TETRA TECH

Feasibility Study, NI 43-101 Technical Report, for PLS Property

PRESENTED TO:

Fission Uranium Corp.

EFFECTIVE DATE: 17 JANUARY 2023

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ACRONYMS AND ABBREVIATIONS

Acronym	Definition
3D	Three-Dimensional
AAS	Atomic Absorption Spectrophotometry
A_i	Abrasion Index
ARD	Acid Rock Drainage
ASA	Aquatic Study Areas
BGC	BGC Engineering Inc.
BNDN	Birch Narrows Dene Nation
BRDN	Buffalo River Dene Nation
BTS	Brazilian Tensile Strength
BWI_{BM}	Ball Mill Bond Work Index
CaCl	Calcium Chloride
CAF	Cemented Aggregate Fill
Cameco	Cameco Corp.
CANMET	Canadian Centre for Mineral and Energy Technology
CanOxy	Canadian Occidental Petroleum Ltd.
CCA	Capital Cost Allowance
CCD	Counter Current Decantation
CCRMP	Canadian Certified Reference Materials Project
CCTV	Closed-Circuit Television
CDA	Canadian Dam Association
CDE	Canadian Development Expenses
CEE	Canadian Exploration Expenses
CHF	Cemented Hydraulic Fill
CHG	Carbonaceous High Grade
C_i	CEET Crusher Index
CIL	Carbon-in-Leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CMP	Caribou Mitigation Plan
CNSC	Canadian Nuclear Safety Commission
COCs	Contaminants of Concern
COG	Cut-off Grade
COSEWIC	Committee on the Status of Endangered Wildlife in Canada
CRDN	Clearwater River Dene Nation

Acronym	Definition
CRF	Cemented Rockfill
CRM	Certified Reference Materials
CSM	Conceptual Site Model
CV	Coefficient of Variation
D/F	Drift and Fill
DCS	Distributed Control System
DD	Diamond Core Drilling
Discovery	Discovery Int'l Geophysics Inc.
E	East
EA	Environmental Assessment
EAA	Environmental Assessment Act
EAS	Exhaust Air Shaft
EASB	Environmental Assessment and Stewardship Branch
ECCC	Environment and Climate Change Canada
EIA	Environmental Impact Assessment
EIC	Electret Ion Chamber
ELOS	Equivalent Linear Overbreak Slough
EM	Electromagnetic
EMS	Environmental Management Systems
ENE-WSW	East-Northeast-West-Southwest
ENV	Saskatchewan Ministry of Environment
EPB	Environmental Protection Branch
EPCM	Engineering, Procurement, and Construction Management
EPR	Environmental Protection Review
FAS	Fresh Air Shaft
FCU	Fission Uranium Corporation
FOS	Factor of Safety
FRS	Fibre-Reinforced Shotcrete
FS	Feasibility Study
FSWT	First-Stage Water Treatment
FW	Footwall
FWD	Footwall Drive
G&A	General and Administration
GAP	Gap Geophysics Australia Pty Limited
GHG	Greenhouse Gases

Acronym	Definition
GIS	Geographical Information System
GroundSAM	Ground Receiver Platform
GSC	Geological Survey of Canada
HAZOP	Hazard and Operability Analysis
HDPE	High-Density Polyethylene
HeliSAM	Helicopter Receiver Platform
HG	High Grade
HLEM	Horizontal Loop EM
HR	Hydraulic Radii
HRIA	Heritage Resources Impact Assessment
HSES	Health, Safety, Environmental, and Security
HSU	Hydrostratigraphic unit
HW	Headwall
HWD	Hanging Wall Drive
I/O	Input/output
IA	Impact Assessment
IAA	Impact Assessment Act
IAEA	International Atomic Energy Agency
IBA	Impact Benefit Agreements
ICP-MS	Inductively Coupled Plasma Mass Spectrometry
ICP-OES	Inductively Coupled Plasma Optical Emission Spectrometry
ID	Inner Diameter
ID^3	Inverse Distance Cubed
IDS	International Directional Services
IEA	International Energy Agency
IRR	Internal Rate of Return
ISO/IEC	International Organization for Standardization / International Electrotechnical Commission
ISP	Intermediate Settling Pond
ITH	In-the-Hole
L	Lower
LAN	Local Area Network
LG	Low Grade
LHD	Load Haul Dump
LiDAR	Light Detection and Ranging

Acronym	Definition
LNG	Liquefied Natural Gas
LOM	Life of mine
LSA	Local Study Area
M	Middle
MAC	Mining Association of Canada
MARS	Mineral Administration Registry System
MCC	Motor Control Centres
MDMER	Metal and Diamond Mining Effluent Regulations
Melis	Melis Engineering Inc.
MG	Medium Grade
MgO	Magnesium Oxide
MHI	Ministry of Highways and Infrastructure
MMC	Magnetometric Conductivity
MRS	Mesh-Reinforced Shotcrete
MS	Mixer Settlers
MSC	Mineral Services Canada Inc
MSDS	Material Safety Data Sheets
MSO	Mineable Shape Optimizer
MSZ	Main Shear Zone
MSZ	Mineralized Shear Zone
MZ	Main Zone
NAG	Non-Acid Generating
NGI	Newmans Geotechnique Inc
NN	Nearest Neighbour
Northspan	Northspan Explorations Ltd.
NPV	Net Present Value
NSCA	Nuclear Safety and Control Act
OIS	Operator Interface Stations
P/O	Pregnant Leach Solution to Ore Ratio
PAG	Potential Acid Generating
PC	Personal Computer
PDC	Preliminary Decommissioning Cost
PDP	Preliminary Decommissioning Plan
PEA	Preliminary Economic Assessment
PERA	Preliminary Environmental Risk Assessment

Acronym	Definition
PFS	Pre-Feasibility Study
PGA	Peak Ground Acceleration
PIMA	Portable Infrared Mineral Analyzer
PLC	Programmable Logic Controllers
PLS	Patterson Lake South
PMP	Probable Maximum Precipitation
PTP-MAL	Proficiency Testing Program for Mineral Analysis Laboratories
QA/QC	Quality Assurance/Quality Control
QFBG-GN	Quartz-Feldspar-Biotite-Garnet Gneiss
QP	Qualified Person
RadonEx	RadonEx Ltd.
RBC	Rotating Biological Contactors
RC	Reverse Circulation
REE	Rare Earth Elements
RES	Remote Exploration Services (Pty) Ltd
RnV	Radon Flux Values
ROM	Run of Mine
RPA	Roscoe Postle Associates Inc.
RPEEE	Reasonable Prospects for Eventual Economic Extraction
RQD	Rock Quality Designation
RRS	Reinforced Ribs of Shotcrete
RSA	Regional Study Area
RTK	Real Time Kinematic
SAM	Sub-Audio Magnetics
SARA	Species At Risk Act
SBL	Soil-Bentonite Liner
SD	Standard Deviation
SEQG	Saskatchewan Environmental Quality Guidelines
SFGG	Sheared Fine-Grained Granitoid
SGS	SGS Canada Inc. – Mineral Services
SIR	Site Investigation Report
SKCDC	Saskatchewan Conservation Data Centre
SLR	SLR International Corporation
SOCC	Species of Conservation Concern
SP	Self-Potential

Acronym	Definition
SPI	Special Projects Inc.
SPI	SAG Power Index
SRC	Saskatchewan Research Council
SRCP	Steel Reinforced Composite Pipe
SSAG	Single-Stage Semi-Autogenous
SSWT	Second-Stage Water Treatment
SX	Solvent Extraction
TDEM	Time-Domain EM
TDH	Total Dynamic Head
TFEM	Total Field Electromagnetic
TMF	Tailings Management Facility
TOR	Terms of Reference
TSS	Total Suspended Solids
TSWT	Third-Stage Water Treatment
U	Upper
UCS	Unconfined Compressive Rock Strength
UTM	Universal Transverse Mercator
UxC	UxC LLC
VOD	Ventilation-on-Demand
VSP	Vertical Seismic Profiling
VWP	Vibrating Wire Piezometer
W	West
WAN	Wide Area Network
WHMIS	Workplace Hazardous Materials Information Systems
Wood	Wood PLC
WRMF	Waste Rock Management Facility
WSP	WSP Global Inc.
WTP	Water Treatment Plant

