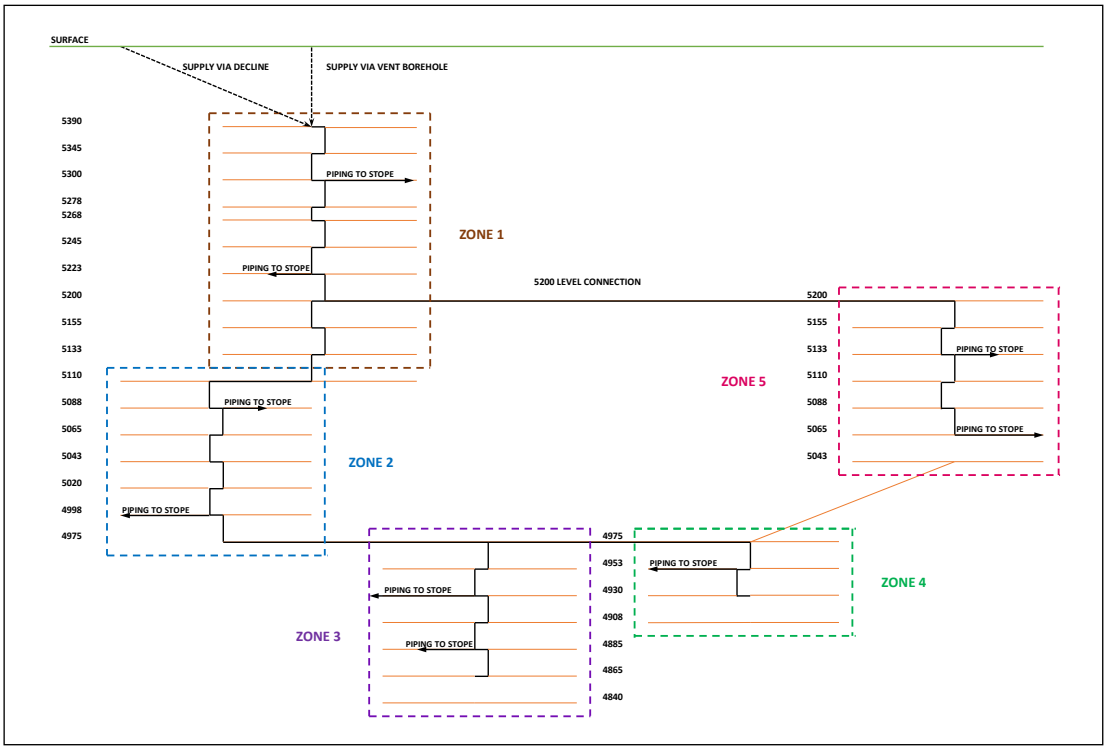


Figure 18-9: Schematic showing underground piping distribution

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.

Based on a nominal flow of 45m³/h, the flow to all areas will be identical. 80 NB piping is used in all zones as the flow will not be sufficient to provide an acceptable flow velocity using 100 NB piping. In zones 1 to 3 50NB choke piping is used to prevent the development of sub atmospheric pressures and has resulted in a common pump pressure for all layouts. The longer lengths of horizontal piping in zones 4 and 5 produce higher pressure losses resulting in higher pressures at the pump. Wear rates within the 50 NB piping would be high and these pipes should be seen as being short lived sacrificial components.

Pressures can be reduced by using larger 90NB piping on the surface feed, 4975 level to supply zone 4 and on the 5200-level connection. However, 90NB is not a common pipe size and has a limited wall thickness range.

Backfill barricades will be located 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. 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

Consideration to types of barricade design included reinforced concrete walls, wire lacing and mesh, wooden barricades, steel fabrications and waste rock piles. 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.

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 zone1. From the production levels the MV reticulation will be installed in the service raises with each feeder accessing alternating main station levels.

Figure 18-10 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 400V.

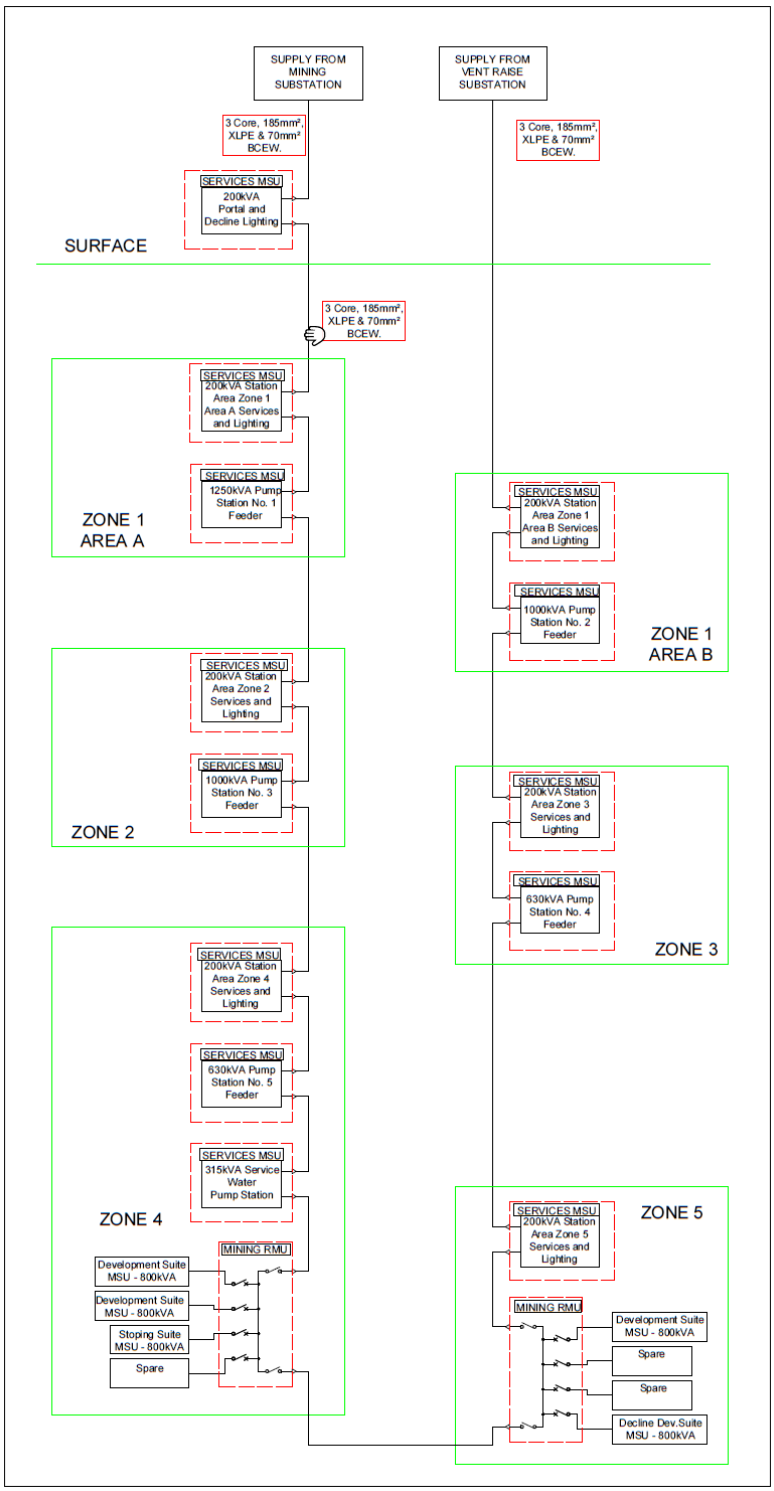


Figure 18-10: Dasa Underground Mine MV Reticulation

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 & 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, supplemented by occasional investor/fund demand (or secondary demand). Currently there are 443 operable reactors supplying about 10% of the world's electricity, representing 29% of low carbon generation (Figure 19-1). Indeed, in advanced economies over the period 1971 to 2018, nuclear power has produced the greatest share of low carbon electricity. As the climate agenda gathers pace, nuclear power is well placed to increasingly support the decarbonisation of electricity grids around the world (Figure 19-2).

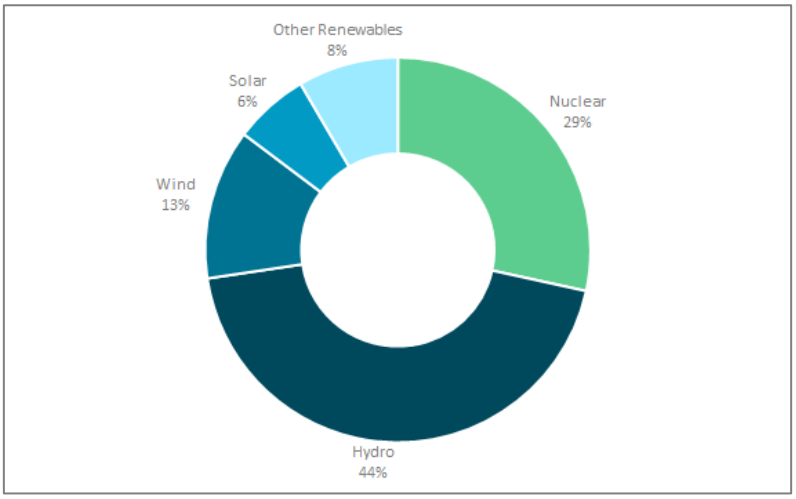


Figure 19-1: Low Carbon Electricity Contribution

Source: IEA

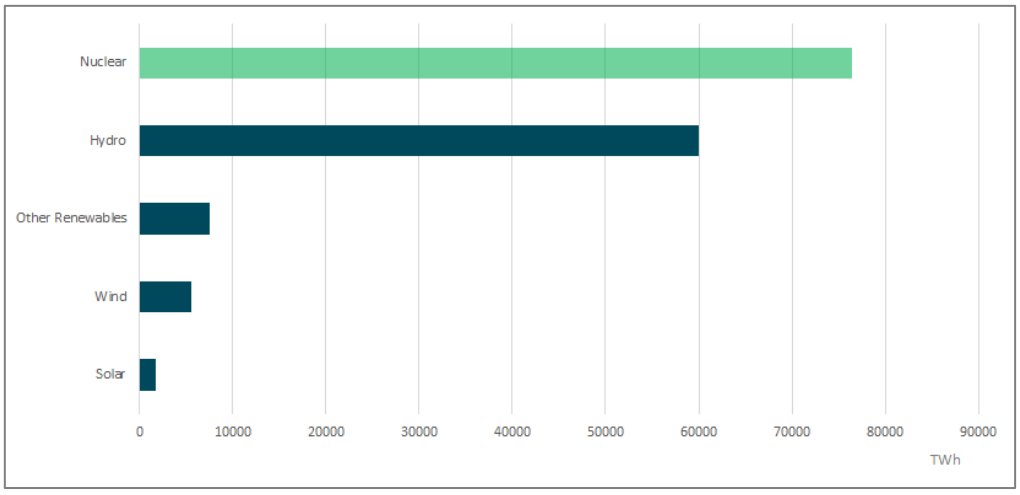


Figure 19-2: Contribution to Low Carbon Electricity in Advanced Economies (1971-2018)

Source: IEA

There are currently 52 reactors under construction, including 14 units in China. At its peak in the late 1970s, 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 93.5% in 2019, compared 34.8% 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 a 1 GW reactor.

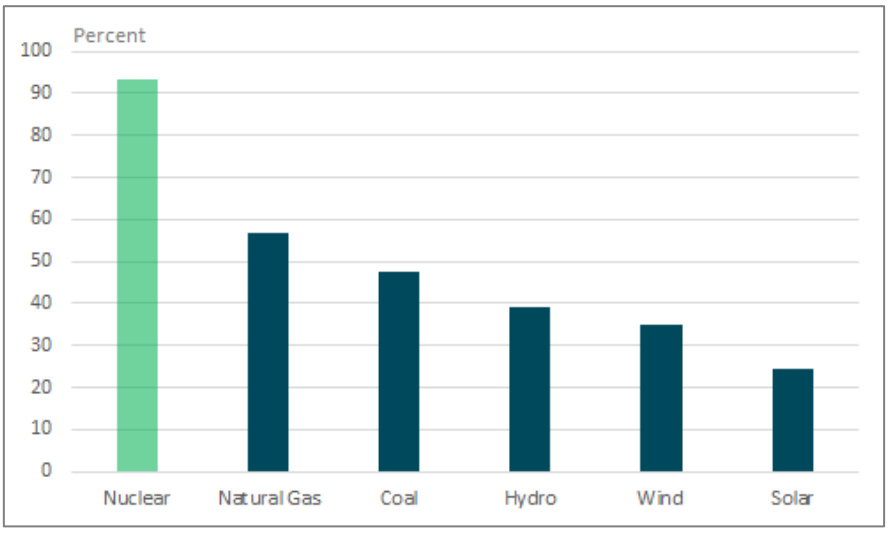


Figure 19-3: U.S. Capacity Factors by Source (2019)

Source: 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.

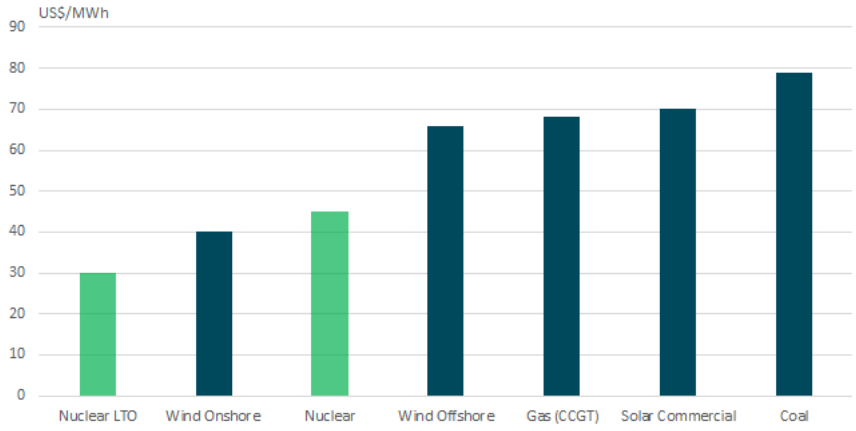


Figure 19-4: Levelized Cost of Electricity with a 3% Discount Rate

Source: EIA

Financing is consequently a pivotal variable in defining the relative competitiveness of nuclear power. The current low interest rate environment combined with post pandemic fiscal stimulus creates a favourable landscape for significant growth in nuclear power. The advent of new technologies will provide further upside to the sector. Small Modular Reactor (SMR) technology is gaining considerable support and sponsorship from regional governments. These units are typically less than 300MW in size and are quicker and cheaper to build. The flexibility of SMRs 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 by 2030.

Moreover, as the COP26 Climate Summit in Glasgow, U.K. came to a close in November 2021 it has become clear that nuclear power will play an increasing role in decarbonising global electricity grids. In a policy U-turn, President Macron declared that France would construct new reactors to maintain energy security and meet its climate change goals. Further, the Chinese government singled out nuclear power as the only energy source to receive an interim target in its official five-year plan. This was followed by the state utility, China General Nuclear Power Corp, defining a need for 200 GW of installed capacity by 2035. This would require approximately 150 GW of new capacity (or about 70 m lbs of additional annual uranium demand). Meanwhile, the UK government backed Rolls Royce's SMR design, intensifying competition in that sector.

Not inclusive of these upsides, but capturing current plans alone, the World Nuclear Association (WNA) in its reference case uranium demand forecast defines a 30% increase in reactor needs over the 10 years to 2031. 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.

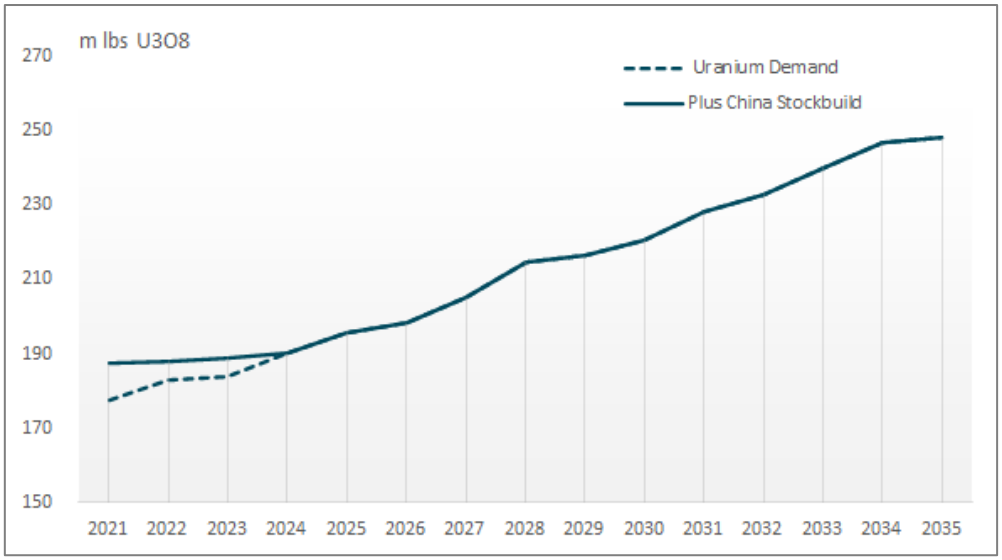


Figure 19-5: Uranium Demand

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 (Figure 19-6). Where excess enrichment capacity exists in the market (as is the case today), more enrichment can be applied to less uranium to generate the required level of enriched uranium. The offset process in effect yields uranium.

Inventory: This tranche of supply is the most opaque component in the uranium market and represents

i) the extent that utilities and suppliers drawdown their strategic inventory and
ii) where historic excess supply has been carried on the balance sheets of producers, utilities, traders and financiers for future use or sale.

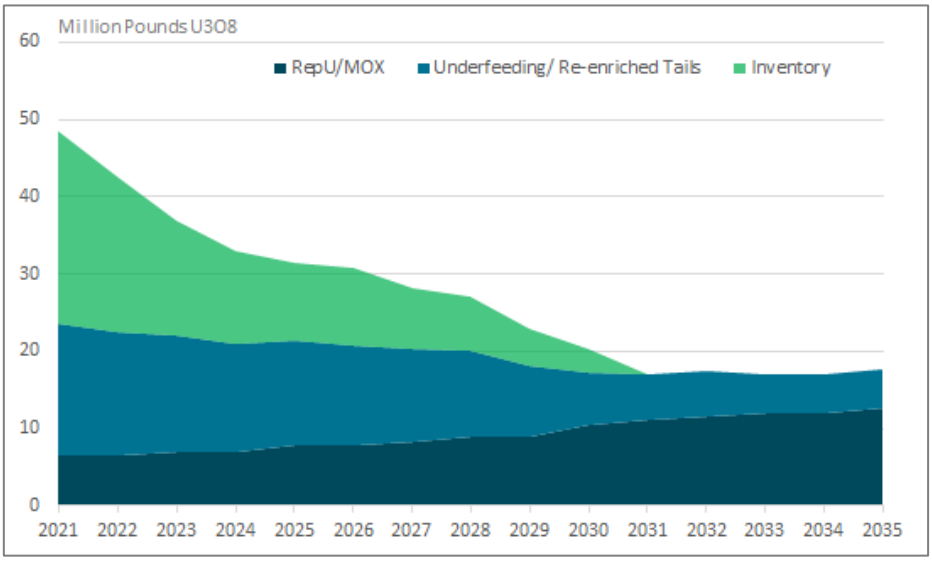


Figure 19-6: Secondary Supply

Source: WNA Reference / Fuel Link

Deducting secondary supply from uranium demand defines the call on primary production – Figure 19-7.

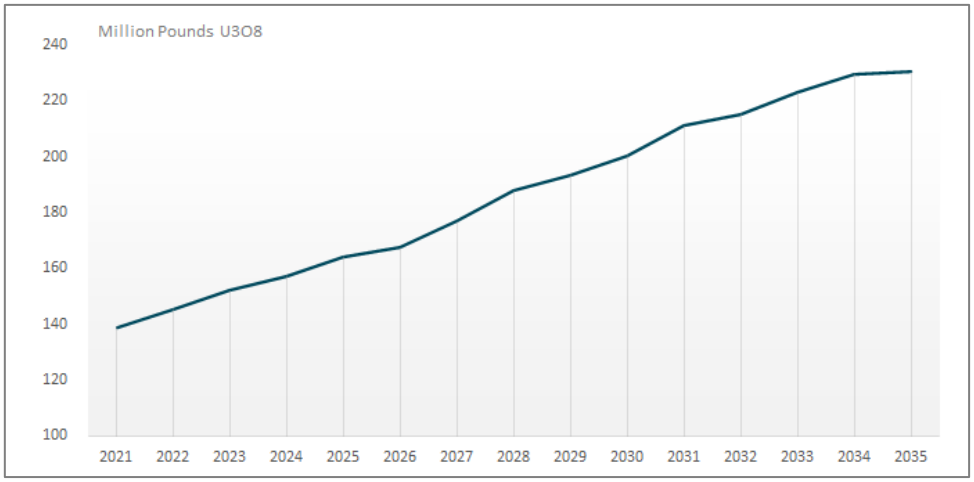


Figure 19-7:Call on Primary Production

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 past 5 years. This in turn leads 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. When undersupply crystallises, yielding a commensurate price reaction, the mining base will again be inelastic. Idled capacity aside, new production will typically take years to come online.

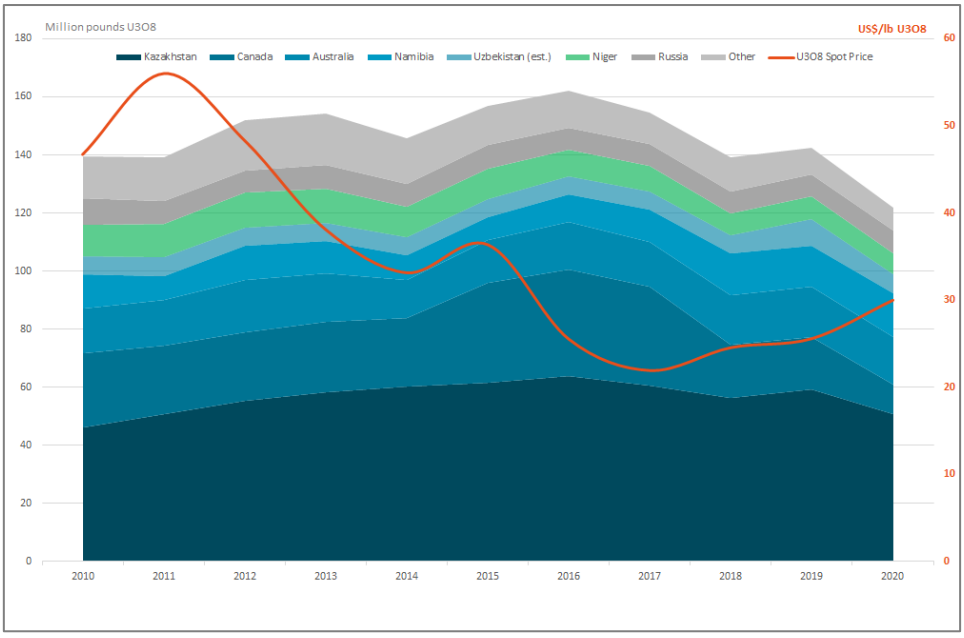


Figure 19-8: Historic Production and Uranium Price

Source: WNA Production / TradeTech Prices

While the WNA identifies the potential for over 80 million pounds U_3O_8 of new annual capacity (Figure 19-9), it is important to note that the vast majority of this output will require prices in the \$40-\$70/lb U_3O_8 range to be commercially viable.

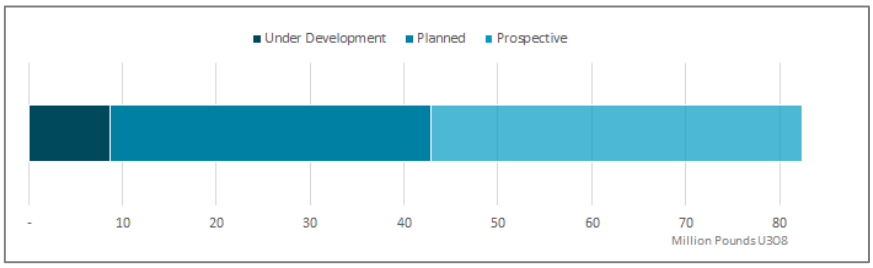


Figure 19-9: New Production Capacity 'Under Development', 'Planned' and 'Prospective'

Source: WNA

The recent past and future uranium production (including the potential of the idled capacity), stockpiling by China and WNA requirements are presented in Figure 19-10

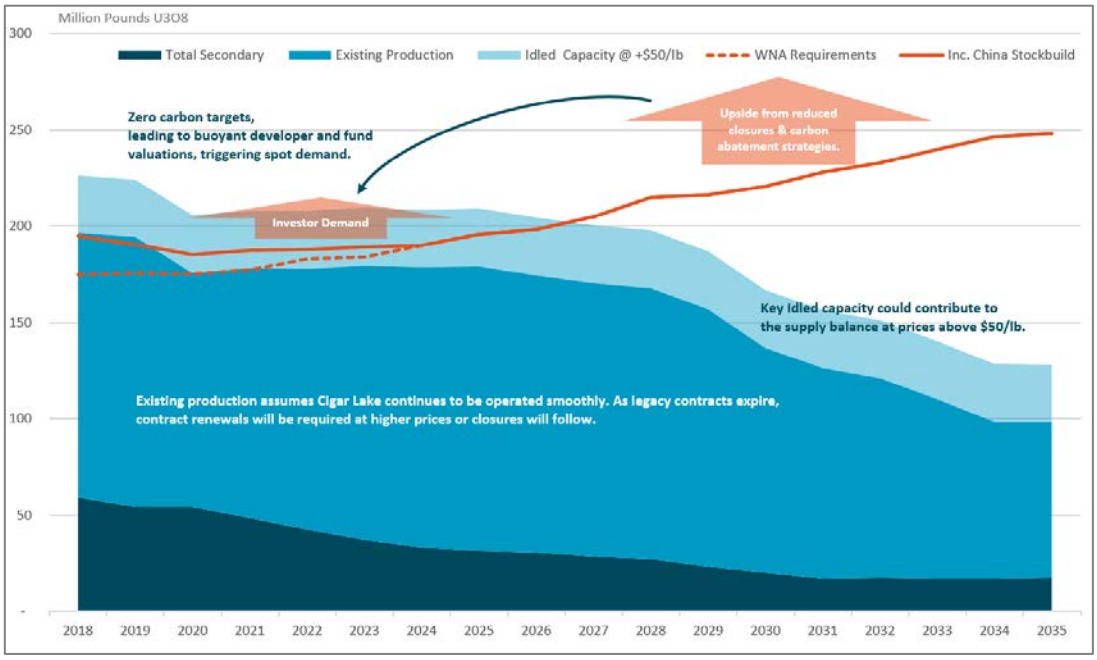


Figure 19-10: Uranium Supply & Demand Projection

Source: WNA / Fuel Link

There is currently a consensus view amongst industry analysts that uranium prices will need to increase to

i) prevent the further erosion of the established supply base, and

ii) incentivise the introduction of new output (Figure 19-10).

and this increase in the price of uranium is reflected in Figure 19-11

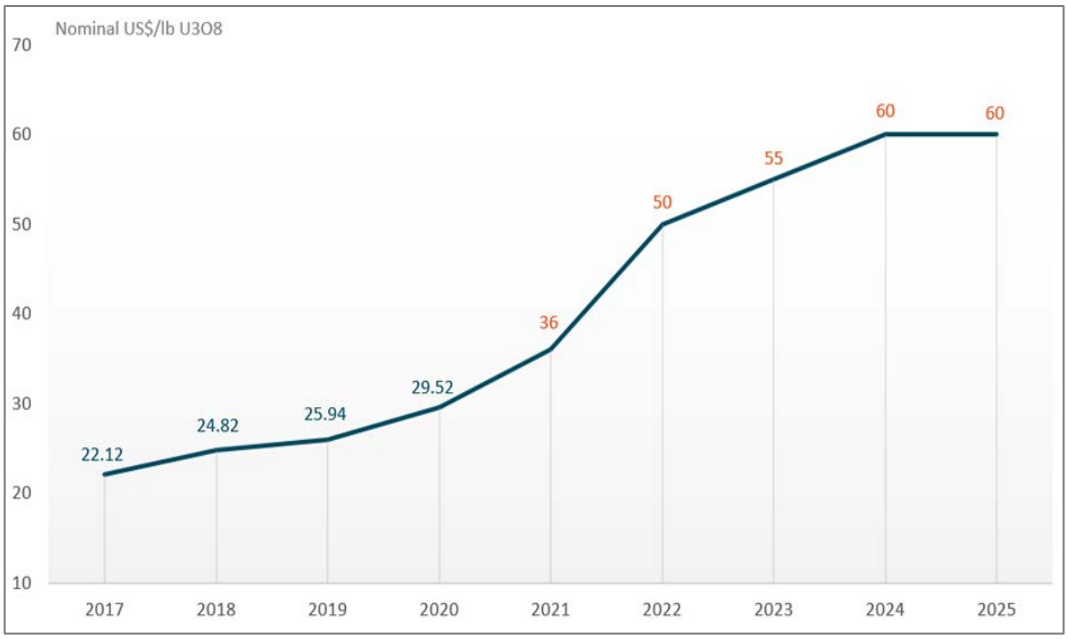


Figure 19-11: Uranium Spot Price Projection

Source: Eight Capital

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

i) the fuel buyer has sufficient confidence and trust in the supplier and
ii) 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 1–3-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 essentially utilise the low cost of capital available to traders to carry surplus spot supply into the future.

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. In Figure 19-12 below, the 2019 and 2020 numbers for the USA reflects a concerted period of contracting for the region – mostly conducted off-market. The latest figures for the EU are for 2019 only.

In addition, there will be notable contract openings in Asia and the rest of the world. China is a significant buyer requiring approximately 25 million lbs U_3O_8 per year, rising to 35 million lbs U_3O_8 by 2025. Moreover, earlier this year, the nation has accumulated significant inventories already, it will need to enter the market for further strategic volumes. Other key buyers include Korea Hydro Nuclear Power (KHNP) with total annual needs of approximately 10 million pound 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.

USA **European Union**

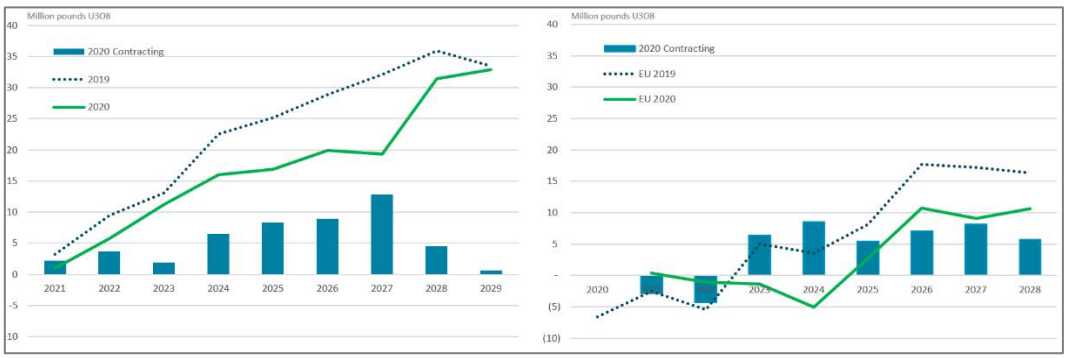


Figure 19-12: *Uncovered Demand*

Source: EIA

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 65%.