UNITS OF MEASURE

above mean sea level	amsl
acre	ac
ampere	A
annum (year)	a
bank cubic metres	bm^3
bags	bgs
billion	B
billion tonnes	Bt
billion years ago	Ga
British thermal unit	BTU
centimetre	cm
cubic centimetre	cm^3
cubic feet per minute	cfm
cubic feet per second	ft^3/s
cubic foot	ft^3
cubic inch	in^3
cubic metre	m^3
cubic yard	yd^3
Curie	Ci
Coefficients of Variation	CVs
day	d
days per week	d/wk
days per year (annum)	d/a
dead weight tonnes	DWT
decibel adjusted	dBa
decibel	dB
degree	°
degrees Celsius	°C
diameter	ø
dollar (American)	USD$
dollar (Canadian)	C$
dry metric ton	dmt
foot	ft
gallon	gal
gallons per minute (US)	gpm
gauge	ga
gigajoule	GJ
gigapascal	GPa
gigawatt	GW
gram	g

grams per litre	g/L
grams per tonne	g/t
greater than	>
hectare (10,000 m^2)	ha
hertz	Hz
horsepower	hp
hour	h
hours per day	h/d
hours per week	h/wk
hours per year	h/a
inch	"
kilo (thousand)	k
kilogram	kg
kilograms per cubic metre	kg/m^3
kilograms per hour	kg/h
kilograms per square metre	kg/m^2
kilometre	km
kilometres per hour	km/h
kilopascal	kPa
kilotonne	kt
kilovolt	kV
kilovolt-ampere	kVA
kilovolts	kV
kilowatt	kW
kilowatt hour	kWh
kilowatt hours per tonne (metric ton)	kWh/t
kilowatt hours per year	kWh/a
less than	<
litre	L
litres per minute	L/m
megabytes per second	Mb/s
megapascal	MPa
megavolt-ampere	MVA
megawatt	MW
metre	m
metres above sea level	masl
metres Baltic sea level	mbsl
metres per minute	m/min
metres per second	m/s
metric ton (tonne)	t
microns	µm

milligram	mg
milligrams per litre	mg/L
millilitre	mL
millimetre	mm
million	M
million bank cubic metres	Mbm^3
million bank cubic metres per annum	Mbm^3/a
million pounds	Mlb
million tonnes	Mt
minute (plane angle)	'
minute (time)	min
month	mo
Neutron	N
ounce	oz
pascal	Pa
pico	p
centipoise	mPa·s
parts per million	ppm
parts per billion	ppb
percent	%
pound(s)	lb
pounds per square inch	psi
revolutions per minute	rpm
second (plane angle)	"
second (time)	s
specific gravity	SG
square centimetre	cm^2
square foot	ft^2
square inch	in^2
square kilometre	km^2
square metre	m^2
twenty-foot equivalent unit	TEU
thousand tonnes	kt
tonne (1,000 kg)	t
tonnes per day	t/d
tonnes per hour	t/h
tonnes per year	t/a
tonnes seconds per hour metre cubed	ts/hm^3
volt	V
week	wk
weight/weight	w/w

wet metric ton .... wmt
year (annum) .... a

1.0 SUMMARY

Fission Uranium Corporation (TSE:FCU) commissioned Tetra Tech Canada Inc. (Tetra Tech) to complete this Feasibility Study (FS) with the assistance of specialist consultants for the PLS Project (the Project), located in northern Saskatchewan, following the NI 43-101 Standards of Disclosure for Mineral Projects. FCU is a Canada-based resource company specializing in the strategic exploration and development of the PLS Property (the Property). The consultants commissioned to complete the FS are presented in Table 1-1.

Table 1-1: List of FS Consultants

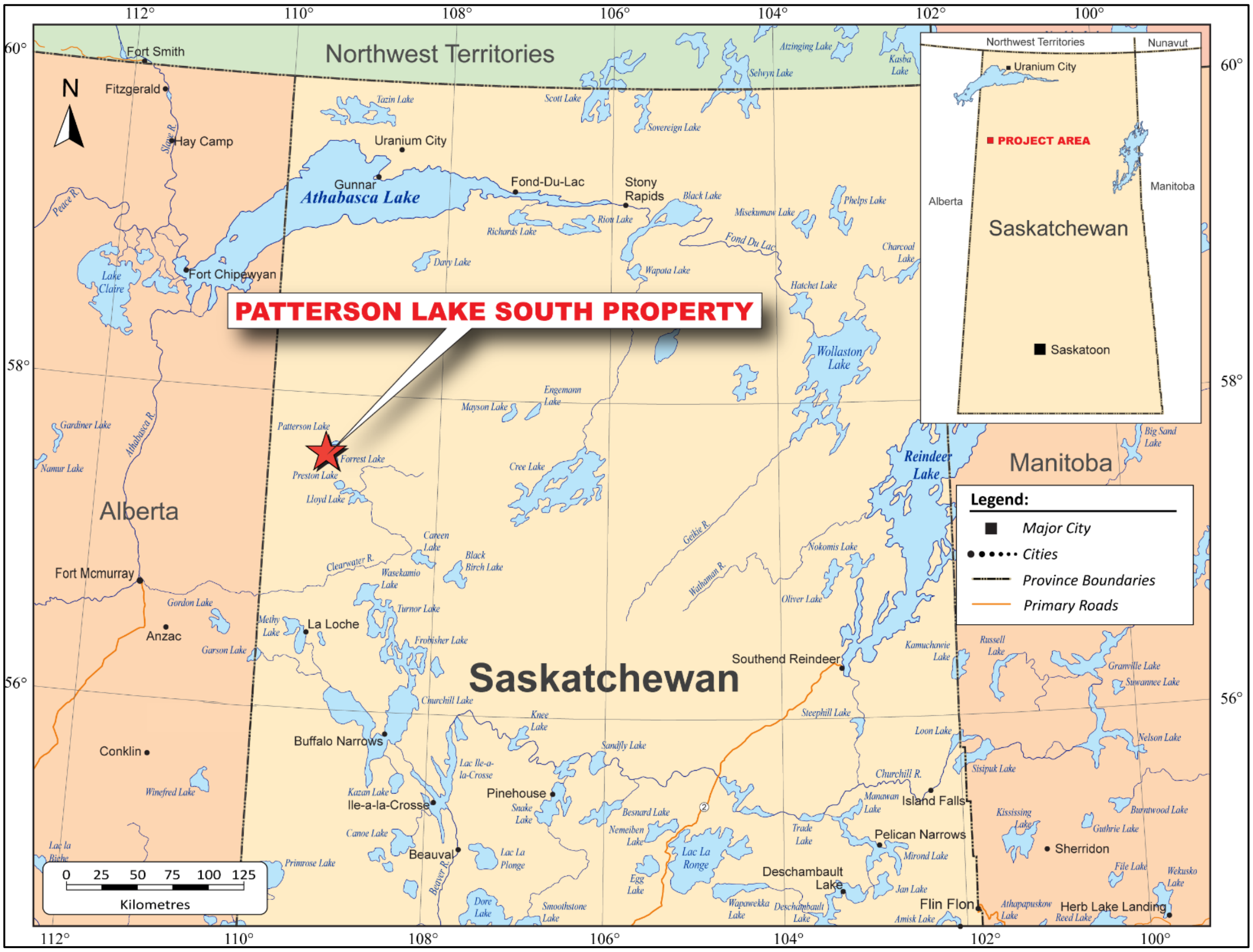
Consultant	FS Components
Tetra Tech Canada Inc.	Overall project management, mineral processing and metallurgical testing, recovery methods, project infrastructure (overall site layout, ancillary infrastructure, and buildings including site roads), marketing studies, summary of initial and sustaining capital and operating cost estimates, economic analysis, project execution plan and overall FS Technical Report compilation
SLR Consulting (Canada) Ltd. (SLR)	Project description and location, accessibility, history, geological setting, deposit types, exploration, drilling, data verification, mineral resource estimate, adjacent properties
BGC Engineering Inc. (BGC)	Waste Rock Stockpile slope design, underground and surface infrastructure geotechnical assessment, hydrogeology
Mining Plus Canada Consulting Ltd. (Mining Plus)	Mineral reserve estimate, waste rock management, mining methods, mining initial, and sustaining capital and operating cost estimates
Clifton Engineering Group Ltd. (Clifton)	Tailings management facility (TMF), environmental, permitting, and socio-economics

Unless otherwise noted, all currencies are expressed in Canadian dollars (C$ or $) in this Technical Report.

1.1 Project Description and Location

1.1.1 Location

The PLS Property is located in northern Saskatchewan, approximately 550 km north-northwest of Prince Albert by air and 157 km north of La Loche by road, as illustrated in Figure 1-1. The geographic coordinates for the approximate centre of the PLS Property are 57°37’ N latitude and 109°22’ W longitude which corresponds to the Universal Transverse Mercator (UTM) geographic coordinates of 600,000mE, 6,387,500mN (NAD83 UTM Zone 12N). The approximate centre of the Triple R deposit is located at UTM coordinates 598,000mE, 6,390,000mN (NAD83 UTM Zone 12N). Elevation on the Property varies between 499 masl and 604 masl.



Source: SLR, 2022

Figure 1-1: Property Location

1.1.2 Land Tenure

The PLS Property consists of 17 contiguous mineral claims covering an area of 31,039 ha located on the southwest margin of the Athabasca Basin. The Triple R deposit is located on claim S-111376. The mineral claims constituting the PLS Property were ground staked and are therefore designated as non-conforming legacy claims. As of the effective date of this Technical Report, all 17 mineral claims comprising the PLS Property are in good standing and registered in the name of FCU.

1.2 Accessibility, Climate, Local Resources, Infrastructure, and Physiography

The PLS Property is accessible via the all-weather gravel Highway 955 (Cluff Lake Mine Road) that originates at La Loche, heads northwards and enters the PLS Property at the 144 km marker. Highway 955 bisects the PLS Property in a north-south direction. Numerous access roads branch off Highway 955, allowing access to the east and west halves of the PLS Property.

The PLS Property is located within the Mid-Boreal Upland Ecoregion of the Boreal Shield Ecozone (Marshall and Schutt 1999). The summers are short and cool, and the winters are long and cold. The ground is snow-covered for six to eight months of the year. The ecoregion is classified as having a sub-humid high boreal ecoclimate. The mean temperature recorded at the Cluff Lake Station is about -20.4°C in January and 16.9°C in July. The average annual precipitation is approximately 451 mm at the Cluff Lake Station.

Various services are available at La Loche, including fuel, and emergency medical services. A greater range of services is available in Prince Albert and Saskatoon. Fixed-wing aircraft are available for charter at Fort McMurray in Alberta and Buffalo Narrows, La Loche, and La Ronge in Saskatchewan. Helicopters are available for charter at Fort McMurray and La Ronge.

Except for the all-weather gravel Highway 955, which traverses the PLS Property, there is no permanent infrastructure on the PLS Property.

The topography of northern Saskatchewan is characterized by low hills, ridges, drumlins, and eskers, with lakes and muskeg common in the low-lying areas. Outcrop of the underlying Athabasca sandstone and basement rocks is rare. Numerous lakes and ponds generally show a north-easterly elongation imparted by the most recent glaciation. Elevation on the property varies between 499 masl and 604 masl.

1.3 Geology and Mineralization

The most significant uranium metallogenic district in Canada is the Athabasca Basin, which covers over 85,000 km^2 in northern Saskatchewan and northeastern Alberta. The east-west elongate Athabasca Basin lies astride two subdivisions of the Western Churchill Province, the Rae Subprovince (Craton) to the west and the Hearne Subprovince (Craton) to the east. These are separated by the northeast-trending Snowbird Tectonic Zone, also known as the Virgin River Shear Zone or Black Lake Shear Zone, south and north of the Athabasca Basin, respectively.

The PLS Property is located within the Clearwater and Taltson Domains of the Rae Subprovince near the southwestern edge of the Athabasca Basin. The western portion of the PLS Property overlies the Clearwater Domain, and the eastern portion overlies the Taltson Domain. The PLS Property lies within the northeastern limits of the Cretaceous Mannville Group (Mannville Group), which covers a large portion of western Saskatchewan. The Lexicon of Canadian Geologic Units (the Lexicon) describes the Mannville Group as interbedded marine and non-marine sands, shales, and calcareous sediments.

As of the effective date of this Technical Report, appreciable high-grade mineralization is known to occur at the PLS Property in five zones, which collectively constitute the Triple R deposit. From west to east, these zones are: 1) R1515W, 2) R840W, 3) R00E, 4) R780E, and 5) R1620E, the most significant of which is the R780E zone. The R780E zone was discovered during the winter 2013 drill program with drill hole PLS13-038. Drill hole PLS13-038 intersected a 34.0 m wide zone of very strong uranium mineralization, beginning at 87.0 m, averaging 4.9% U_3O_8. Uranium mineralization at the PLS Property is hosted primarily within metamorphosed basement lithologies and, to a much lesser extent, within overlying Meadow Lake Formation sedimentary rocks.

1.4 Drilling

To date, FCU and its predecessors have completed a total of 844 drill holes, totalling 227,775 m across the PLS Property. Drilling includes exploration, geotechnical, metallurgical, water wells, and hydrogeology drill holes.

From November 2011 to September 2015, 142,832 m of drilling was completed in 454 diamond drill holes on the PLS Property. During the winter 2015 drill program, an initial Inferred Mineral Resource estimate for the Triple R deposit was published. Following the spring 2015 drill program, Roscoe Postle Associates Inc. (RPA) completed a preliminary economic assessment (PEA) on the Triple R deposit.

From January 2016 to December 2018, FCU continued to conduct both delineation and step-out drilling programs along the strike of the Triple R deposit by completing 52,983 m of drilling in 169 holes. Drill holes were primarily designed to both infill in support of an Indicated Mineral Resource classification in the R780E high grade (HG) and R780E Main Zone (MZ) domain and materially expand the footprint of Inferred mineralization in the R00E and R780E areas. Step-out regional drilling during this time was also successful in identifying two significant new areas of mineralization (R1515W and R1620E) and expanding mineralization at R840W. The goal of the summer 2018 program, which consisted of nine holes totalling 2,928 m drilled, was to drill key areas of the R780E HG zone that were classified in 2015 as "Inferred" and upgrade them to "Indicated". To that extent, the nine drill holes intersected the width and strength of mineralization where expected and allowed for upgrading the classification in these areas. Following the summer 2018 drill program, RPA, along with Clifton and Wood PLC (Wood), completed a pre-feasibility study (PFS) on the PLS property based on a total of 197,651 m of drilling in 636 drill holes.

Since September 19, 2019, FCU has completed an additional 181 drill holes totalling 27,392 m over the PLS Property, primarily focused on the R780E and R840W deposits.

The core from the first drilling programs was stored at the Big Bear Lodge on Grygar Lake, but since August 2013, all the core has been stored at a purpose-built storage facility located west of Patterson Lake.

1.5 Mineral Processing and Metallurgical Testing

A series of bench scale and bulk tests were conducted at SGS Canada Inc. – Mineral Services (SGS) Lakefield to support the feasibility level design of the process plant.

- High uranium extractions were achieved in a 12-hour leach, averaging 98.4% for all the tests, regardless of composite type, leach solid density, feed grind size, head grade, oxidant type, oxidation potential and free acid levels. The bulk leach test generated a pregnant leach solution for testing downstream processes.
- The counter current decantation (CCD) simulation showed that a six-stage thickener circuit would operate with a 99.5% wash efficiency based on a 3:1 pregnant leach solution to leach feed ratio.
- A five-day continuous solvent extraction (SX) mini pilot test showed 99.9% uranium recovery using four extraction stages and, on average, 99.4% stripping efficiency using five striping stages.
- Gypsum precipitation tests were completed for removing sulphates from the pregnant strip liquor before yellowcake precipitation. A two-stage washing of the gypsum cake could decrease the final washed gypsum cake grade to roughly 0.025% U_3O_8, representing greater than 95% uranium re-dissolution.
- Yellowcake precipitation using hydrogen peroxide and magnesia for pH control produced products averaging 80% U_3O_8, within refinery specifications.
- The uranium grade in the calcined yellowcake product was 95% U_3O_8 at a temperature of 450°C.
- Effluent treatment tests yielded a treated effluent meeting Canadian Metal and Diamond Mining Effluent Regulations (MDMER) guidelines.

1.6 Mineral Resource Estimate

Mineral Resources have been classified in accordance with the definitions for Mineral Resources in the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves dated May 10, 2014 (CIM 2014). Table 1-2 summarizes Mineral Resources based on a US$50/lb uranium price at a cut-off grade (COG) of 0.25% U_3O_8 and a potential underground scenario. Indicated Mineral Resources total 2.69 Mt at an average grade of 1.94% U_3O_8 for a total of 114.9 Mlb U_3O_8. Inferred Mineral Resources total 0.64 Mt at an average grade of 1.10% U_3O_8 for a total of 15.4 Mlb U_3O_8. Gold grades were also estimated and averaged 0.61 g/t for the Indicated Mineral Resources and 0.44 g/t for the Inferred Mineral Resources. Mineral Resources are inclusive of Mineral Reserves. The cut-off date of the Mineral Resource database is December 22, 2021, which represents the date on which all assays were received from FCU's Summer 2021 drill program. The effective date of the Mineral Resource estimate is May 17, 2022.

Table 1-2: Mineral Resource Statement – May 17, 2022

Category	Tonnage	Metal Grade		Contained Metal	
	(000 t)	(% U_3O_8)	(g/t Au)	(Mlb U_3O_8)	(000 oz Au)
Indicated	2,688	1.94	0.61	114.9	52.7
Inferred	635	1.10	0.44	15.4	9.0

Notes:
1. CIM (2014) definitions were followed for Mineral Resources.
2. Mineral Resources are reported at a COG of 0.25% U_3O_8, based on a long term price of US$50/lb U_3O_8, an exchange rate of C$1.00/US$0.75, and cost estimates derived during the PFS with a metallurgical recovery of 95%.
3. Minimum mining width of 1 m was applied to the resource domain wireframe.
4. Mineral Resources are inclusive of Mineral Reserves.
5. Numbers may not add due to rounding.