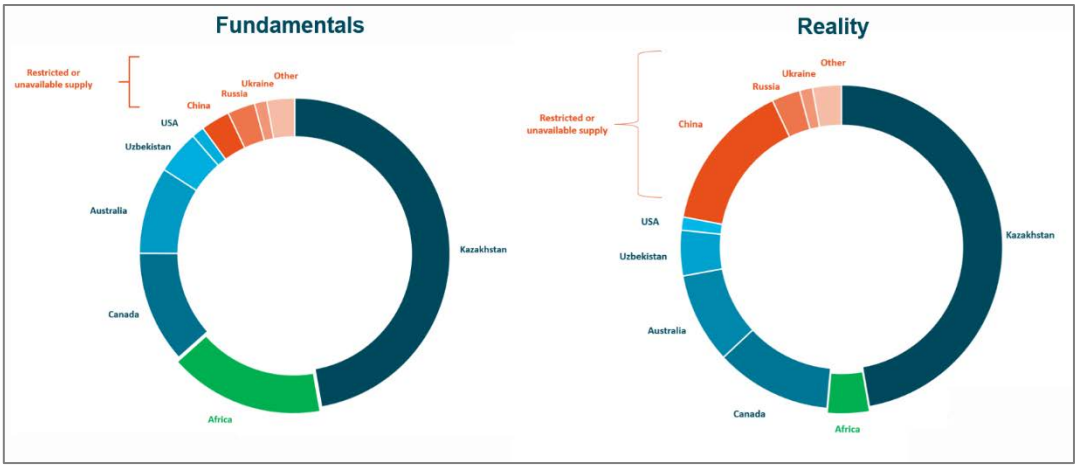


Figure 19-13: 2025 Existing Production & Geographical Diversity

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.

Low market prices have combined with Covid disruptions to reduce 2020 primary production. While 2021 output is expected to be improved, most idled capacity will require in excess of \$45/lb U_3O_8 to be viable (Figure 19-14 - Installed Capacity and Production (2020)).

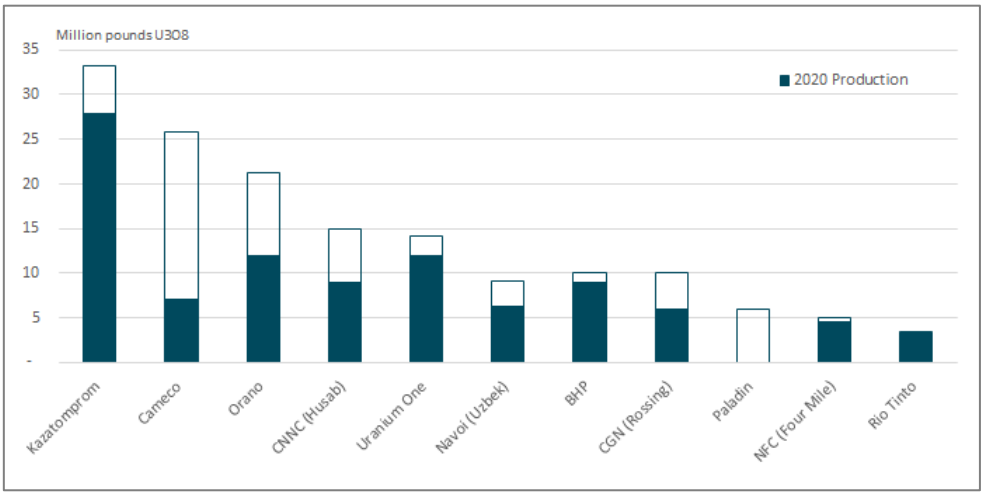


Figure 19-14: Installed Capacity and Production (2020)

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. Since 2019, the premium has averaged 20% (Figure 19-15)

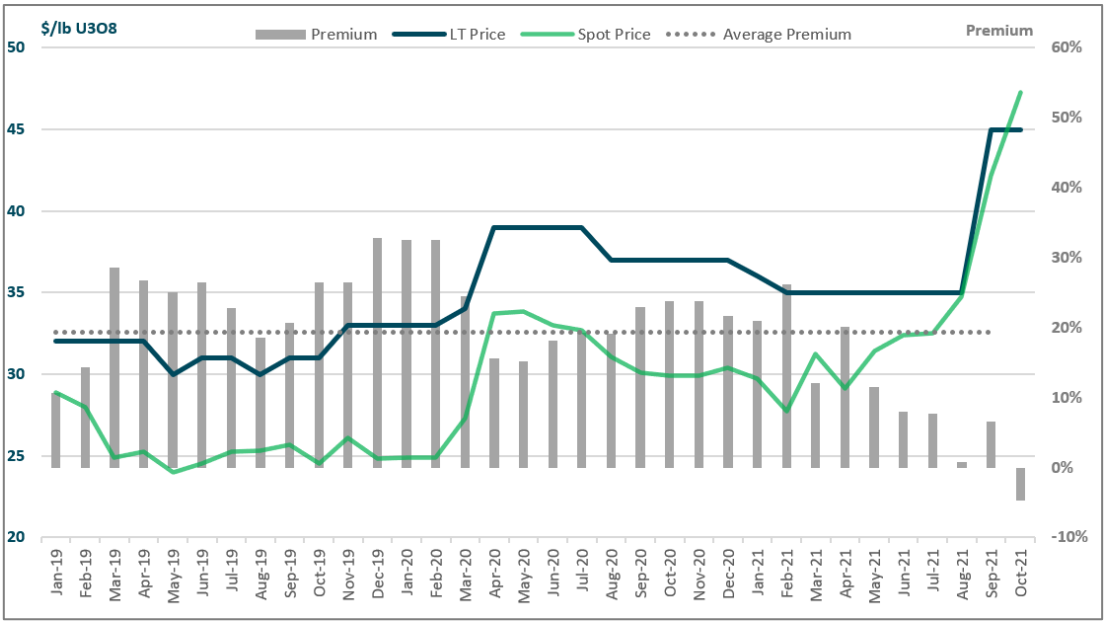


Figure 19-15: Spot Price vs Long Term Price

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 is also increasingly becoming a feature of the market. Relative to utility

demand, investor volumes are 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. In addition, 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 approximately $15bn 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. Consequently, they have become 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. 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 19-16.



Figure 19-16: Uranium Spot Price and Fund Inferred Price

Source: TradeTech / UPC

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 to a $50/lb level in September.

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. However, it is important to note that utilities have largely been spectators to the price rise. As discussed, they make the vast majority of their purchases in the forward market, and so far, have been fairly inactive.

For further context, the uranium market typically operates in ‘contango’, where in real terms the spot price is lower than the forward price. However, the recent price volatility has created a position of ‘backwardation’, where forward prices are lower than the spot price. Importantly, however, the uranium term price has also experienced good support. TradeTech’s monthly term indicator increased from $35/lb in July to $45/lb in September (Figure 19-15).

Looking forward, funds, are likely to continue providing a positive impact on market dynamics, as well as help to lock away surplus spot supply and improve market fundamentals.

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 has subsequently been developed and approved by the board. The strategy will remain dynamic and dovetail with the financing requirement of the project. Fuel Link has initiated a market introduction program and will continue to educate the fuel buying community on the attributes of Global Atomic. The key objectives are to convey the compelling characteristics of Global Atomic's uranium supply capability, ensure the company is included on utility Request for Proposal (RFP) bidder lists, and to progress off-market opportunities where possible.

Crucially, there is no other uranium developer that combines the following competitive supply characteristics:

- Cashflow from an established business.
- Recent 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 and permitted.

These factors are demonstrated in Figure 19-18

The marketing strategy aims to balance business security and finance support with market exposure. While the market fundamentals point to firming prices, the level of inventory overhang is opaque. Inventory will inevitably expire, but it is a difficult to forecast the timing of when undersupply will crystallise. Moreover, when prices rise beyond the \$45/lb level there will be increased competition for contracts as returning production capacity conflates with higher cost developer pounds.

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 Atomic's ability to establish 'seed' contracts in the current market environment, which are then leverageable in the future, creates a competitive differentiator in the market.

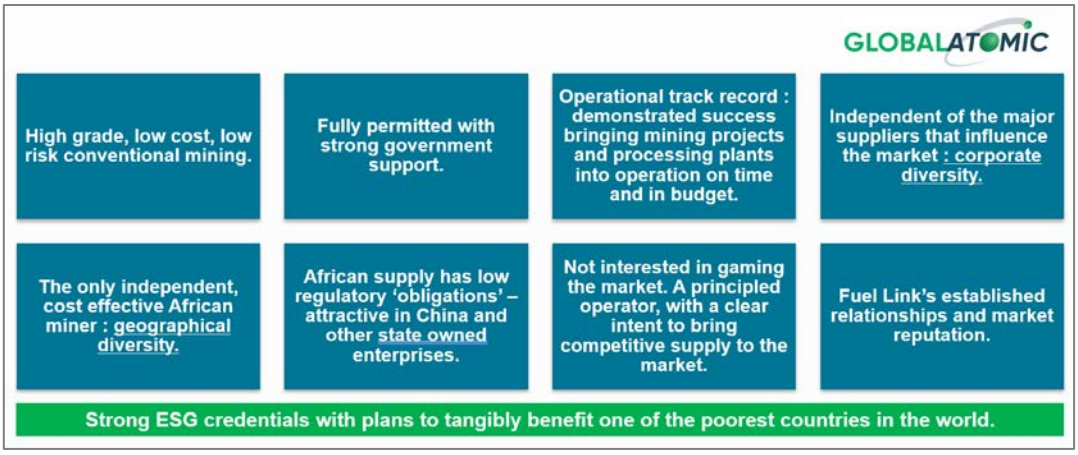


Figure 19-17: Global Atomic Uranium Supply Characteristics

The marketing strategy will remain flexible, be reviewed regularly, and adapt to the changing market environment.

Safeguards, Physical Delivery & 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 19.19). 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.

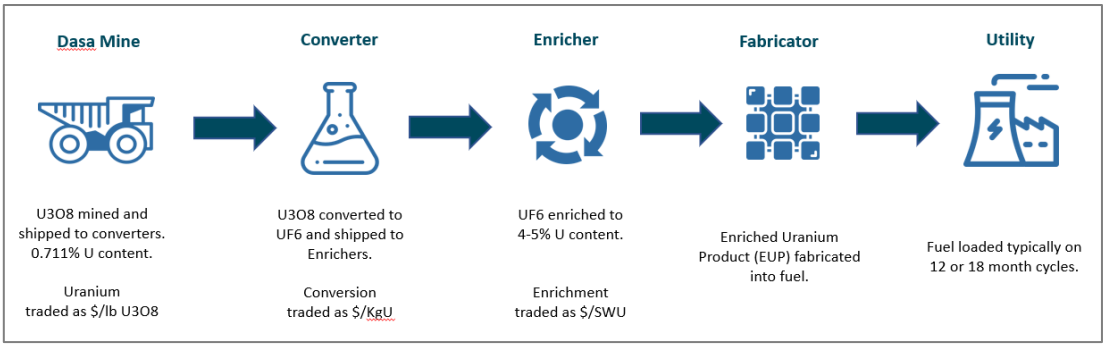


Figure 19-18: Front End Nuclear Fuel Cycle

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") specific to the Dasa Project Area associated with the Phase I Mine Plan subject of this Feasibility Report. The ESIA updated previous environmental and socio-economic base-line 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 Environment and Social Management Plan ("ESMP") at the start of the ESIA process. The MESUDD is responsible for developing guidelines for the ESIA; analysis and evaluation of the ESIA; validation of the ESIA through stakeholder meetings and the negotiation of the ESMP.

The 2020 ESIA serves to update the ECS completed by Groupe Art & Genie in 2011 and includes hydrological and air quality baseline data compiled during the interim period and addresses environmental, social and economic impacts associated with construction of the mine, mining operations and reclamation and closure. The ESIA includes an analysis of alternative design and operating scenarios in order to reduce environmental and social impacts associated with the Mine.

A key element of the ESIA is public consultation to inform stakeholders of the proposed project, and technical studies related thereto. Community meetings were held in area nomadic villages and small towns as part of the ESIA process and were organised by MESUDD and attended by Government and GAC personnel. A copy of the ESIA is kept at the MESUDD to enable stakeholders the opportunity to access and review the documentation. 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 Environmental and Social Management Plan ("ESMP"). The ESMP governs the Company's activities from construction through operations and mine closure and defines the protocols for Project monitoring and reporting to government ministries and agencies. The ESMP also outlines the management procedures, mitigation, monitoring and

reporting protocols GAC will implement to avoid, reduce and manage environmental and social impacts during the construction, operations and closure phases of the Dasa project.

The ESIA was completed and submitted to the MESUDD for review and approval 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 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 is issued for an initial period of ten (10) years subject to automatic five (5) 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, capacity building and budgets for the relevant ministries and agencies and, the “Environment and Social Charges Book” which establishes implementation, monitoring and reporting protocols related to the implementation of 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.

20.2. Existing Environmental and Social Setting

Location, Climate and Terrain

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 kms to the north, Tchirozerine 60kms and Agadez 95 kms respectively to the south.

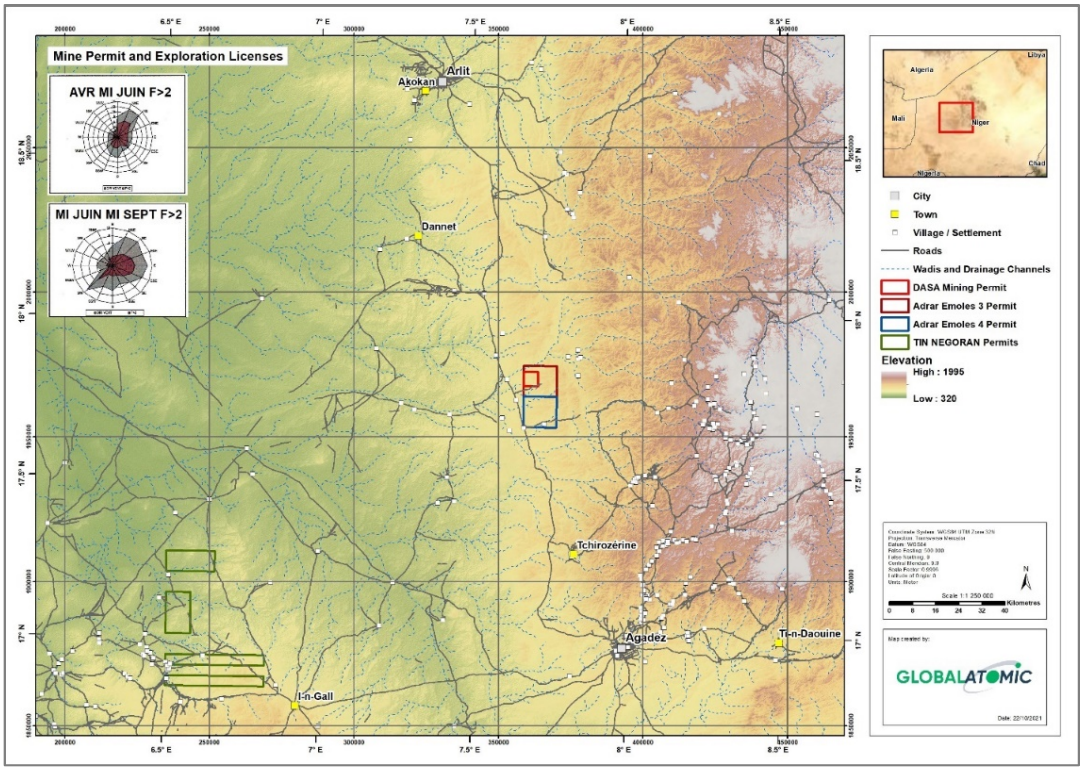


Figure 20-1: Regional location of the Mining and exploration permits

Under the 2021 ESIA Groupe Art & Genie identified a 7km radius area of influence zone and a 15km radius extended area of influence zone around the mine site. The 7km 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 7km perimeter is considered the Project safety perimeter. The 15km zone is defined as an extended area of influence and includes the towns located on the Agadez – Arlit Highway.