1.7 Mineral Reserve Estimate

The Mineral Reserves for the PLS Property are based on the Mineral Resources with an effective date of May 17, 2022. Detailed mine designs have been generated, and modifying factors have been applied. For consistency with the resource table, the USD$65/lb and relevant exchange rate were applied. The Mineral Reserve includes a nominal amount of material above the mineralized waste cut-off of 0.03% U_3O_8 and below the incremental COG of 0.19% U_3O_8 that has been included based on the requirement to access certain mining areas or manage geotechnical conditions in a production area.

Estimates of mineralization and other technical information included herein have been prepared in accordance with the NI 43-101 – Standard of Disclosure for Mineral Projects. The Mineral Reserves are summarized in Table 1-3.

Table 1-3: Mineral Reserve Statement, FCU – PLS Property

Category	Tonnes (000 t)	Grade (% U_3O_8)	Contained Metal (Mlb U_3O_8)
Probable	--	--	--
R780E Zone	2,630	1.46	84.8
R00E Zone	56	1.24	1.5

table continues…

Category	Tonnes (000 t)	Grade (% U_3O_8)	Contained Metal (Mlb U_3O_8)
R840W Zone	322	1.04	7.4
Total Probable	**3,007**	**1.41**	**93.7**

Notes:

1. CIM Definition Standards (2014) were followed for the classification of Mineral Reserves.
2. The Mineral Reserves are reported with an effective date of January 17, 2023.
3. Mineral Reserves were estimated using a long-term metal price of US$65 per lb of U_3O_8 and a US$/C$ exchange rate of 0.75 (C$1.00 = US$0.75)
4. Underground Mineral Reserves were estimated by creating stope shapes using Datamine's Mineable Shape Optimizer (MSO). The MSO outputs were evaluated in the context of the mine design, and then a 0.25% U_3O_8 cut-off was applied. For longhole stoping, a minimum mining width of 4 m (including hanging wall and footwall dilution) and stope height of 20 m was used. Following MSO, the mineable shapes were further subdivided in Deswik to produce a maximum width of 12 m (including hanging wall and footwall dilution). Drift and fill mining is designed at 5 m wide by 5 m high for development shapes located in the crown pillar areas of the orebodies.
5. Mining recovery of 95% was applied to all stopes, while all development mining assumes 100% extraction.
6. The density varies based on block model values. An estimated waste density of 2.42 t/m^3 was used for areas outside the block model boundary.
7. By-product credits were not included in the estimation of Mineral Reserves as the mill is not designed to recover gold (Au).
8. Numbers may not add due to rounding.

1.8 Mining Methods

The FS is based on accessing the deposit using a decline developed from a position southwest of the deposits in close proximity to the processing plant and waste stockpile areas. The area of the decline is temporarily dewatered while the development progresses through the overburden. The decline excavation is planned to use a tunnel shield method utilizing a hydrostatic segmental concrete liner for ground support. In addition to the decline, two vertical shafts are excavated sequentially to provide a dedicated ventilation system for the mine (one fresh air intake shaft [FAS] and one exhaust air shaft [EAS]). After the decline extends through the overburden and transition bedrock zone, more typical hard rock development can commence. Mining uses the Longhole Stoping method in a longitudinal retreat orientation with cemented rock fill as the backfill.

A partial recovery of the mineralized material approaching the contact between the overburden and bedrock is achieved by utilizing artificial ground freezing to achieve a bulk freeze. The ground is frozen by way of drilling holes into the overburden and shallow bedrock using underground drilling collared from a dedicated freeze drift below the crown area. Upon completion of the ground freezing holes and installation of freeze pipes, a refrigeration plant pumps a chilled brine solution through the pipes to create a frozen cap to provide increased ground stability and reduced groundwater inflow. Once frozen, a low disturbance drift and fill (D/F) mining method with cemented hydraulic fill (CHF) is utilized to extract the mineralized material. Roadheader tunneling equipment will be used in the crown pillar areas to remove the need for explosives. A portion of the Mineral Resources approaching the overburden contact will be sterilized due to geotechnical constraints; however, this sterilized material could be further evaluated for eventual extraction in future analysis.

1.9 Recovery Methods

Tetra Tech completed the design for the process plant and related infrastructure facilities for this FS using proven uranium extraction technology, processes and equipment and has drawn on its knowledge of other Athabasca uranium plants, including Rabbit Lake, Key Lake, and McClean Lake. The processing plant has been designed to process ore at a nominal throughput of 1,000 t/d to produce market-grade uranium concentrate. The average life of mine (LOM) mill feed grade will be 1.41% U_3O_8, and the anticipated overall U_3O_8 recovery will be 97.0%.

A conventional grinding and leaching circuit will be used for the uranium extraction process. The ore will be trucked from the mine to the run of mine (ROM) pad and ground in a single-stage semi-autogenous (SSAG) grinding circuit to 80% passing 150 µm. The ground ore will be leached using sulphuric acid and hydrogen peroxide at 50°C. The leached slurry will be fed to a CCD circuit followed by a clarification stage to produce the pregnant leach solution. An SX circuit will purify and concentrate uranium in the solution for yellowcake precipitation. The precipitated yellowcake will be calcined at 450°C as U_3O_8 before packaging in drums.

Tailings will be neutralized and deposited in the TMF. Effluent and contact water will be treated, monitored, and sampled before being discharged.

1.10 Project Infrastructure

The Project will require the development of several infrastructure items. The locations of Project facilities and other infrastructure items were determined with considerations in local topography, environment, and capital and operating costs. FS project infrastructure will include:

- Fresh and exhaust air ventilation shafts, a decline for ore transport from underground to the surface, a freeze plant, dewatering wells, a backfill plant and an intermediate settling/polishing pond
- Process facilities including ore stockpile, process plant with SX circuit, acid plant, effluent treatment facility, surface run-off and monitoring ponds, and assay laboratory
- A TMF to safely manage the tailings and water associated with mill feed processing, tailings transport and disposition, and water reclamation from the TMF.
- On site connective access roads among site infrastructure and Highway 955 with site access controls
- Ancillary facilities, including:
  - Truck shop, machine shop and warehouse
  - Power plant and distribution system
  - Liquefied natural gas (LNG) storage and laydown area
  - Waste rock management facility (WRMF)
  - Accommodation and administration offices
  - Communications infrastructure
  - Fuel storage and fuel farm

1.10.1 Tailings Management Facility

The design scenario for the TMF is a subaqueous deposition of thickened slurry tailings into a lined pervious surround pit. Tailings will be transported to the TMF in a pipeline as a thickened slurry. Spill prevention and control measures for the slurry pipeline will be incorporated to provide protection against leaks and spills along the tailings pipeline corridor. The tailings will be sub-aqueously deposited using a relocatable barge in a manner that facilitates even distribution of tailings and prevents particle segregation to produce a uniform, low permeability consolidated tailings mass. A water cover consisting of a clarified tailings solution will be maintained to support barge deposition of tailings while preventing freezing of the tailings and providing a barrier to low energy radiation, dust and radon release. Excess water will be returned to the process plant for treatment and release into the environment. The TMF can also act as emergency storage for site water if there is a large storm event that overwhelms the site storage (e.g. a probable maximum precipitation [PMP] event). Excess water would be returned to the process plant for treatment over time and released into the environment.

After processing and the generation of precipitates, a total of 1,120 t of tailings solids will be produced daily. Provision has also been made for additional capacity by assuming that a 25% increase in daily tailings production will occur over the scheduled 10-year mine life. The tailings slurry, as deposited in the pit, will have a bulk density of approximately 40% solids by mass and will rapidly settle to a bulk density of approximately 50%. Sizing of the TMF was based on this rate of settlement plus ongoing consolidation of the tailings, the inclusion of a 3.0 m thick water cover, and provision for 2.0 m of freeboard in the final year of operation. The total storage available in the TMF is approximately 8,200,000 m^3.

The geotechnical design of the TMF will be in accordance with the Canadian Dam Association (CDA) guidelines and the technical bulletin on the application of the guidelines to mining dams (CDA 2014). The engineered double barrier system is essential for the successful operation of the TMF. The engineered double barrier on the TMF floor will consist of a thick soil-bentonite liner (SBL) overlain by a geomembrane. The SBL has been designed with a low hydraulic conductivity to provide a second barrier to seepage loss from the TMF. In addition, the ion exchange capacity of the soil-bentonite barrier will further attenuate releases of metals and radionuclides that may pass through the geomembrane liner.

The barrier system for the berm slopes will consist of a double geomembrane liner without an SBL underlay. The second membrane will maintain secure containment on the slopes where the applied head will be small due to the overlying free-draining filter that will conduct the tailings solution to the underdrain.