Villages proximal to the 7km 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 where water is available on a seasonal basis. The towns of Tagaza and Egatrak are on or proximal to the 7km perimeter and located on the Agadez – Arlit Highway.

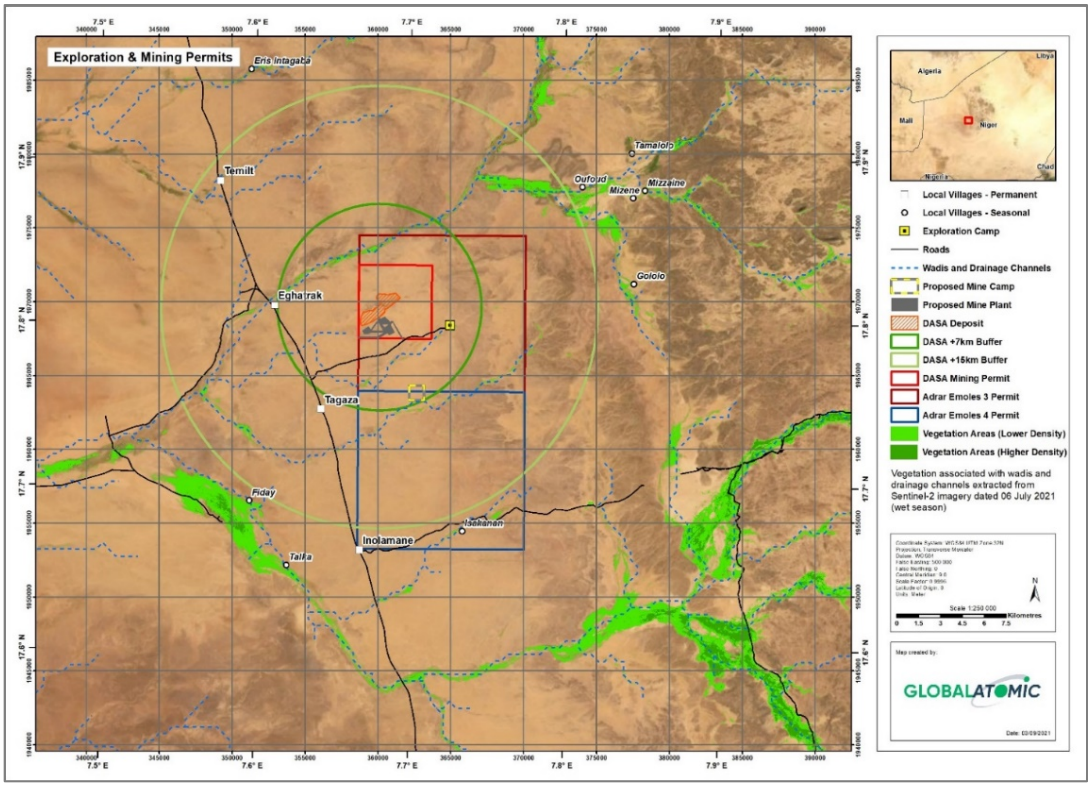


Figure 20-2: Map of the project area showing 7km and 15km buffer zones and proximal communities

The Dasa area is in the desert climate region of the Sahel-Sahara, characterized by an arid climate with a rainy season from July to August, a cool dry season from September to February and a long, hot dry season from Feb to June. 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 109.8mm Temperatures range between 31° and +50°C in the hot season and between 0° and 20°C in the cold season. The average annual temperature is 37°C; the average annual maximum temperature is 45°C and the average annual minimum temperature is 11°C.

The Dasa area has little wind. Prevailing winds in the dry season are from the north-east followed by those from the north-north-east – the Harmattan winds characterized by the dry and dusty north easterly trade wind, of the same name, which blows from the Sahara over West Africa into the Gulf of Guinea. During the rainy season, prevailing winds are from the south-west with 50% of these winds greater than 9 km/h with a maximum of 35 km/h. Average annual velocity of the wind is 17.6 km/h. Sandstorms and windstorms occur primarily during the rainy season. Average annual maximum humidity is 34% and minimum is 13%.

The regional relief in the permit area has an average elevation of approximately 500mamsl and is marked by alternating outcrops and sandy plains in a desert landscape, traversed in places by koris (river valleys, most often dry). The highest elevations are found in the Aïr Mountains (the source of most of the waterways) located approximately 80 kms east of the Project Area.

The lowest elevations are found in the Isouwarwar Valley, with an average of 460 metres. The deposit is located in an area with an elevation of approximately 470 m.

Air quality

The current sources of airborne emissions in the area are largely natural, consisting 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 region are associated with the coal fired power station outside Agadez and from existing uranium mining operations around Arlit, albeit at some distance from the project. No air quality data (gaseous, particulate or deposition) has so far been collected. Baseline data collection for dust deposition, directional dust, SOx and NOx (sulphur and nitrogen oxides) at appropriate receptor locations will be established during Q1 2022.

Noise and vibration

The remote location and the absence of human habitation and industry mean that anthropogenic noise levels are currently low; no 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

The hydrology and hydrogeology of the Dasa area has been studied by Groupe Art & Genie with a hydrology report completed in 2013 and again by CSA Global Mining Industry Consultants (“CSA Global”) in 2021.

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 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 hydrogeological boreholes installed at the Dasa site providing in-depth insights of individual lithology hydraulic properties and hydrogeological characteristics of the project site.

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 ranges from 27-47m 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 5mm per year. Recharge, when it occurs, will primarily occur via infiltration along alluvial drainage channels.

Groundwater quality is generally of good chemical quality with the majority of parameters tested falling within World Health Organisation (WHO) drinking water standard limits. 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. It was noted that the WHO limit for gross alpha activity in drinking water (0.5 Bq/l) was exceeded in 8 of the 15 boreholes sampled (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 in 2 of the 15 boreholes sampled (exceedance values of 1.55 and 2.45 Bq/l).

The hydrology of the Dasa area is determined by the presence of koris (ephemeral watercourses). Several koris traverse the Dasa area, the most significant being the Solomi, Issakanan, Elagozan and Agatara. Water flow is generally from east to west and the koris are essentially dry outside of the rainy season. A phenomenon characteristic of koris is flash flooding during the rainy season, with the rainfall events causing observed high flows often originating several kilometres upstream of the drainage basin. Extensive study and mapping of the basins and koris has been conducted to identify suitable locations for the plant and camp sites.

Groundwater quality

Groundwater quality samples have been collected from 23 borehole locations within the project area between 2013 and 2021. The groundwater quality data has been collated and includes in-situ chemical testing for field parameters, laboratory chemical testing for major anions and cations, radiological analysis, and bacteriological analysis.

A review of the groundwater quality data indicates that groundwater is generally of good chemical quality with the majority of parameters tested falling below World Health Organisation (WHO) drinking water standard limits, with the following exceptions:

One sample tested in 2013 at the Global Atomic Base-Camp borehole marginally exceeded WHO limits for nitrite, however concentrations of nitrite from samples tested in 2020 and 2021 fell below the WHO limits.

One sample tested in June 2021 at the HYDRO3 borehole exceeded WHO limits for fluoride.

One sample tested in March 2021 at the Ikakan borehole exceeded WHO limits for fluoride.

A review of the bacteriological test results indicate that some 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. A review of the results indicates 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.

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.

Whilst the ground water is generally of a good chemical quality however, once the ground water enters the underground workings, it will potentially be contaminated with nitrates from the explosives and oils and greases from underground mining equipment. Mine water will report to the water treatment plant prior to discharge to the environment. Other alternatives to manage the volume of ground water entering the mine workings, will be investigated in the next phase of the project, including upstream extraction of ground water through a borehole field and reintroduction into the aquifer some distance from the mine site.

Soil and Land use

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

The Dasa area is sparsely occupied; the only habitation consists of nomad camps and small sedentary villages. Agriculture in the form of seasonal market gardening occurs immediately proximal to a number of koris outside the Dasa Project area.

Livestock herding and breeding is the main economic activity in the project area. It is carried out by all sectors of the population, irrespective of ethnic group, gender, or social category. In general, family stock farming involves several species of animals: goats, sheep, donkeys, and camels. Livestock farmers have traditional lands with which they have almost emotional relationships and to which they return at certain times of the year. The primary activity is leading their livestock between sources of fodder and water. Herding routes follow the koris which run from the Air Mountains in the east to the lowlands in the west. Koris proximal to the Dasa Project Area are outside the 7km buffer zone. Traditionally, the land belongs to the village's founding family and is acquired by households through inheritance, donation or extended families.

No seismic events or earthquakes have been recorded in the Dasa area.

Biodiversity and Critical Habitat

A desktop study of the Project and surrounding areas was recently completed in accordance with the requirements of 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 ongoing biodiversity field work.

The previous ECS and ESIA identified several 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 The 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 with the Project Area.

A Critical Habitat screening has been carried out in accordance with PS6 and GN6 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 50km buffer around it. More detailed screening was then undertaken within a more localised study area (the Dasa mining Project location with a 15km 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 and meet 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 Critical Habitat Desktop Study.

Acid Rock Drainage and Metals Leaching

Test work was conducted by SGS, Lakefield, Ontario in March 2011, on a 100 kg bulk sample of “ore” which included Modified Acid Base Accounting (ABA) and Net Acid Generation (NAG) testing, and the following results were reported:

	D1 Comp	D2 Comp	D3 Comp	Overall Comp
Net NP (t $CaC0_3$/1000t)	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@pH7.0 (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.

Additional representative samples of development waste rock have been collected and sent for comparable testwork and analysis by SGS Canada. The objective of the testwork is to confirm that development waste rock is similarly non - acid generating and will be completed during Q1 2022.

Radiological baseline

The Project area is located in a region of elevated background radiation due to the natural presence of high concentrations of uranium in the rocks and soils. Key exposure (dose) routes 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 drinking water which contains radionuclides.

Baseline surveys have focused on:

measurements of external exposure dose rates at several points in the area of the deposit;

measurements of uranium 238 concentrations in soil samples;

measurements of the global alpha and global beta volumetric activity in samples from the water supply sources (wells and boreholes) in the nomadic villages and towns within a 20 km radius around the Dasa deposit.

The ESIA provides results for 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 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 varies 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).

Historical baseline data is being updated in line with the local requirements for baseline monitoring over a period of 12 months. GAC has established four dosimeter stations which monitor alpha, beta, gamma and radon on a quarterly basis. When the baseline data collection is complete, an additional assessment of potential exposure will be calculated and compared with exposure rates elsewhere to Naturally Occurring Radioactive Materials (NORM) and health-based dose rates. This assessment will take into account the nomadic characteristics of the local population and their reliance on their livestock for milk, cheese, and meat. A realistic exposure scenario will be defined with the relevant state institutions in order to establish a formula for calculating the cumulative exposure doses that these populations are likely to receive in a year including any increase in exposure that can be attributed to the uranium mining operations.

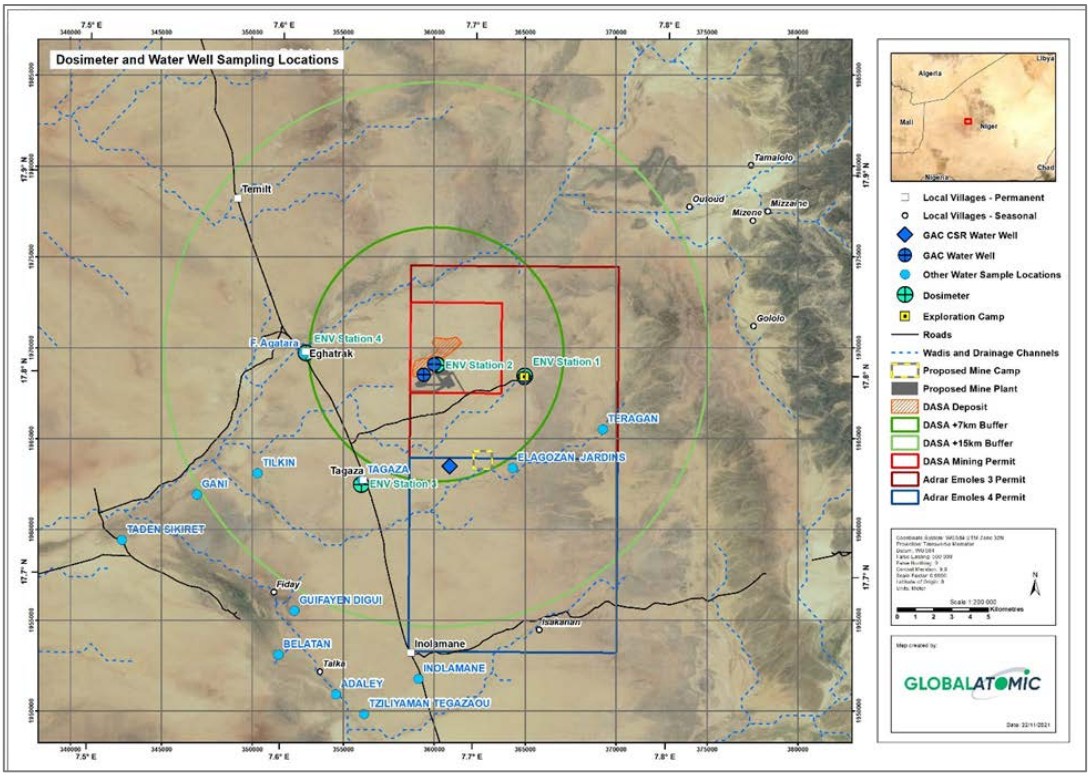


Figure 20-3: Dosimeter and Water well Sampling locations

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 of Tuareg origin is made up 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).

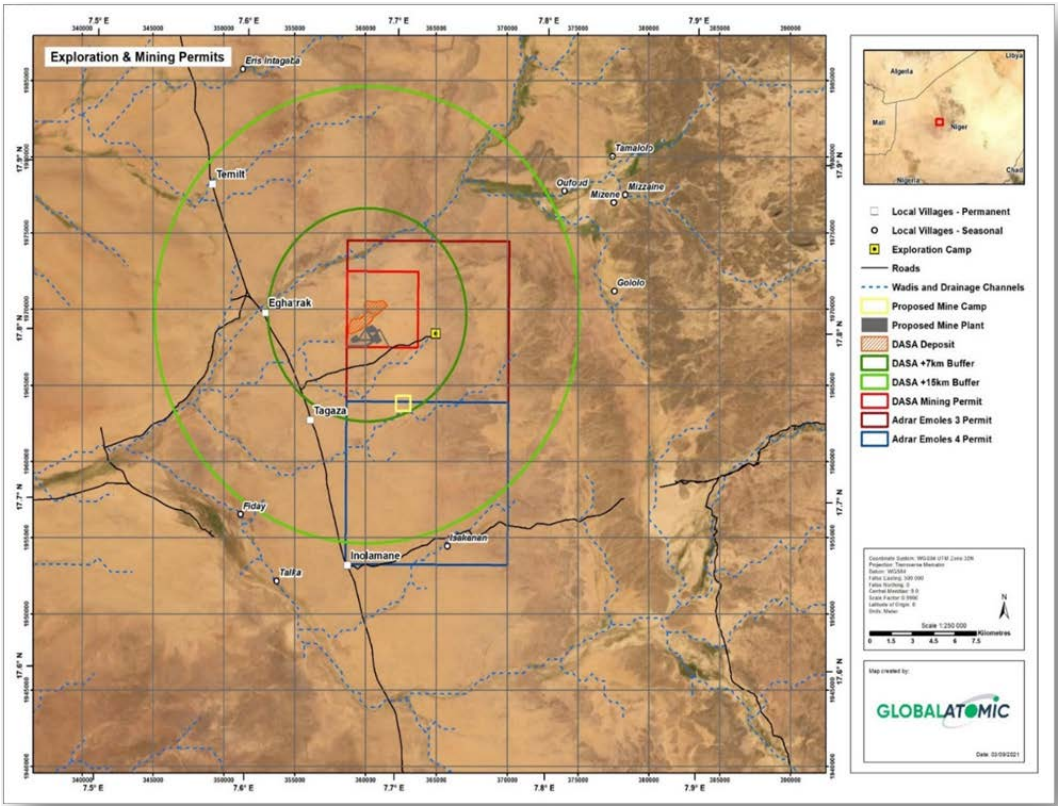


Figure 20-4 - Project location showing 7km and 15km buffer zones and nearby communities

The nomadic Tuaregs have historical connections with their natural environment and temporarily migrate 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. The social group hierarchy remains very much intact with the Tuaregs. The elders and opinion leaders react, talk, and make decisions on behalf of the group.

With the decline in traditional livelihoods, young people in particular migrate to urban areas in search of alternative lifestyles. Economic activities are traditional and mostly based on livestock. The Tuareg are true nomads and follow their herds following source of browse, mostly in or near wadis. Most Tuaregs live in tented camps near wadis and move readily seeking pasture for their livestock.

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 entity; 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.

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. It is anticipated that this migration to the zone will further increase with the development of new mining operations proposed by mining companies, particularly the Dasa Mine. Other opportunities for paid work are limited, although the existing uranium mining industry around Arlit is a significant employer. Orano's Cominak Mine has recently closed, which has led to the loss of long term, good paying mining sector jobs. This represents an opportunity to the Dasa mine which will have a pool of skilled local workers to draw from in staffing the mine.

The Dasa Mine will ensure the continuity of local, regional and national tax revenue, contribute to social and economic stability of Arlit and provide significant spin-off economic opportunities through the procurement of goods and services for the mine, mill and camp.

Archaeology and Cultural Heritage

GAC has recently completed a baseline field program to identify archaeological and cultural artifacts within the Project Area. None of the sites identified fall within the project footprint. Project area maps will be updated to include the location of any such items found and appropriate policies and procedures to monitor and protect such items will be established, including a chance finds procedure.

A rock carving known as the Dabous Giraffe (circa 6,000 BCE) is located approximately 10 kms north-west of the Mine site. Once a tourist attraction, the Giraffe site is largely unvisited as tourism is a fraction of what it once was due to security concerns. Access is by sideroad off the main highway approximately 10kms north of the turn-off for the Dasa Project.

Two graveyards are located in close proximity to the Dasa Project exploration camp. GAC plans to erect fencing around the graveyards and include signage to restrict access to family members.

20.3. Impact Assessment

Potential impacts were identified arising from the construction, operating and closure phases in the 2020 ESIA. These include impacts to the biophysical environment and human environment. Impacts were rated as Major, Moderate and Minor (without mitigation) as summarised below. Post mitigation, residual risks were not assessed. It is important to note the remote location of the project in relation to human receptors. No resettlement or economic displacement will be required. It should be noted that with the integration of industry standard mitigation measures into project design and management, these rankings will be reduced.

Valued environmental and social components	Construction	Operation	Closure
Air quality	Moderate	Major	Moderate
Soil	Moderate	Major	Minor
Surface Water resources	Moderate	Moderate	Minor
Ground Water resources	Moderate	Major	Moderate
Vegetation	Major	Moderate	Moderate
Wildlife	Moderate	Moderate	Moderate positive
Landscape	Moderate	Moderate	Minor
Noise and vibration	Moderate	Moderate	Minor
Jobs and income	Major positive	Major positive	Moderate
Local and regional economy	Major positive	Major positive	Moderate
Health and Safety	Moderate	Major	Not assessed
Archaeology and cultural heritage	Minor	Minor	Not assessed
Population and community changes	Moderate	Major positive	Moderate

Table 20-1: Summary of unmitigated impacts

Analysis of Alternatives

The following alternatives were considered and described in the ESIA:

mining methods;

ore processing method;

power supply;

mine water supply;

staff accommodation and services;

management of waste rock and mine tailings;

management of non-mining waste;

wastewater treatment;

stormwater management;

redevelopment, rehabilitation, and management of the site post closure.

Project alternatives considered in the Feasibility Study will be evaluated and summarised in the ESIA Addendum which is scheduled for completion H2 2022.

Mitigation Measures and ESMP

Across the project, mitigation measures in line with good international industry practices (GIIP) have been proposed. These are described in the ESIA and also in the ESMP, which forms part of the package submitted to the MESUDD and approved prior to the issuance of the Mining Permit.

The ESMP is a framework for managing activities for the effective and efficient implementation of the various measures proposed for the effective integration of environmental and social concerns. It describes the methods for implementing, supervising, and monitoring the recommended measures to prevent, minimize, mitigate, or compensate for negative environmental and social impacts or to increase positive impacts. The ESMP ensures compliance with GAC's environmental commitments and will contribute to strengthening GAC's participation in the sustainable socio-economic development of local communities. The ESMP is structured around:

- a program for mitigating and/or improving impacts
- an environmental inspection/oversight program;
- an environmental monitoring program;
- a program to strengthen the capabilities of stakeholders.

The Environmental Permit (Cahier des Charges or CCES) takes the same format as the ESMP included in the ESIA and addresses a number of project phases:

- Construction impacts mitigation and management
- Operations impacts mitigation and management
- Closure impacts mitigation and management
- Environmental and Social inspection/oversight program
- Environmental Monitoring program
- Roles and responsibilities
- Training
- Budget for capacity building, monitoring, and reporting programs
- Total costing to implement the management plan

The planned ESIA addendum offers an opportunity to align the mitigation hierarchy with the project design during Detailed Engineering, to ensure that as far as possible, impacts are

avoided, minimized, and managed. As well as a review of impacts in the light of the Feasibility Study design and additional baseline information, the project ESMP will be further developed to incorporate the requirements of the environmental permit, including a series of Management plans, detailed in Section 20.6 below.

20.4. Community Engagement & Support

Global Atomic has been engaging with local communities since their arrival in the area in 2008. Initiatives consist of 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/procurement

Regional and national procurement

The sectors which have received support since 2008 are detailed in Table 20.2 below. To date, education and training has been limited to employees of the project and annual contributions to the Ministry of Mines for the training of Ministry employees.

Additional future development support will be delivered in collaboration with NGOs currently active in country and will 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.

Table 20-2 also indicates anticipated increased levels of support and new programs during the construction and underground development phases in the period 2022-2014.

As the project ramps up into commercial operations, the Corporate Social Responsibility contributions will be reviewed with reference to the success of projects to date and priorities identified in consultation with communities.

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

Table 20-2: Summary of community support initiatives undertaken and planned by GAC

Formal consultation engagement undertaken as part of the 2020 ESIA took the form of a series of meetings in the communities around the project area, including Tagaza, Agatara, Issakanan, Sikiret/Tadant, Oufound, Mizeine, Ghalab, the Kelezeret Tribe and Inolamane.

Environmental concerns included potential effects of uranium mining, contamination of the food chain, human and animal health risks; occupation and loss of pastoral areas and crop lands, destruction of vegetation and loss of wildlife habitat; impacts to water resources, management of waste from the mining operations and restoration and rehabilitation of the mine.

Social concerns included population displacement, marginalization of local communities, the security of cultural and tourist sites, employment opportunities for young people from the local communities and management of labour risks. Community development concerns included youth training and jobs for young people (8 villages each), a water well (7 villages), sanitation (6 villages), construction of a health post (4 villages), market gardening (3 villages), business

opportunities and a food bank (2 villages each) and support for animal health and protection of the koris (1 village each). The consultations assisted GAC in selecting initial priority projects that will be monitored on a regular basis as the project moves into the construction and operations phases. A Stakeholder Engagement Plan will be developed to formalize community engagement and provide a Grievance Mechanism.

20.5. Next steps

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 onsite station

Systematic biodiversity studies (further detailed below)

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) (IFC 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.

The field work and related studies are being completed by a local consultancy using scopes of work prepared by various international specialists, and will include community surveys to include ecosystem services, timing and routes of nomadic migrations, numbers of permanent and seasonal residents; biodiversity surveys to include mapping vegetation types present in the 15km radius zone and surrounding area and classifying them as either Natural or Modified Habitats, identifying the presence and/or likelihood of species of conservation concern, particularly focussing on but not limited to the presence of threatened species identified from the CH screening.

GAC will Interview local residents regarding animal sightings and hunting practices; known cultural heritage and archaeological sites within the study area and the likelihood that unknown sites of interest may be encountered during construction and operation of the Project. Other data collection will include noise surveys at sensitive receptors, dust deposition and directional dust as well as air quality parameters (NOx and SOx). Human Rights and gender-based violence (GBV) assessments will also be undertaken.

The ESIA Addendum will be compiled by international environmental specialists with extensive experience of mining projects and include the following:

Introduction

Regulatory framework including national legislation, EP4 and GIIP

Project Description (from FS)

Baseline descriptions: climate, geology, soils, biodiversity, land use, air quality, noise, hydrology and hydrogeology, geochemistry, socioeconomics (including Indigenous people), human rights and community health, ecosystem services, archaeology and cultural heritage

Impact Assessment post mitigation, including cumulative impacts

Alternatives assessment

ESMP

Consultation and Disclosure

Non-Technical Summary

Along with the ESIA addendum, GAC will develop a series of management plans which will incorporate the requirements under the CCES and additional mitigation and management measures. Where appropriate, mitigation measures will be incorporated into project design in the Detailed Design or FEED stage. These management plans are likely to include:

Environment Social Management System (“ESMS”) framework

Health and Safety including radiological protection of workers (from FS)

Human Resources (including GBV in the workplace, forced labour, child labour etc.)

Water Management Plan

Air quality and GHG

Noise and vibration

Hazardous materials (including NORM)

Waste Management

Mine Waste Management (geochemistry, waste rock and tailings)

Archaeology and Cultural Heritage (including Chance Find Procedure)

Community Health, Safety and Security (including Human Rights, influx and a focus on IPs)

Stakeholder Engagement Plan and Grievance Mechanism

Biodiversity Action Plan

Contractor Environmental Management

Land Acquisition/Livelihood Replacement

Emergency Preparedness & Response

Closure Plan

20.6. Rehabilitation & Closure

The ESMP and CCES include provisions for the management of mine closure, to include the dismantling of buildings and demolition of facilities, recontouring the site and restoration of habitats and landscape. Environmental and social costs of closure are included in the ESMP budget as agreed with the MESUDD.

The current Rehabilitation and Closure Strategy (the Closure Plan") includes the removal and sale of mine site buildings, rehabilitation of the tailings management facility and other areas of site disturbance and post closure monitoring.

The Closure Plan will be further developed in the next phase of study in line with international good practice, such as the ICMM Mine Closure Toolkit to meet the requirements of the environmental permit and establish a closure budget.

The Closure Plan will address social implications and costs of closure as well as engineering and environmental aspects and will be regularly reviewed at completion of construction and at regular intervals during the life of mine. Extensions to mine life and the expansion of or construction of facilities will also trigger a review.

The Closure Plan will aim to present a vision for closure with clearly defined outcomes and closure criteria and will incorporate social and environmental considerations. Structures which will remain in place, such as waste rock dumps and tailings management facilities will be designed to remain physically and chemically stable after mine closure, using appropriate return periods.

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 the production of the first pound of uranium, 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, mining equipment, underground facilities, and services up to the point where uranium is first produced.
- Working costs up to the point where uranium is first produced
- Indirect and 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 21-1. 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	1 127 820	-	1 127 820
Surface Facilities	7 400 729	46 591	7 447 319
Emergency Exit & Ventilation holes	2 538 009	-	2 538 009
Surface Utilities and Reticulation	5 800 794	334 926	6 135 720
U/G Facilities and Services	3 816 443	4 020 778	7 837 221
U/G Utilities and Reticulation	907 084	5 962 922	6 870 006
Mining Equipment	11 301 162	32 672 152	43 973 315
Mine Development	14 309 981	69 994 432	84 304 412
Indirect Costs	2 698 860	1 295 652	3 994 512
Contingency	7 485 132	17 149 118	24 634 250
Capitalised Working Cost	4 503 127	-	4 503 127
Total	**61 889 141**	**131 476 570**	**193 365 711**

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	67 315	-	67 315
Box Cut	840 000	-	840 000
Bus Stop (Drop Off Zone)	4 500	-	4 500
Earthworks and Terracing	62 257	-	62 257
General Laydown Area	34 081	-	34 081
Parking Areas	5 678	-	5 678
Site Access Control and Fencing	113 990	-	113 990
Total	**1 127 820**	**-**	**1 127 820**

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	25 333	-	25 333
Canteen and Restroom	53 056	26 309	79 365
Change house and Laundry	425 531	-	425 531
Emulsion Silos	277 979	-	277 979
Explosive Magazines	34 105	-	34 105
Lamp Room	746 698	-	746 698
Lubrication and Diesel Refuelling Stations	415 002	-	415 002
Offices	961 930	-	961 930
Prayer Room	40 902	20 282	61 184
Stores	79 162	-	79 162
Workshops	4 341 031	-	4 341 031
Total	**7 400 729**	**46 591**	**7 447 319**

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 & Ventilation holes Capital Cost

Emergency Exit & Ventilation holes	Project Capital Cost	Sustaining Capital Cost	Total
Lifting Equipment	2 515 450	-	2 515 450
Site Preparation	22 559	-	22 559
Total	**2 538 009**	**-**	**2 538 009**

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 BOQs 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	13 280	66 400	79 680
Dewatering	759 719	3 038 877	3 798 596
Service Water Booster Pump Station	-	198 360	198 360
Underground Satellite Workshop	95 739	95 739	191 478
Ventilation	2 914 675	-	2 914 675
Ventilation Raise Equipping	33 031	621 402	654 433
Total	**3 816 443**	**4 020 778**	**7 837 221**

Table 21-6: U/G Utilities and Reticulation Capital Cost

U/G Utilities and Reticulation	Project Capital Cost	Sustaining Capital Cost	Total
Backfill Reticulation	323 520	1 811 088	2 134 608
Compressed Air	11 294	81 237	92 531
Control and Instrumentation	164 148	1 180 698	1 344 845
Dewatering	96 391	693 332	789 723
Electrical Reticulation	252 802	1 818 383	2 071 185
Pipe Support	38 709	232 742	271 451
Service Water	20 220	145 442	165 663
Total	**907 084**	**5 962 922**	**6 870 006**

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 the table 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	4 707 439	-	4 707 439
Trackless Mining Equipment	6 593 723	32 672 152	39 265 875
Total	**11 301 162**	**32 672 152**	**43 973 315**

The mining fleet used by the contractor will be purchased by GAC, who plan to take over the operation of this equipment on conclusion of the 24-month contract. A summary of the capital cost of the mining fleet is shown below.

Table 21-8: Summary of mining fleet capital cost

Item	Qty	Capital Cost	Total
Development drill rigs Twin boom	3	1,706,656	5,119,968
Bolter	2	1,306,695	2,613,390
Long Hole Drill Rig	1	756,245	756,245
LHDs (14t)	4	1,332,260	5,329,040
ADTs (42 t)	5	1,156,800	5,784,000
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	2	174,118	348,236
Troop carrier	2	157,647	315,294
Total Cost			**22,426,173**

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 cross cuts. The development costs comprise consumable (stores) costs and labour costs.

Table 21-9: Mine Development Capital Cost

Mine Development	Project Capital Cost	Sustaining Capital Cost	Total
Consumables	209 240	25 946 003	26 155 243
Contractor Cost	14 100 740	21 840 738	35 941 478
Labour	-	4 394 159	4 394 159
TMM	-	17 813 531	17 813 531
Total	**14 309 981**	**69 994 432**	**84 304 412**

While these costs were determined from first principles, a mining contactor cost for the first 24 months of development was obtained and included in the cost estimate. A summary of the applicable contract rates is tabled below.