1.11 Environmental Studies and Permitting

The extensive baseline work has described the typical northern Saskatchewan terrain of the Athabasca Basin region. It has not identified anything that should significantly delay a project if proper planning and mitigations are incorporated into the Project design. Such mitigations would include but are not limited to, habitat compensation for any fish habitat disturbed by the Project, possibly terrestrial habitat compensation for woodland caribou habitat, and sufficient consultation with local First Nations and communities. The primary environmental goal will be the protection of Patterson Lake and the downstream water quality in the Clearwater River system, as this will likely be the focus of any concerns under the underground mining-only scenario.

Overall, the Project appears to be following applicable regulations governing exploration, drilling and land use, and FCU staff and contractors are aware of their duties to environmental and radiation protection. Early in the exploration program, there were some issues related to the excess clearing of trails and access to water bodies, but FCU has worked to repair those areas and reclaim them. The operations are neat and orderly, and the level of clearing and disturbance is commensurate with similar projects in northern Saskatchewan. The Project is visited frequently by Saskatchewan Conservation Officers to ensure compliance.

At a high level, the preliminary environmental risk assessment (PERA) was done to look at potential interactions of the project with the environment. Under the underground mining-only scenario, the main area of concern is the development and operation of the TMF and the protection of surface and groundwater quality. The mitigations proposed for the TMF appear protective of the environment in the long-term, post-decommissioning. The TMF will use the proven sub-aqueous deposition and pervious surround methodologies, and modelling results continue to show that the TMF, as proposed, will be protective of the environment in the long term. The TMF design is optimized for the existing geological and hydrogeological conditions and avoids widespread dewatering during operation. The potential impacts on Patterson Lake will be much less in the underground mining scenario and are largely related to protecting the water quality. This will need to be demonstrated in the Environmental Impact Assessment (EIA) and subsequent licensing.

Most of the remaining environmental risks are similar to those at existing uranium operations, which, in the modern era, have been demonstrated to have minimal impact on the local and regional environments with proper mitigation. Regardless, for all aspects of the Project, a detailed ERA will be required to ensure that nothing is missed and that all reasonable mitigations are included in the EIA and the Project design.

The ongoing baseline from 2013 to 2020 was adequate to include in an EIA; however, in 2021, FCU was informed by the Saskatchewan Ministry of Environment (ENV) Environmental Assessment and Stewardship Branch (EASB) that older data may not be sufficient for the EIA. FCU commissioned CanNorth to complete an updated baseline program to refresh the data and provide continuity with the data that has been collected since 2013. This updated baseline work also addressed any gaps in previous data collection, including areas now identified as part of the project footprint that had previously not been included. A refreshed heritage resources study was also part of the 2022 program. Moving forward, FCU will need to do some level of on-going monitoring to maintain the baseline database throughout the construction and operation periods.

The level of environmental review was commensurate with an FS and was not an exhaustive examination of all documentation nor a compliance audit, although it did include updating the PFS modelling for potential impacts from the TMF. The interpretation relies on the author’s more than 40 years of experience with Saskatchewan uranium projects and familiarity with mining and the federal and provincial requirements that accrue to such projects. The Project is at a stage where, with proper planning, areas of concern can be addressed in a timely fashion within an orderly project approvals process.

Some of the items required to support an EIA, particularly consultation, need to be undertaken in a manner that does not materially affect Project timing. This will require ongoing consultation with the Canadian Nuclear Safety Commission (CNSC) and the Saskatchewan Government to ascertain the level of First Nations, Métis, and stakeholder consultation they expect as well as their expectations in other areas. With the signing of agreements related to engagement and information sharing during the EIA period with the main Indigenous rights holders, FCU has continued to leverage its good relations with these groups in a respectful manner. These agreements establish the basis for FCU's ongoing relationship with these groups and set the stage for any accommodation agreements.

FCU’s level of governance continues to be adequate for the level of work on-site and the EIA regulatory period, but it will require significant improvement to support the policy-driven management systems required to support a uranium project and the CNSC’s safety and control requirements. FCU will be working on this next step in 2023. The feasibility level engineering done to support this FS will be sufficient to support the EIA process with minor amounts of additional detail, as necessary. While it is not sufficient to support most of the licensing requirements for construction and operations, that additional work will be started in 2023, the FS work provides evidence that the Project can be constructed in a manner that protects the environment and public health and safety.

1.12 Capital and Operating Costs

1.12.1 Capital Cost Estimate

The total estimated initial and sustaining capital cost for the design, construction, installation, and commissioning of the Project is $1,539 million. This includes all direct costs, indirect costs, owner’s costs, and contingency. The capital cost estimate is consistent with an Association for the Advancement of Cost Engineering (AACE) Class 3 estimate with the expected accuracy of ±15%. A summary breakdown of the capital cost is provided in Table 1-4.

Table 1-4: Capital Cost Summary

Capital Cost Area	Value ($ millions)
Mining	176
Processing	141
Infrastructure	159
TMF	235
Direct Costs	**711**
Indirect Costs	198
Owner's Costs	109
Contingency	137
Total Initial Capital Cost	**1,155**
Total Sustaining Capital Cost	384
Total Capital Cost	**1,539**

1.12.2 Operating Cost Estimate

The Project operating cost estimate consists of mining, processing, and general and administration (G&A) costs, are summarized in Table 1-5. The average operating cost is estimated at $393/t ore processed, or $13.02/lb U_3O_8 produced.

Table 1-5: Average LOM Operating Cost Summary

Description	LOM Cost ($ millions)	Unit Cost ($/t processed)	Unit Cost ($/lb U_3O_8)
Mining	458.8	152.55	5.05
Processing	489.6	162.78	5.39
G&A	234.9	78.12	2.59
Total LOM Capital Cost	**1,183.3**	**393.45**	**13.02**

1.13 Economic Analysis

The Project has been evaluated using a constant U_3O_8 market price of US$65/lb U_3O_8, reflecting the recent upturn in the spot price. The LOM base case Project net cash flow before and after tax is presented in Table 1-6. Applying an annual discount rate of 8%, the Project base case post-tax cash flow evaluates to a net present value (NPV) of $1,204 million and an internal rate of return (IRR) of 27%. The post-tax payback period is 2.6 years when discounted at 8% per year.

Table 1-6: Summary of Economic Analysis Results

Parameter	Unit	Pre-Tax	Post-Tax
Undiscounted Net Cash Clow (NCF)	$ billion	4.508	2.787
NPV @ 8% discount	$ billion	2.095	1.204
IRR	%	35.5%	27.2%
Payback Period	year	2.3	2.6

1.14 Conclusions and Recommendations

The PLS Project is considered to be technically and economically viable based on the FS parameters and results.

It is recommended that FCU advance the PLS Project by completing the front-end engineering and design (FEED), the Project permitting process, detailed engineering, planning and scheduling, and source financing. Summary of estimated cost to complete is presented in Table 26-1.

2.0 INTRODUCTION

FCU commissioned Tetra Tech to complete the FS with the assistance of specialist consultants for the PLS Project, in accordance with the NI 43-101 Standards of Disclosure for Mineral Projects. FCU is a Canada-based resource company specialising in the strategic exploration and development of the PLS Property.

The PLS Property is currently 100% owned by FCU. Description and location of the PLS Property is presented in Section 4 of this Technical Report, which also includes an updated Mineral Resource estimate and Mineral Reserve estimate of the Property. The Qualified Persons (QPs) that authored this Technical Report are independent of FCU and the Property.

The list of consultants responsible for each report section is presented in Table 2-1.

Table 2-1: FS Technical Report Sections, Consultants, and QPs

No.	Section	Company	QP
1.0	Summary	Tetra Tech	Sign-off by Section
2.0	Introduction	Tetra Tech	Hassan Ghaffari, P.Eng.
3.0	Reliance on Other Experts	Tetra Tech	Hassan Ghaffari, P.Eng.
4.0	Property Description and Location	SLR Consulting	Mark Mathisen, C.P.G.
5.0	Accessibility, Climate, Local Resources, Infrastructure, and Physiography	SLR Consulting	Mark Mathisen, C.P.G.
6.0	History	SLR Consulting	Mark Mathisen, C.P.G.
7.0	Geological Setting and Mineralization	SLR Consulting	Mark Mathisen, C.P.G.
8.0	Deposit Types	SLR Consulting	Mark Mathisen, C.P.G.
9.0	Exploration	SLR Consulting	Mark Mathisen, C.P.G.
10.0	Drilling	SLR Consulting	Mark Mathisen, C.P.G.
11.0	Sample Preparation, Analyses, and Security	SLR Consulting	Mark Mathisen, C.P.G.
12.0	Data Verification	SLR Consulting	Mark Mathisen, C.P.G.
13.0	Mineral Processing and Metallurgical Testing	Tetra Tech	Jianhui (John) Huang, P.Eng.
14.0	Mineral Resource Estimates	SLR Consulting	Mark Mathisen, C.P.G.
15.0	Mineral Reserve Estimates	Mining Plus	Maurice Mostert, P.Eng.
16.0	Mining Methods	Mining Plus	Maurice Mostert, P.Eng.
	Geotechnical	BGC	Catherine Schmid, P.Eng.
	Hydrogeology	BGC	Randi Thompson, P.Eng.
17.0	Recovery Methods	Tetra Tech	Patrick Donlon, FSAIMM, FAusIMM
18.0	Project Infrastructure	Tetra Tech	Hassan Ghaffari, P.Eng.