Table 21-10: Summary of Contract Development Rates

Item	Unit	Rate
Development excavation		
Ramp Development 4,6mW x 5,4mH	m	2,273
Cubby 4,6mW x 5,1mH	m	2,063
FW Drive 4,6mW x 5,1mH	m	2,063
Access X-cut 4,6mW x 5,1mH	m	2,063
Stope X cut Waste 4,5mW x 4,5mH	m	1,577
Stope X-cut Ore 4,5mW x 4,5mH	m	1,577
FAR Drop raise 4,5 diameter with Man way	m	5,724
RAR Drop raise 5,0 diameter	m	1,662
FAR Raisebore or Alimak slash 4,4m diameter With Man way	m	8,227
RAR Raisebore or Alimak slash 5,0m diameter	m	7,387
Ground support		

Item	Unit	Rate
Split set	#	
0,9m bolt 35mm	#	18.00
1,8m bolt 35mm	#	27.81
1,8m bolt 46mm	#	37.00
Mechanical bolts 1,2m Vent raises	#	28.00
Resin bolts	#	
2,4m bolt 20mm	#	40.55
3,0m bolt 20mm	#	50.00
Cable bolts	#	47.00
100mm x 100mm weld mesh 5,6mm	m^2	5.00
Shotcrete unreinforced 25mm thickness 25% estimation	m^2	17.73
Haulage		
One way haulage to Tip Point 0-1,5km haul distance	t.km	0.69
One way haulage to Tip Point 2+km haul distance	t.km	0.60

In addition to these rates, a manpower cost of $12.2 million over the 24-month period, which amounts to $1451 per metre developed, has been included. The total cost per metre for the contractor development is $4270 per metre.

Other Capital Costs

Indirect cost includes an allowance for engineering design and drawings. It is estimated at 12.5 per cent of the engineering bill. Indirect costs are estimated at 4 million.

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 24 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.

21.2. 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. 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 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 3D 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 2 week on and 1 week off rotation cycle.

Drawings for typical accommodation units representing 3 levels of seniority in the operation team 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. Due to the scarcity

of local building resources, a decision was made to price all accommodation, camp, and infrastructure buildings as typical premanufactured insulated steel panel modular units. The market was investigated for suitable and experienced remote accommodation supply and construction companies and an extensive list of prospective vendors were identified from Europe, West Africa, and South Africa.

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. Typical structure drawings and quantities of accommodation units were issued formally to the market via the RFQ process and pricing was obtained for the supply, erection/construction, and operation of a suitable camp. The bids received were then adjudicated and a recommendation made for the inclusion of an appropriate price for inclusion into both the capital cost estimate as well as the operational cost estimate.

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. 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 dry stack tailing storage facility (DSTSF) has been determined to $8.5 million including contingency with an accuracy of +20-15% based on quantities measured by Epoch and rates obtained from De Simone Ltd, as provided by METC.

The selection of a suitable site for the TSF was done, taking into consideration:

the method of deposition

the route to deliver the tailing to the TSF

the prevailing wind direction, and the potential impact of dust from the TSF 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, 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.

The TSF will be constructed progressively over the life of mine. In total, three compartments 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 ($2.6 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 and the starter walls until the design height 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 the TSF will begin. Compartment three will follow and finally the entire TSF will be closed and capped in terms of the environmental requirements. Compartments two and three, together with the capping and closing of the facility are included in the sustaining capital cost as reflected in the financial analysis.

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 132kV 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 450 people

Project insurance

Operational readiness

Fuel for vehicles for the owners and EPCM team on site

Commissioning services, and First Fills

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 (P&G's) Costs

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 costs from the various disciplines for the onsite construction / installation work a consolidated P&G 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 the table below.

Table 21-11: Process plant capital cost summary

	Base Price	Contingency %	Contingency Value	Total Value (carried through)
Supply				
Mechanical	$ 27 940 423	6.2%	$ 1 731 696	$ 29 672 119
Structural Steel	$ 6 888 001	7.5%	$ 516 600	$ 7 404 601
Plate Work	$ 2 376 775	7.5%	$ 178 258	$ 2 555 033
Piping	$ 3 898 509	7.5%	$ 292 388	$ 4 190 897
Electrical	$ 6 946 839	8.6%	$ 600 248	$ 7 547 088
Instrumentation	$ 1 791 665	16.8%	$ 301 146	$ 2 092 812
Valves	$ 486 082	13.7%	$ 66 593	$ 552 676
Logistics	$ 6 098 131	7.5%	$ 457 360	$ 6 555 491
Sub Total - Supply:	**$ 56 426 427**	**7.3%**	**$ 4 144 290**	**$ 60 570 717**
Installation				
SMPP	$ 3 802 478	9.3%	$ 353 630	$ 4 156 108
Electrical	$ 845 591	10.0%	$ 84 559	$ 930 150
Instrumentation	$ 423 565	10.0%	$ 42 356	$ 465 921
Earthworks (Supply & Install)	$ 6 802 785	7.5%	$ 510 209	$ 7 312 993
Civils (Supply & Install)	$ 12 292 655	7.5%	$ 921 949	$ 13 214 605
P&Gs	$ 7 088 722	25.0%	$ 1 772 181	$ 8 860 903
Sub Total - Install:	**$ 31 255 796**	**11.8%**	**$ 3 684 885**	**$ 34 940 680**
Plant Total	**$ 87 682 223**	**8.9%**	**$ 7 829 174**	**$ 95 511 397**
Other Costs				
Owners Costs	$ 16 991 183	5.0%	$ 849 559	$ 17 840 742
EPCM	$ 8 567 249	10.0%	$ 856 725	$ 9 423 974
Spares	$ 1 295 383	6.2%	$ 80 314	$ 1 375 697
Surface Mobile Plant	$ 5 875 711	4.0%	$ 235 028	$ 6 110 740
Infrastructure	$ 12 021 856	7.2%	$ 866 623	$ 12 888 479
Sub Total - Other Costs:	**$ 44 751 381**	**6.5%**	**$ 2 888 249**	**$ 47 639 630**
Total: Plant + Other	**$ 132 433 604**	**8.1%**	**$ 10 717 423**	**$ 143 151 027**

Note 1: TSF Initial Capital and Mining are excluded from the above costs (see *Table 21-1* for Mining and section 21.7.18 for TSF)

Items in the above table with a high contingency warrant discussion:

Supply – Instrumentation and Valves

Despite the accuracy of the BOQ, which was determined from the intelligent P&ID's, prices were not obtained from the market specific for the Dasa Project. Recent data base pricing was used from another project and although detailed and precise, the scoring criteria utilised to determine the contingency resulted in a high contingency

Installation – P&G's

The installation proposals received were accurately detailed and a rates comparison did not highlight any significant outliers, the vendors were (in most disciplines) local contractors and their perceived knowledge and understanding of preliminary and general costs created a measure of doubt. Given the limited number of mining projects in the country, there was some doubt as to whether the contractors had a clear understanding of P&G's.

Currency Exchange Rates

This capital cost estimate is provided in US$. The capital cost estimate has exposure to various currencies. The project accepted a table of fixed rates of exchange on 4 February 2021, with the principal currencies and exchange rates shown in Table 21-12. Any variations in these exchange rates will require adjustment to the final (US$) estimate total.

Table 21-12: Exchange Rates

Exchange	Rate
ZAR to USD	ZAR 15.06 = USD 1.00
EUR to USD	EUR 0.83 = USD 1.00
AUD to USD	AUD 1.32 = USD 1.00
CAD to USD	CAD 1.28 = USD 1.00
CFA (West African Franc) to USD	CFA 548.30 = USD 1.00
INR (Indian Rupees) to USD	INR 72.95 = USD 1.00

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 21-13. 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 21-13: Summary of Mining Operating Cost

Operating Cost	Phase 1 Total $	Cost / t [$ / t]	Cost / lb [$ / lb]
Mining	268 325 246	63.08	5.91
Consumables	102 703 868	24.15	2.26
TMM Consumables	48 351 678	11.37	1.06
Labour	23 487 892	5.52	0.52
Power	93 026 809	21.87	2.05
Other / Contractors	755 000	0.18	0.02
General and Administration	12 074 693	2.84	0.26
Consumables	2 126 813	0.50	0.05
Labour	8 437 880	1.98	0.18
Other / Contractors	1 510 000	0.35	0.03
Total	**280 399 939**	**65.92**	**6.17**

Note: Power costs include pumping costs related to surface ponds

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 $0.1725 per kWh as per the tariff provided by METC.

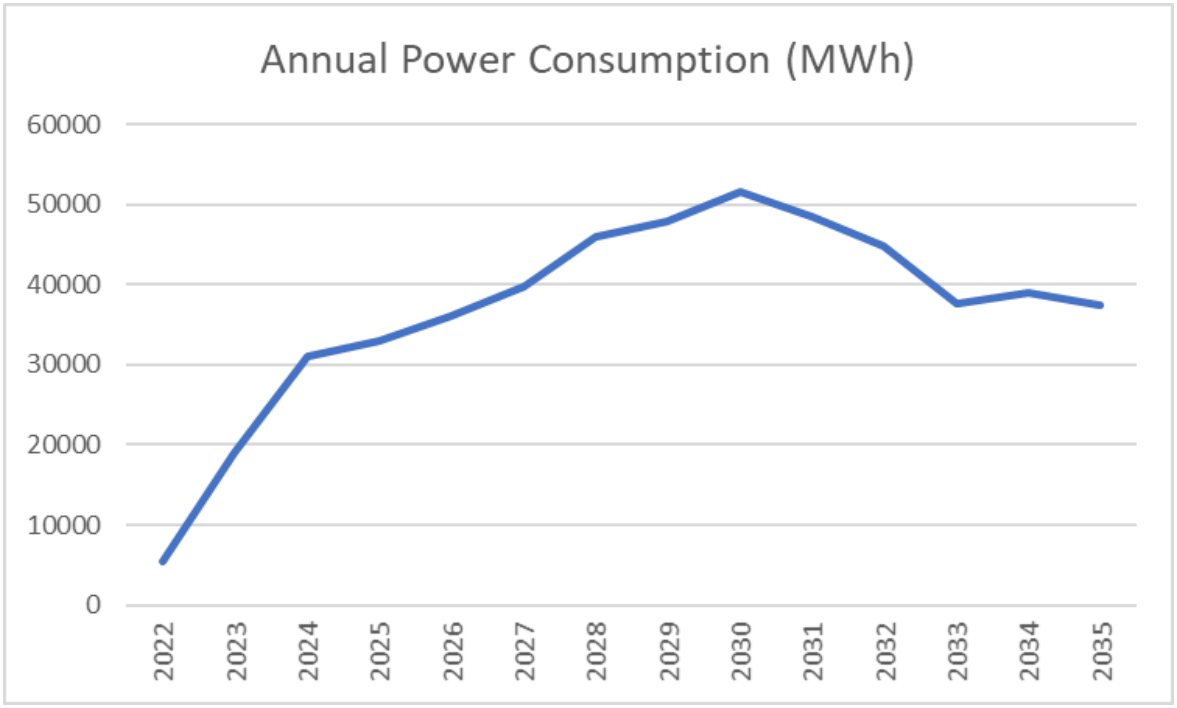


Figure 21-1: Dasa Phase 1 Mine Power Consumption

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 Table 21-14 and Table 21-15. 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 21-14: Cost Structure for Nationals

Designation	Category	Monthly Salary	Annual Salary
Management	1	$ 2 676.00	$ 32 112.00
Middle Management	2	$ 2 007.00	$ 24 084.00
Foreman / Supervisor / Technician	3	$ 1 624.00	$ 19 488.00
Training / Safety	4	$ 1 346.00	$ 16 152.00
Artisans	5	$ 1 171.00	$ 14 052.00
Control Room Controller	6	$ 892.00	$ 10 704.00
Operator	7	$ 830.00	$ 9 960.00
Team Leader / Clerk	8	$ 432.00	$ 5 184.00
Attendant / Eng. Assistant	9	$ 335.00	$ 4 020.00

Table 21-15: 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.

21.4. Processing Plant Operational Costs

The operating cost estimate is based on the following:

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

Grinding media consumption rates have been calculated using the abrasion index as an indication of expected wear

Reagent consumption rates have been estimated based on the metallurgical test work.

The expected electrical consumption and unit rates

Maintenance costs associated with equipment and machinery

Basis of Process Operating Cost

The operating cost estimate for the Process Plant consists of the following:

Power

The power for the project will be provided from the national grid and will be supplemented by a solar PV plant. Grid power in the region is provided by a coal fired power station some 50km from the mine site with the distribution power line running approximately 5km east of the Dasa site. The solar PV plant will be operated on a power purchase agreement (PPA) arrangement. The expected annual power generated by the solar PV plant will be purchased at a lower unit cost than the grid power, effectively reducing the overall annual electricity unit cost from $0.20 / kWh to $0.1725 / kWh.

Reagents

The reagent requirement was developed from the test work (detailed in section 13 of this report). Where test work was not available, benchmarking against currently operating unit technologies was examined. Reagent costs were obtained from vendors as delivered costs to the port of Cotonou in the neighbouring country of Benin. Freight costs from the Port of Cotonou to the Dasa site were obtained from Freight Companies specialising in transport and logistics in the region.

Consumables

Consumables were identified as non-reagent requirements and are related to mainly the crushing, grinding and filtration circuits. The following items have been included under consumables:

Primary Crusher liners

SAG mill liners

SAG mill grinding media

Screen mesh/panels

Belt filter cloths

Consumables costs for these items were obtained from vendors as delivered costs to the Port of Cotonou. Freight costs from the Port of Cotonou to the Dasa site were obtained from Freight Companies specialising in transport and logistics in the region

Maintenance

The process plant annual maintenance costs were derived by applying a factor of 5% of the total installed mechanical equipment cost.

Laboratory Services

The operating cost estimate for the laboratory and assay activities were based on the average number of assays per year, as well as the type of assays, and calculated back to a cost per tonne of ore feed. The number and type of assays to be undertaken include:

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 estimate was determined from benchmarking against similar projects with comparable unit processes (crushing, grinding, leaching, and laboratory). Labour was estimated for administrative staff, maintenance personnel, and plant operations personnel based on appropriate 8-hour and 12-hour shifts per activity or requirement.

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

Operational costs related to the Tailing Storage Facility are included the cash flow model

An organizational roster outlining the labour requirement for the process plant was developed for the estimation of head count required for the plant.

Diesel

The diesel requirement for air heating for drying ore in the SAG mill was calculated based on moisture content of the ore and heating energy and air volume calculations for moisture evaporation. Mobile equipment diesel consumption rates were estimated from Equipment data books. Annual equipment operating hours for mobile equipment were estimated from experience.

Table 21-16: Processing operating cost summary

Category	$/t milled	$/lb U_3O_8 recovered	% of OPEX
Consumable	6.27	0.59	9.8%
Diesel	6.05	0.57	9.4%
Reagents	25.58	2.40	39.9%
Power	15.92	1.49	24.8%
Maintenance	4.46	0.42	6.9%
Laboratory	2.16	0.20	3.4%
Personnel – Process Plant	2.24	0.21	3.5%
Personnel - Maintenance	1.45	0.14	2.3%
TOTAL	64.13	6.01	100%

These costs, excluding the operating costs associated with the deposition, placement and compaction of the tailings are represented in the following pie chart Figure 21-2.

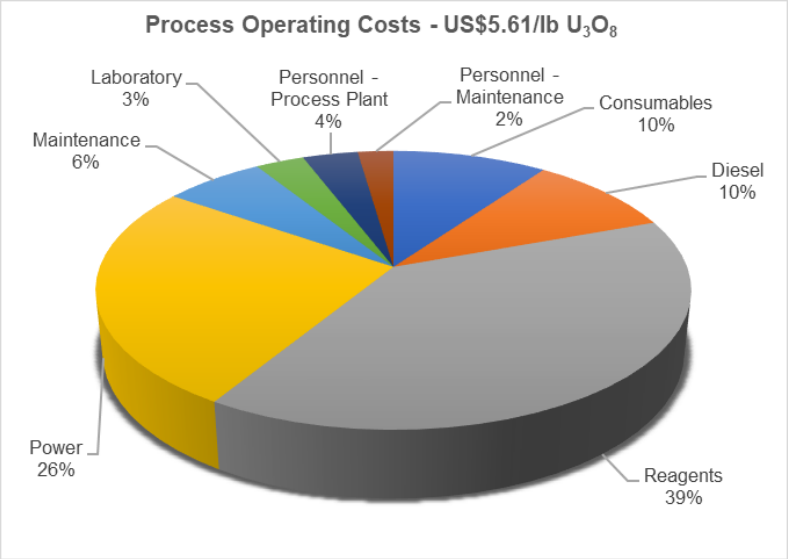


Figure 21-2: Plant operating cost breakdown

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 on the basis of $1 per lb of U_3O_8.

Annualized Dasa and Niamey sites G&A costs are estimated at $ 26.25 per tonne of ore, or $2.46 per lb of U3O8. The full G&A costs ($36.93) are summarised in Table 21-17.

Table 21-17: Breakdown of annualized G&A costs (thousands)

Category	$ 000’	$/t milled
Management Salaries	2 392	6.92
Vehicles – Fuel & Maintenance	325	0.94
Communications	502	1.45
Insurances	425	1.23
Administration Expenses	199	0.57
Medical Costs	98	0.29
Camp Costs	3 149	9.11

Category	$ 000’	$/t milled
Property Costs	801	2.32
Travel	118	0.34
Community Relations	517	1.50
Land Lease	232	0.67
Marketing & Transportation	3 691	10.67
Other	316	0.91
TOTAL	**12 770**	**36.93**

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 below

Table 22-1: Key metrics for the Phase 1 Dasa project.

Project economics	**Unit**	**Value**
NPV @ 8 % after tax	$M	$157
IRR after tax	%	22.7%
Payback period from Jan 2022	years	5
Payback period from start-up	years	3
Cash flow (net of capex)	$M	$677
Free cash flow	$M	$332
Operating costs		
Cash cost	$/lb U3O8	18.91
All-in sustaining cost	$/lb U3O8	21.93
Capital costs		
Initial capital costs	$M	$208
Sustaining capital costs	$M	$137

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 US dollars.
- All values are in 2021 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 \$35/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 \$35 /lb U_3O_8.

The mine production has been forecast to begin 17 months from the start, while the first product from the plant is forecast to begin 20 months from the start. The plant recovery is estimated to be 94.15% at steady state. The plant is assumed to have an operating capacity of 1 000 t/d of run-of-mine feed. Excess mined material is stockpiled for later processing.

Mining royalty

The Government of Niger collects a mining royalty that is proportional to operating profitability. The royalty rate is assessed on a sliding scale based on the operating profitability of the mine and plant, and then applied to revenues. The operating profitability is determined as a ratio of EBIT to revenues, where EBIT refers to earnings before interest and taxation. The royalty is deductible for the determination of corporate income tax.

The sliding scale is as follows:

If operating profitability < 20%	mining royalty rate is 5.5%
If operating profitability > 20% and < 50%	mining royalty rate is 9.0%
If operating profitability > 50%	mining royalty rate is 12.0%

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 $65.9 /t milled and the processing costs are $64.2/t milled.

Corporation 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 $ 47.5 million is included in the calculation.

Niger has a value-added tax system with a rate of 19%. The project is exempt from VAT until commencement of production. Once production commences, the VAT rate is set at 0%.

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. Only changes in working capital are required 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. 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 – project	61,889
Processing – project	143,151
TSF – project	2,597
Total Initial Capital Expenditure	**207,637**
Sustaining and closure costs	137,347
Total Initial plus Sustaining Capital expenditure	**344,985**

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 $ 157 M, and IRR is 22.7% for the base case estimates.

Table 22-3: Forecast of the project economics over the life-of-mine.

Item	LOM Total, $M
Revenue	1 589
Mining royalties	148
Operating costs	710
Operating income	**731**
Taxation	54
Capital costs	345
Free cash flow	**332**

Note: values may not add up due to rounding.

Table 22-4: Financial indicators over the life-of-mine.

Item	Unit	Value
NPV_8after tax	$M	$157
IRR after tax	%	22.7%
Payback period	years	5
Cash flow (net of capex)	$M	$677
Free cash flow	$M	$332
Operating costs		
Cash cost	$/lb U_3O_8	18.91
All-in sustaining cost	$/lb U_3O_8	21.93
Capital costs		
Initial capital costs	$M	$208
Sustaining capital costs	$M	$137

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.

Free cash flow and payback period

The free cash flow is calculated as revenue less operating costs, capital expenditure, taxation, royalties, and changes in working capital. The annual free cash flow and cumulative free cash flow are shown in Figure 22-1. The cumulative free cash becomes positive 5 years after the start of the Project. The Payback period is 3 years after the construction of the project.

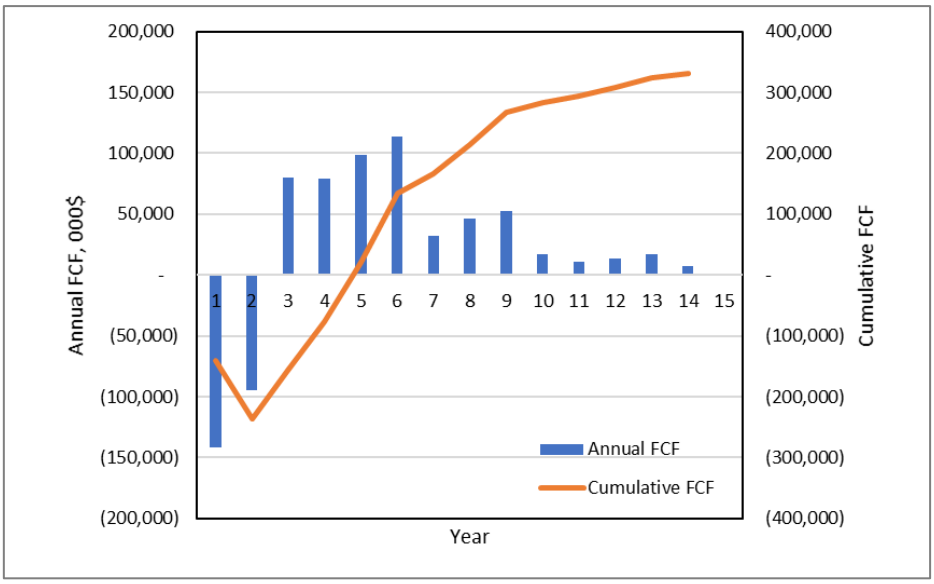


Figure 22-1: Annual free cash flow profile for the Dasa Phase 1 project.

Composition of cash flow

The free cash flow and the composition of the monthly free cash are shown in Figure 22-2 and Figure 22-3.

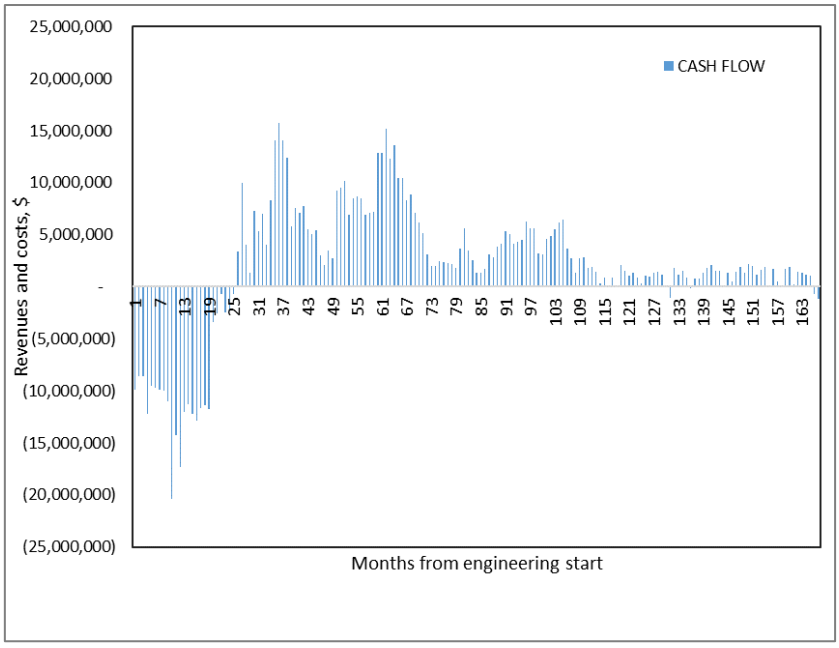


Figure 22-2: Monthly free cash profile for the Dasa Phase 1 project

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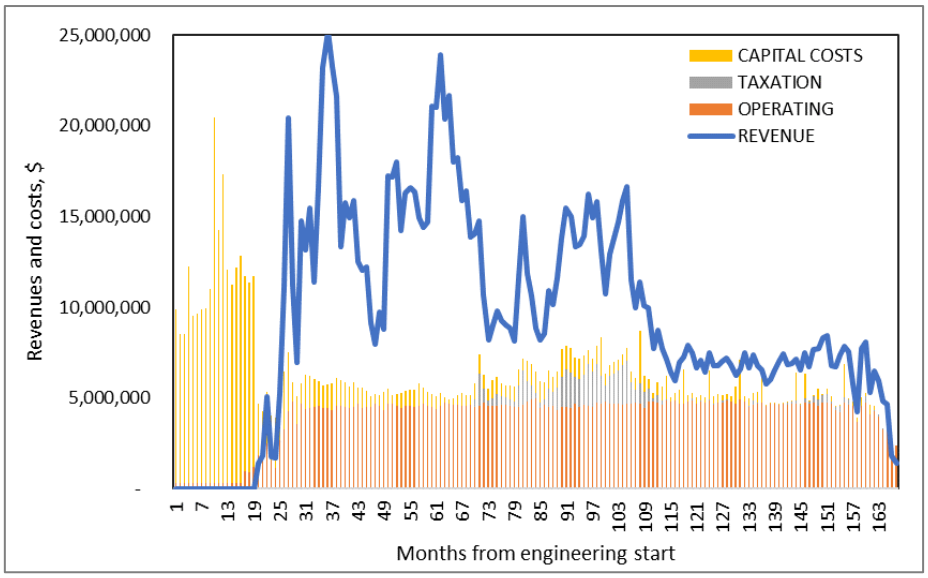


Figure 22-3: Monthly contributors to the free cash profile for the Dasa Phase 1 project

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 22-4, Figure 22-5, and Table 22-5.

The model is most sensitive to the price of U_3O_8.

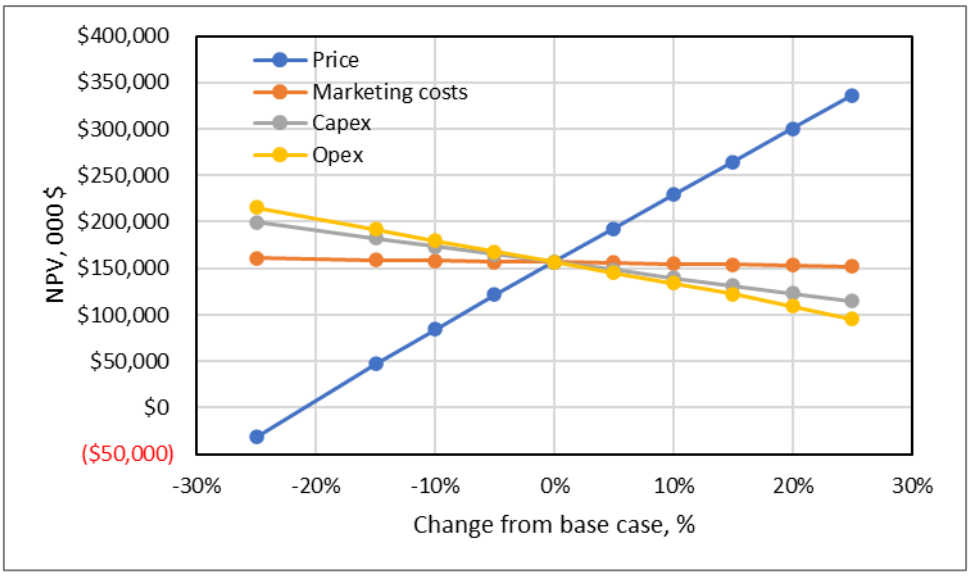


Figure 22-4: Sensitivity of the NPV8 to changes from the base case in the input costs for the financial model for the Dasa Phase 1 project

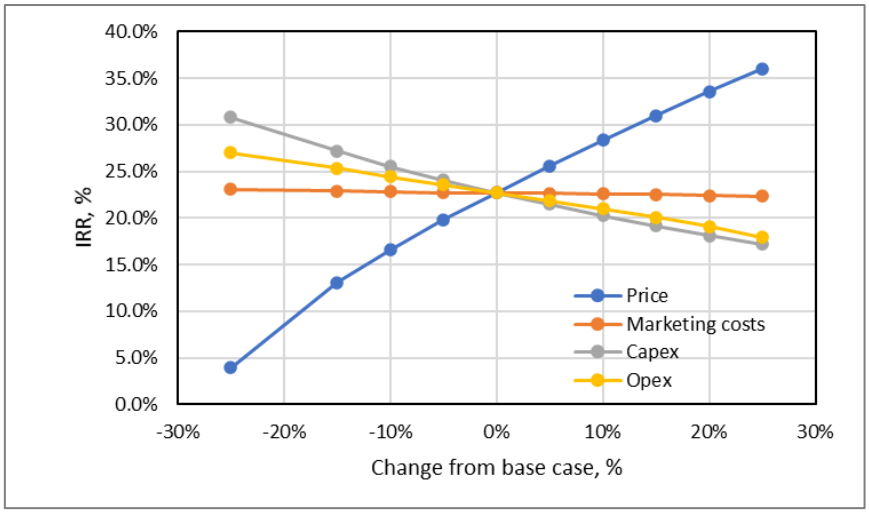


Figure 22-5 - Sensitivity of the IRR to changes from the base case in the input costs for the financial model for the Dasa project.

Table 22-5 Economic Sensitivity to Varying Price of U_3O_8

U_3O_8 price (per pound)	**$35**	**$40**	**$50**	**$60**
Before-tax NPV8	$187M	$309M	$556M	$804M
After-tax NPV8	$157M	$259M	$468M	$676M
After-tax IRR	22.7%	30.6%	44.6%	57.2%

Monte Carlo analysis

The project is most sensitive to the price of U_3O_8. This parameter was used as a single parameter input for the Monte Carlo simulation to determine the robustness of the project. Standard deviation of 10% about the mean was used. The results are shown in Figure 22-6. The low probability of negative NPV_8 suggests that there is a degree of robustness to this project.

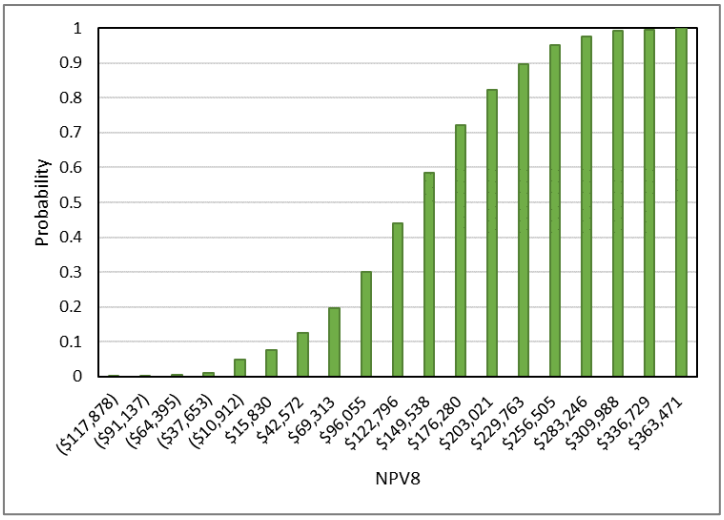


Figure 22-6: Probability function for the NPV8 for a price variation of 10% about the mean for the full life of mine of the Dasa Phase 1 project.