table continues…

No.	Section	Company	QP
	Tailings Management Facility	Clifton	Wayne Clifton, P.Eng.
19.0	Market Studies and Contracts	Tetra Tech	Hassan Ghaffari, P.Eng.
20.0	Environmental Studies, Permitting, and Social or Community Impact	Clifton	Mark Wittrup, P.Eng., P.Geo
21.0	Capital Costs and Operating Costs	Tetra Tech	Hassan Ghaffari, P.Eng., Jianhui (John) Huang, P.Eng.
	Capital Costs and Operating Costs (Mining)	Mining Plus	Maurice Mostert, P.Eng.
22.0	Economic Analysis	Tetra Tech	Hassan Ghaffari, P.Eng.
23.0	Adjacent Properties	SLR Consulting	Mark Mathisen, C.P.G.
24.0	Other Relevant Data and Information	Tetra Tech	Hassan Ghaffari, P.Eng.
25.0	Interpretation and Conclusions	Tetra Tech	Sign-off by Section
26.0	Recommendations	Tetra Tech	Sign-off by Section
27.0	References	Tetra Tech	Sign-off by Section
28.0	Certificates of Qualified Persons	Tetra Tech	Sign-off by Section

Mineral Resource Estimate Update

The Mineral Resource estimate is current as of the effective date of this Technical Report and is presented in Section 14 of the Report.

Mineral Reserve Estimate Update

The Mineral Resource estimate is current as of the effective date of this Technical Report and is presented in Section 15 of the Report.

FS News Release

The results of the FS were disclosed in FCU's press release dated January 17, 2023. This Technical Report is filed in support of the disclosure of the FS results.

2.1 Sources of Information

The key information sources for this Technical Report were:

- Documents referenced in Section 3 (Reliance on Other Experts) of this Technical Report
- Documents referenced in Section 27 (References) of this Technical Report
- Additional information provided by FCU personnel where required

2.2 Effective Dates

This Technical Report has the following effective dates:

- Mineral Resource estimate: May 17, 2022
- Mineral Reserve estimate: January 17, 2023
- The overall effective date of this Technical Report is January 17, 2023

2.3 Qualified Persons

The name of the QPs of this Technical Report and their QP certificates are included in Section 28.

2.4 Personal Inspections

The following QPs conducted a site visit of the Property:

- Jianhui (John) Huang (Ph.D., P.Eng.) of Tetra Tech conducted a personal inspection of the Property on October 13 and 14, 2021.
- Mark B. Mathisen (C.P.G.) of SLR conducted the most recent personal inspection of the Property from August 6 to 8, 2018.
- Mark Wittrup (P.Eng., P.Geo.) of Clifton conducted the most recent personal inspection of the Property on June 13 and June 14, 2019.
- Maurice Mostert (P.Eng., FSAIMM.) of Mining Plus conducted a personal inspection of the Property on October 13 and 14, 2021.
- Catherine Schmid (P.Eng.) of BGC conducted the most recent personal inspection of the Property from June 16 to June 21, 2021.

3.0 RELIANCE ON OTHER EXPERTS

This Technical Report has been prepared by Tetra Tech, SLR, Clifton, Mining Plus and BGC for FCU. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to Tetra Tech, SLR, Clifton, Mining Plus, and BGC at the time of preparation of this Technical Report.
- Assumptions, conditions, and qualifications as set forth in this Technical Report.

For the purpose of this Technical Report, the authors have relied on ownership information provided by FCU. The authors have not researched property title or mineral rights for the PLS Property and express no opinion as to the ownership status of the PLS Property. SLR did review the status of the mineral claims on the web site of the Saskatchewan Ministry of Economy (http://economy.gov.sk.ca/mining). The information for the mineral claims constituting the PLS Property is as noted in Section 4 of this Technical Report as of May 17, 2022, the date of SLR's review.

Tetra Tech has relied on SLR for guidance on market studies.

Tetra Tech has relied on FCU and their tax advisors for guidance on applicable taxes, royalties, and other government levies or interests, applicable to revenue or income from the Project.

Except for the purposes legislated under provincial securities laws, any use of this Technical Report by any third party is at that party's sole risk.

4.0 PROPERTY DESCRIPTION AND LOCATION

The PLS Property, which includes the Triple R deposit discovered in 2012, is located in northern Saskatchewan, approximately 550 km north-northwest of the city of Prince Albert by air and 157 km north of the community of La Loche by road, as illustrated in Figure 4-1. The PLS Property is accessible by vehicle along all-weather gravel Highway 955, which bisects the PLS Property in a north-south direction and continues north through the area to the past producing Cluff Lake uranium mine.

The geographic coordinates for the approximate centre of the PLS Property are 57°37' N latitude and 109°22' W longitude which corresponds to the UTM geographic coordinates of 600,000mE, 6,387,500mN (NAD83 UTM Zone 12N). The PLS Property is located within 1:50,000 scale NTS map sheets 74F/11 (Forrest Lake) and 74F/12 (Wenger Lake). It is irregularly shaped and extends for approximately 29 km in the east-west direction and for approximately 19 km in the north-south direction. The approximate centre of the Triple R deposit is located at UTM coordinates 598,000mE, 6,390,000mN (NAD83 UTM Zone 12N).

4.1 Mineral Rights

In Canada, natural resources fall under provincial jurisdiction. In the province of Saskatchewan, the management of Mineral Resources and the granting of exploration and mining rights for mineral substances and their use are regulated by the Crown Minerals Act and The Mineral Tenure Registry Regulations, 2012, that are administered by the Saskatchewan Ministry of the Economy. Mineral rights are owned by the Crown and are distinct from surface rights.

As of December 6, 2012, mineral dispositions are defined as electronic mineral claims parcels within the Mineral Administration Registry System (MARS) using a Geographical Information System (GIS). MARS is a web-based electronic tenure system for issuing and administrating mineral permits, claims, and leases. Mineral claims are now acquired by electronic map staking and administration of the dispositions is also web based.

In order to maintain mineral claims in good standing in the province of Saskatchewan, the claim holder must undertake prescribed minimum exploration work on a yearly basis. The current requirements are $15/ha per year for claims that have existed for 10 years or less and $25/ha per year for claims that have existed in excess of 10 years.

Mineral claims in good standing may be converted to mineral lease(s) upon application. Mineral leases allow for mineral extraction, have 10-year terms, and are renewable. A lease proffers the holder with the exclusive right to explore for, mine, work, recover, procure, remove, carry away, and dispose of any Crown minerals within the lease lands which are nonetheless owned by the Province.

Surface rights are a distinct and separate right from subsurface or mineral rights. To obtain surface rights in support of mining operations, negotiation with the landowner may be required in the case of private property. In the case of Crown lands, a surface permit must be obtained.

Surface facilities and mine workings constructed in support of mineral extractions require a surface lease from the province of Saskatchewan. A surface lease carries a maximum term of 33 years, and may be extended as necessary to allow the lessee to develop and operate the mine and plant, and thereafter to carry out the reclamation of the lands involved.

4.2 Land Tenure

The PLS Property consists of 17 contiguous mineral claims covering an area of 31,039 ha located on the southwest margin of the Athabasca Basin, as illustrated in Figure 4-2. The Triple R deposit is located on claim S-111376. Table 4-1 lists the relevant tenure information for the PLS Property.