The probability of a negative NPV_8 is 4.7%. Note that this analysis assume that the price will change at the beginning of the project and remaining at the value for the duration of the project.

Summary of economic model

The economic model is summarized in Table 22-6.

Table 22-6 Summary of economic forecast for the Dasa project.

Year			1	2	3	4	5	6	7	8	9	10	11	12	13	14
REVENUE																
Mine production to ROM stockpile	kt	4,254	-	58	326	366	373	374	373	369	365	368	365	361	334	222
Feed grade	ppm	*5,184*	-	*2,827*	*7,665*	*6,323*	*7,241*	*7,698*	*4,475*	*5,957*	*5,903*	*3,384*	*3,110*	*3,042*	*3,403*	*3,275*
Plant throughput	kt	4,254	-	*57.7*	*323.8*	*365.0*	*365.0*	*365.0*	*366.0*	*365.0*	*365.0*	*365.0*	*366.0*	*365.0*	*362.7*	*222.4*
ROM Stockpile (Monthly Average)	kt	*33*	-	*4.22*	*1.16*	*3.15*	*9.19*	*16.61*	*24.75*	*30.16*	*32.51*	*32.17*	*32.53*	*32.73*	*20.36*	*0.21*
Throughput	000 lb U3O8	48,611	-	*544*	*5,508*	*5,110*	*5,759*	*6,340*	*3,651*	*4,571*	*4,899*	*2,837*	*2,511*	*2,464*	*2,707*	*1,711*
Recovery	%	*93.41%*	-	*26.09%*	*89.93%*	*94.15%*	*94.15%*	*94.15%*	*94.15%*	*94.15%*	*94.15%*	*94.15%*	*94.15%*	*94.15%*	*94.15%*	*86.30%*
Production	000 lb U3O8	45,406	-	337	4,999	4,811	5,422	5,969	3,438	4,303	4,612	2,671	2,364	2,319	2,549	1,611
Revenue	000 $	1,589,216	-	11,806	174,971	168,373	189,774	208,904	120,322	150,621	161,419	93,500	82,757	81,182	89,216	56,370
Charges	$	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Net Revenue	**000 $**	**1,589,216**	**-**	**11,806**	**174,971**	**168,373**	**189,774**	**208,904**	**120,322**	**150,621**	**161,419**	**93,500**	**82,757**	**81,182**	**89,216**	**56,370**
MINING ROYALTIES																
Mining royalty	**000 $**	**148,318**	**-**	**631**	**17,822**	**17,113**	**21,149**	**23,638**	**10,401**	**16,117**	**16,788**	**5,977**	**4,422**	**4,337**	**6,375**	**3,549**
OPERATING COSTS																
Mining	000 $	275,629	-	5,735	20,313	21,876	22,316	22,672	22,476	21,950	23,714	24,527	24,772	23,500	23,230	18,548
Processing	000 $	265,597	-	3,601	20,221	22,791	22,791	22,791	22,853	22,791	22,791	22,791	22,853	22,791	22,647	13,886
Evaporation pond costs	000 $	4,771	-	65	363	409	409	409	410	409	409	409	410	409	407	249
TSF	000 $	7,194	-	213	512	512	512	512	1,388	512	512	512	512	512	512	470
G&A (Dasa)	000 $	79,766	1,735	1,735	6,402	6,402	6,402	6,402	6,402	6,402	6,402	6,402	6,402	6,402	6,402	5,869
G&A (Naimey)	1 $	31,912	1,810	1,810	2,374	2,374	2,374	2,374	2,374	2,374	2,374	2,374	2,374	2,374	2,374	2,176
Marketing and Transportation	000 $	45,406	-	337	4,999	4,811	5,422	5,969	3,438	4,303	4,612	2,671	2,364	2,319	2,549	1,611
Total costs including royalty	**000 $**	**858,593**	**3,546**	**14,129**	**73,007**	**76,289**	**81,376**	**84,767**	**69,743**	**74,860**	**77,602**	**65,665**	**64,111**	**62,645**	**64,496**	**46,359**
COST METRICS																
Cash cost	000 $	18.91	-	41.88	14.60	15.86	15.01	14.20	20.29	17.40	16.83	24.58	27.11	27.01	25.30	28.78
All-in-sustaining cost	000 $	21.93	-	108.45	19.03	18.54	16.86	15.45	23.39	20.42	19.40	27.77	30.39	28.88	27.02	29.95
EBITDA																
EBITDA	000 $	730,623	(3,546)	(2,322)	101,964	92,085	108,398	124,137	50,579	75,762	83,816	27,835	18,647	18,536	24,720	10,011
TAXATION																
Income Tax	**000 $**	**53,624**	**-**	**-**	**-**	**-**	**-**	**3,004**	**8,113**	**16,147**	**18,980**	**2,599**	**327**	**749**	**3,103**	**602**
CAPITAL COSTS																
Mining - project	000 $	61,889	44,868	17,021	-	-	-	-	-	-	-	-	-	-	-	-
Processing - project	000 $	143,151	90,411	52,740	-	-	-	-	-	-	-	-	-	-	-	-
TSF - project	000 $	2,597	2,597	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining and closure costs	000 $	137,347	-	22,453	22,118	12,903	10,016	7,435	10,665	13,018	11,876	8,529	7,740	4,337	4,372	1,885
Total Capital expenditure	**000 $**	**344,985**	**137,877**	**92,213**	**22,118**	**12,903**	**10,016**	**7,435**	**10,665**	**13,018**	**11,876**	**8,529**	**7,740**	**4,337**	**4,372**	**1,885**
FREE CASH FLOW																
Cash flow	000 $	676,999	(3,546)	(2,322)	101,964	92,085	108,398	121,133	42,467	59,615	64,836	25,236	18,320	17,787	21,616	9,409
Free cash flow	000 $	332,014	(141,423)	(94,536)	79,847	79,182	98,382	113,698	31,802	46,596	52,960	16,707	10,580	13,450	17,245	7,524
Cumulative FCF	000 $		(141,423)	(235,958)	(156,112)	(76,930)	21,452	135,150	166,952	213,548	266,508	283,215	293,795	307,245	324,489	332,014
METRICS																
NPV @ 8%	000 $	$156,535														
IRR	%	22.7%														

23. Adjacent Properties

There are no third-party properties currently in production adjacent to Dasa.

The Imouraren uranium project is located approximately 40km 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 operating license for 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 (2019a) reports that the deposit area holds total (100% share) Mineral Reserves of 306,048,000 tonnes of Probable ore grading 0.07% U (700 ppm U) as at 31 December 2018.

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 km² (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 km² (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). At Imouraren, the mineralization, located between approximately 100 m and 150 m below the surface, will be extracted by open pit operation and treated by heap leaching in an acidic environment, a first for a deposit of this size. The project has a proposed annual production capacity of 5,000 tonnes and lifespan of 35 years. However, since 2015, production start-up work has been suspended and the site has been placed in care and maintenance, pending more favourable market conditions (Orano, 2019b).

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 4.254 million tonnes of mineralised material from a sub-vertical section of the deposit on the flank of the graben, from an average depth of 300m below surface. The average grade of 5184 ppm represents the ability to produce 45.406 million lb U_3O_8 for an expected revenue of $1.589 billion at a price of $35/lb U_3O_8. Value opportunities exist in extending the mine-life beyond an initial 12 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.

An agreement was signed in July 2017 (Memorandum of Understanding “MOU”) between Global Atomic and Orano Mining, to supply a minimum 100,000 tonnes of uranium-bearing rock per annum to Orano’s operations in Arlit, approximately 105 kilometres north of the Dasa Project. These discussions are ongoing between the two companies and a successful conclusion would result in Global Atomic having a revenue stream to support the project whilst the processing plant is built.

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 remains unchanged from the PEA that was concluded in May 2020 and undertaken by CSA Global. 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 previously reported in a NI 43-101 Technical Report with an effective date of 1 June 2019 (CSA Global, 2019) 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 1 June 2019.

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 1 June 2019

Mineral Resource category	Tonnes (Mt)	eU_3O_8 (ppm)	Contained metal (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 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, (CSA Global).

The Effective Date of the MRE is 1 June 2019.

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

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

It can be reasonably expected that

The majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration

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 14 years (including the mine development phase)

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 $35/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 testwork 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:

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

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

25.4. Economic Assessment

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 $ 157 M, and IRR is 22.7% for the base case estimates.

Table 25-2: Forecast of the project economics over the life-of-mine.

Item	LOM Total, $M
Revenue	1 589
Mining royalties	148
Operating costs	710
Operating income	**731**
Taxation	54
Capital costs	345
Free cash flow	**332**

Note: values may not add up due to rounding.

Table 25-3: Financial indicators over the life-of-mine.

Item	Unit	Value
NPV_8after tax	$M	$157
IRR after tax	%	22.7%
Payback period	Years	5
Cash flow (net of capex)	$M	$677
Free cash flow	$M	$332
Operating costs		
Cash cost	$/lb U_3O_8	18.91
All-in sustaining cost	$/lb U_3O_8	21.93
Capital costs		
Initial capital costs	$M	$208
Sustaining capital costs	$M	$137

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 $35.00 / 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 25-3 and Figure 25-4. The model is most sensitive to the price of U_3O_8.

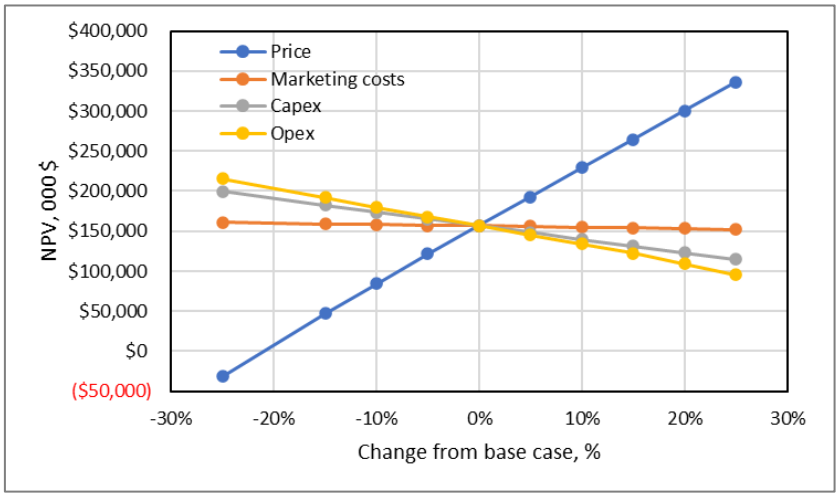


Figure 25-3: Sensitivity of the NPV8 to changes from the base case in the input costs for the financial model for the Dasa Phase 1 project

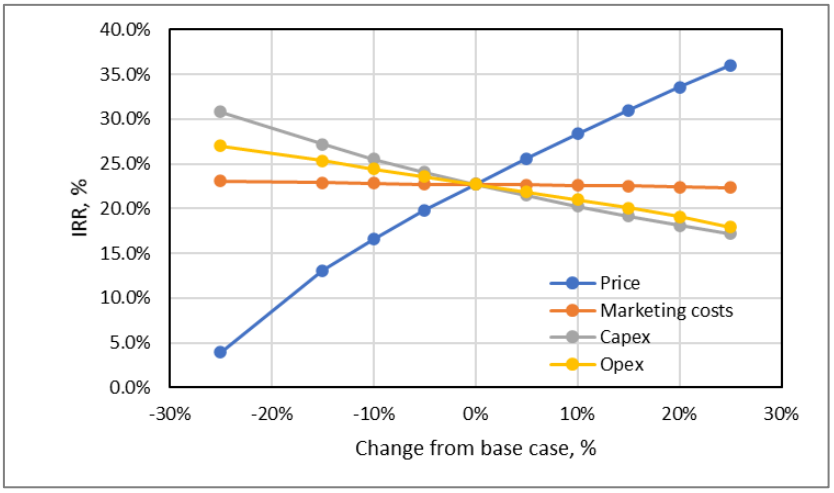


Figure 25-4: Sensitivity of the IRR to changes from the base case in the input costs for the financial model for the Dasa Phase 1 project

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 are 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. The solid liquid separation processes in the final process of uranium precipitation could be further refined, to reduce equipment sizing and capital costs, through additional test work, however the process incorporated into the feasibility study is well known and understood.

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 onsite 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

Economic Outcomes

The project is most sensitive to the price of U_3O_8. This parameter was used as a single parameter input for the Monte Carlo simulation to determine the robustness of the project. Standard deviation of 10% about the mean was used. The results are shown in Figure 25-6. The low probability of negative NPV_8 suggests that there is a degree of robustness to this project.

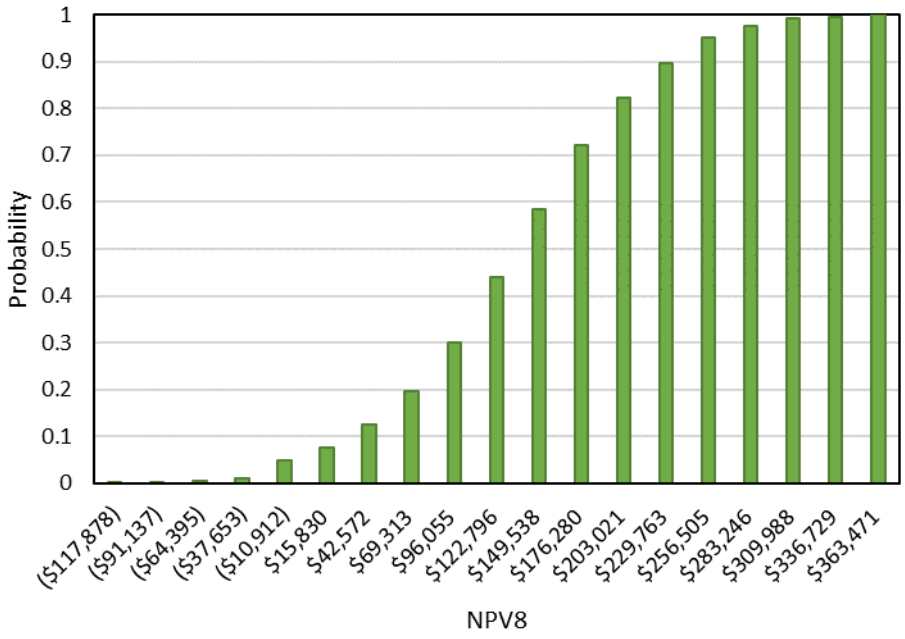


Figure 25-1: Probability function for the NPV8 for a price variation of 10% about the mean for the full life of mine of the Dasa Phase 1 project.

The probability of a negative NPV_8 is 4.7%. Note that this analysis assume that the price will change at the beginning of the project and remaining at the value for the duration of the project.

A sensitivity analysis at varying U3O8 prices was conducted and the results are presented here:

Uranium price (per pound)	$35	$40	$50	$60
Before-tax NPV8	M$187	M$309	M$556	M$804
After-tax NPV8	M$157	M$259	M$468	M$676
After-tax IRR	22.7%	30.6%	44.6%	57.2%

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 mine 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 testwork 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 testwork 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 testwork completed.

Equipment has been sized relative to testwork 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

To enhance the work done in the feasibility study the following activities should be undertaken:

- additional test work to optimise the solid liquid separation in the uranium precipitation process
- further understand the optimum means of supplying the process plant with sulphur for the generation of sulphuric acid

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).

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