Table 4-1: Land Tenure

Claim	Effective Date	Anniversary Date	Good Standing Date	Area (ha)	Status
S-110707	28-Mar-07	27-Mar-19	25-Jun-39	812	Active
S-110955	31-May-07	30-May-19	28-Aug-39	1,327	Active
S-111375	13-Jun-08	12-Jun-19	10-Sep-39	2,493	Active
S-111376	13-Jun-08	12-Jun-19	10-Sep-39	3,310	Active
S-111377	13-Jun-08	12-Jun-19	10-Sep-39	1,645	Active
S-111783	30-Apr-10	29-Apr-19	28-Jul-39	1,004	Active
S-112217	13-Dec-11	12-Dec-19	12-Mar-39	1,202	Active
S-112218	13-Dec-11	12-Dec-19	12-Mar-39	1,299	Active
S-112219	13-Dec-11	12-Dec-19	12-Mar-39	987	Active
S-112220	13-Dec-11	12-Dec-19	12-Mar-39	1,218	Active
S-112221	13-Dec-11	12-Dec-19	12-Mar-39	2,621	Active
S-112222	13-Dec-11	12-Dec-19	12-Mar-39	846	Active
S-112282	22-Jun-11	21-Jun-19	19-Sep-39	3,789	Active
S-112283	22-Jun-11	21-Jun-19	19-Sep-39	1,003	Active
S-112284	22-Jun-11	21-Jun-19	19-Sep-39	2,021	Active
S-112285	22-Jun-11	21-Jun-19	19-Sep-39	5,404	Active
S-112370	23-Nov-11	22-Nov-19	20-Feb-39	58	Active

The mineral claims constituting the PLS Property were ground staked and are therefore designated as non-conforming legacy claims. As of December 6, 2012, the PLS Property and component claim locations were defined as electronic mineral claim parcels within the MARS. As of December 31, 2022, assessment credits totaling C$13,342,750.00 will be available for claim renewal. Required expenditures totaling C$775,975.00 will be required to renew the property claims upon their respective annual anniversary dates. In the absence of sufficient assessment credits, there is a provision in Saskatchewan to keep the claims in good standing by making a deficiency payment or a deficiency deposit.

As of the effective date of this Technical Report, all 17 mineral claims comprising the PLS Property are in good standing and registered in the name of Fission Uranium. The Project is located on Provincial Crown land; surface rights are obtained after successful ministerial decision, after an environmental decision, and following successful negotiation of a mineral surface lease agreement. FCU currently has a surface lease agreement that covers the core storage, core handling, Hub Camp, and laydown areas.

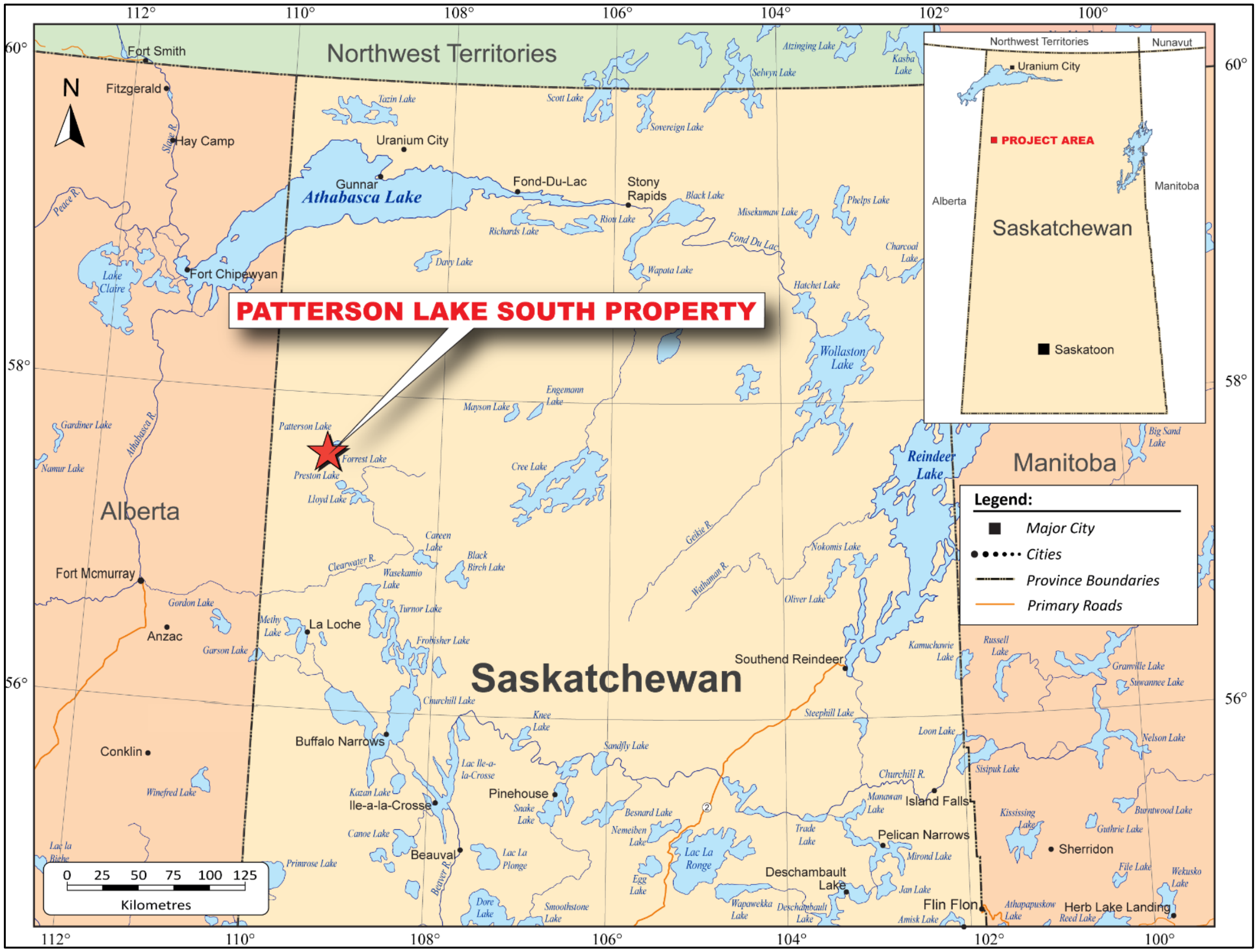
4.3 Royalties and Other Encumbrances

SLR is not aware of any royalties due, back-in rights, or other encumbrances to the Project including current and future permitting requirements and associated timelines, permit conditions, and violations and fines.

4.4 Permitting

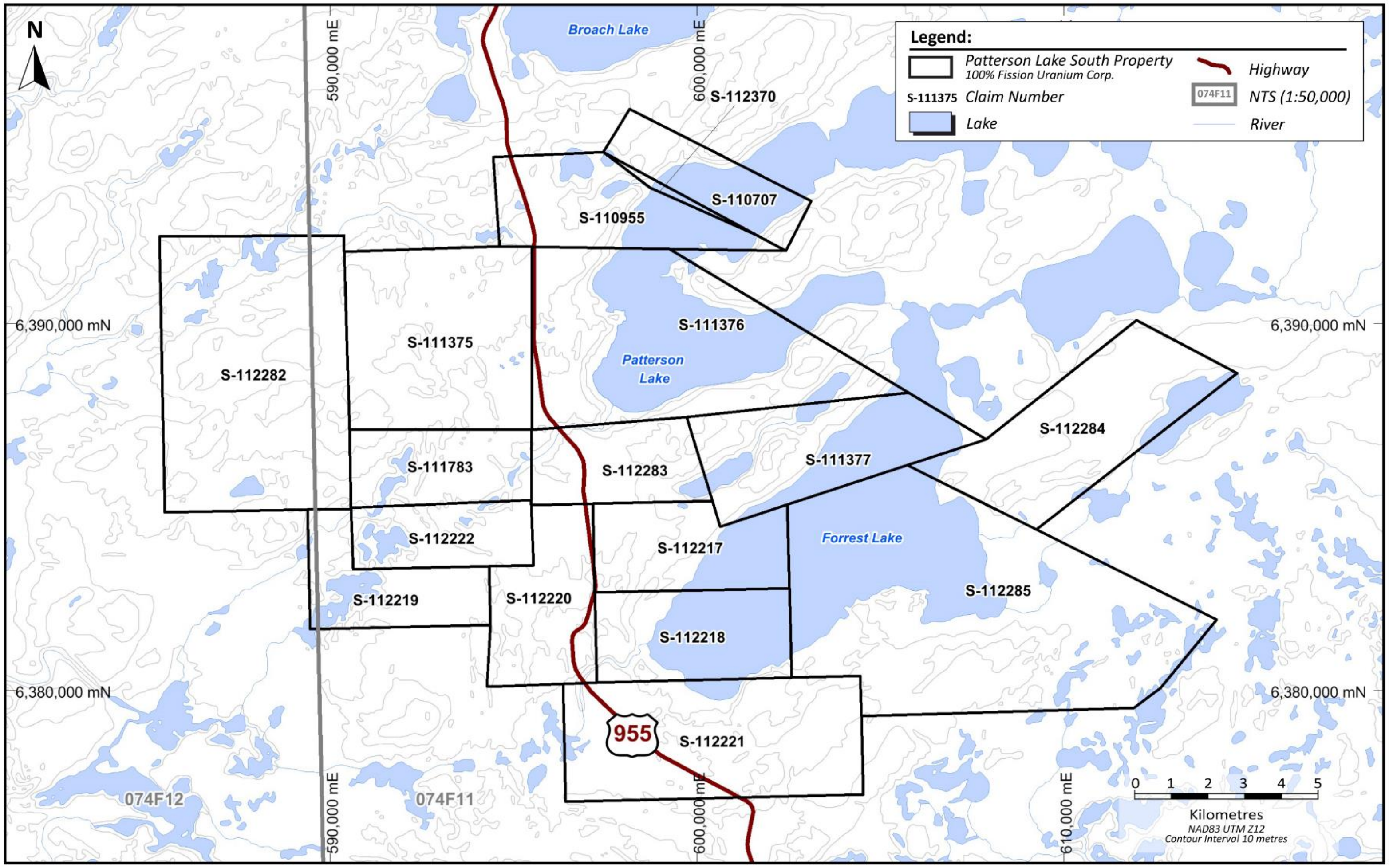
SLR is not aware of any environmental liabilities associated with the PLS Property.

SLR understands that FCU has all the required permits to conduct the proposed work on the PLS Property. SLR is not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform the proposed work program on the PLS Property.



Source: SLR, 2022

Figure 4-1: Property Location



Source: FCU, 2022

Figure 4-2: Land Tenure

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCRE, AND PHYSIOGRAPHY

5.1 Accessibility

The PLS Property is located approximately 550 km north-northwest of the city of Prince Albert, Saskatchewan. Prince Albert is serviced by multiple flights daily from Saskatoon. The PLS Property can be reached by driving northward along paved Highway 155 for a distance of approximately 300 km to the community of La Loche. At La Loche, the all-weather gravel Highway 955 (Cluff Lake Mine Road) heads northwards and enters the PLS Property at the 144 km marker. Highway 955 bisects the PLS Property in a north-south direction. Numerous access roads branch off from Highway 955 allowing access to the east and west halves of the PLS Property.

5.2 Climate

The PLS Property is located within the Mid-Boreal Upland Ecoregion of the Boreal Shield Ecozone (Marshall and Schutt 1999). The summers are short and cool, and the winters are long and cold. The ground is snow covered for six to eight months of the year. The ecoregion is classified as having a sub-humid high boreal ecoclimate. Table 5-1 illustrates the climatic data for the two most proximal Environment Canada weather stations.

Table 5-1: Climatic Data – Cluff Lake, Fort Chipewayan, and Fort MacKay

	Cluff Lake (SK) 58°22'N 109°31'W	Fort Chipewayan (AB) 58°46'N 111°07'W	Fort MacKay (AB) 56°58'N 111°27'W
Mean January temperature	-20.4°C	-21.9°C	-21.0°C
Mean July temperature	16.9°C	17.0°C	17.0°C
Extreme maximum temperature	36.0°C	34.7°C	37.0°C
Extreme minimum temperature	-49.0°C	-50.0°C	-50.6°C
Annual precipitation	451.0 mm	365.7 mm	414.0 mm
Annual rainfall	319.3 mm	250.4 mm	302.3 mm
Annual snowfall	162.8 cm	116.9 cm	133.8 cm

Despite the winter conditions, drilling and geophysical surveys can be performed year-round. Surface geochemical surveys are generally restricted to the snow-free months.

5.3 Local Resources

Various services are available at La Loche including fuel, and emergency medical services. A greater range of services is available in Prince Albert and Saskatoon. Fixed wing aircrafts are available for charter at Fort McMurray in Alberta and Buffalo Narrows, La Loche, and La Ronge in Saskatchewan. Helicopters are available for charter at Fort McMurray and La Ronge.

5.4 Infrastructure

With the exception of the all-weather gravel Highway 955, which traverses the PLS Property, there is no permanent infrastructure on the PLS Property.

5.5 Physiography

The topography of northern Saskatchewan is characterized by low hills, ridges, drumlins, and eskers, with lakes and muskeg common in the low-lying areas. Outcrop of the underlying Athabasca sandstone and basement rocks is rare. Numerous lakes and ponds generally show a north-easterly elongation imparted by the most recent glaciation. Elevation on the property varies between 499 masl and 604 masl.

Loamy, grey soils produce taller trees than in the Canadian Shield. Aspen, white spruce, jack pine, black spruce, and tamarack are common.

Wildlife consists of moose, woodland caribou, mule deer, white-tailed deer, elk, black bear, timber wolf, and beaver. Birds include white-throated sparrow, American redstart, bufflehead, ovenbird, and hermit thrush. Fish include northern pike, pickerel, whitefish, lake trout, rainbow trout, and perch.

The PLS Property is at the advanced feasibility and mine planning stage. The SLR QP is of the opinion that, to the extent relevant to the mineral project, there is a sufficiency of surface rights and water.

6.0 HISTORY

6.1 Prior Ownership

Claims comprising the PLS Property were ground staked from February 2007 to December 2011. Claim S-110707 was originally staked on behalf of ESO Uranium Corp. (ESO). Claim S-110955 was originally staked on behalf of Strathmore Minerals Corp. (Strathmore) and transferred to Fission Energy Corp. (Fission Energy) in its plan of arrangement. In January 2008, Fission Energy and ESO entered into a 50/50 joint venture and contributed the claims existing at that time. As part of the agreement, Fission Energy contributed mineral claims S-110954 and S-110955 while ESO contributed S-110707 and S-110723. Mineral claims S-110954 and S-110723 were eventually allowed to lapse. Subsequently, additional claims were staked for the benefit of the joint venture, including S-111376 which is now known to host the Triple R deposit.

On March 7, 2013, Fission Energy announced that it had entered into an agreement (the Agreement) with Denison Mines Corp. (Denison) whereby Denison agreed to acquire all the issued and outstanding shares of Fission Energy. Under this Agreement, Fission Energy spun out certain assets, including its 50% interest in the PLS Property, into a newly formed, publicly traded company, FCU, by way of a court-approved plan of arrangement.

Pursuant to the Agreement, Denison acquired a portfolio of uranium exploration projects including Fission Energy's 60% interest in the Waterbury Lake uranium project, as well as Fission Energy's exploration interests in all other properties in the eastern part of the Athabasca Basin, its interests in two joint ventures in Namibia, plus its assets in Quebec and Nunavut. FCU's assets consisted of the remaining assets of Fission Energy including the 50% interest in the PLS Property.

6.2 Exploration and Development History

The following description of historic exploration work conducted on the PLS Property and its immediate vicinity is taken from Armitage (2013).

The PLS Property was geologically mapped as part of a larger area by W.F. Fahrig for the Geological Survey of Canada (GSC) in 1961 (Hill 1977). Another geological mapping project completed in 1961 by L.P. Tremblay of the GSC covered the PLS Property and Firebag River area at a scale of four miles to the inch (Hill 1977).

In 1969, photogeologic mapping and airborne radiometric and magnetic surveys were completed on the PLS Property for Wainoco Oil and Chemicals Ltd. The surveys did not detect any notable structures or anomalies (Atamanik, Downes and van Tongeren 1983).

CanOxy completed extensive exploration on and around the PLS Property from 1977 to 1981. Exploration comprised an airborne Questor INPUT electromagnetic (EM) survey; horizontal loop EM (HLEM) and magnetic geophysical surveys; geological, geochemical, alphameter (radon), and radiometric surveys; and diamond drilling.

In 1977, CanOxy discovered a very strong six station alphameter (radon) anomaly with dimensions of 1.2 km by 1.7 km on what is now claim S-111375. This anomaly coincides with high uranium in soil values and anomalous scintillometer (radiometric) values. It was suggested that this alphameter anomaly was responding to radioactive exotic boulders within the till of the Cree Lake Moraine, however, no follow-up work was carried out (Hill 1977).

CanOxy's 1977 ground EM survey delineated the Patterson Lake Conductor Corridor that traverses the centre of Patterson Lake on claim S-111376 and extends onto claim S-111375. Several disrupted conductors and inferred cross cutting features were identified as priority 1, 2, and 3 drill targets on claim S-111376.

CanOxy drill hole CLU-12-79 was positioned based on an airborne EM conductor, which was later refined by ground EM surveys. This drill hole is located on the northernmost conductor of the Patterson Lake conductor corridor and is on the west shore of Patterson Lake within claim S-111376. Drill hole CLU-12-79 was highlighted by a 6.1 m wide sulphide-graphite "conductor" that contained anomalous uranium, copper, and nickel concentrations. Strong hematite and chlorite alteration were observed in the regolith and fresh basement rock, and two spikes in radioactivity occurred in the fresh basement lithologies (Robertson 1979).

6.3 Previous Resource Estimates

An initial Mineral Resource estimate was reported for the Triple R deposit in an NI 43-101 Technical Report by RPA dated February 12, 2015 (RPA 2015a). An updated Mineral Resource estimate for the Triple R deposit was prepared by RPA on September 14, 2015 (RPA 2015b). A further updated Mineral Resource estimate for the Triple R was prepared by RPA, Wood, and Clifton on May 30, 2019 (RPA 2019a), and again on September 19, 2019 (RPA 2019b). RPA is now part of SLR.

All previous Mineral Resource estimates are superseded by the updated Mineral Resource estimate in Section 14 of this Technical Report.

6.4 Past Production

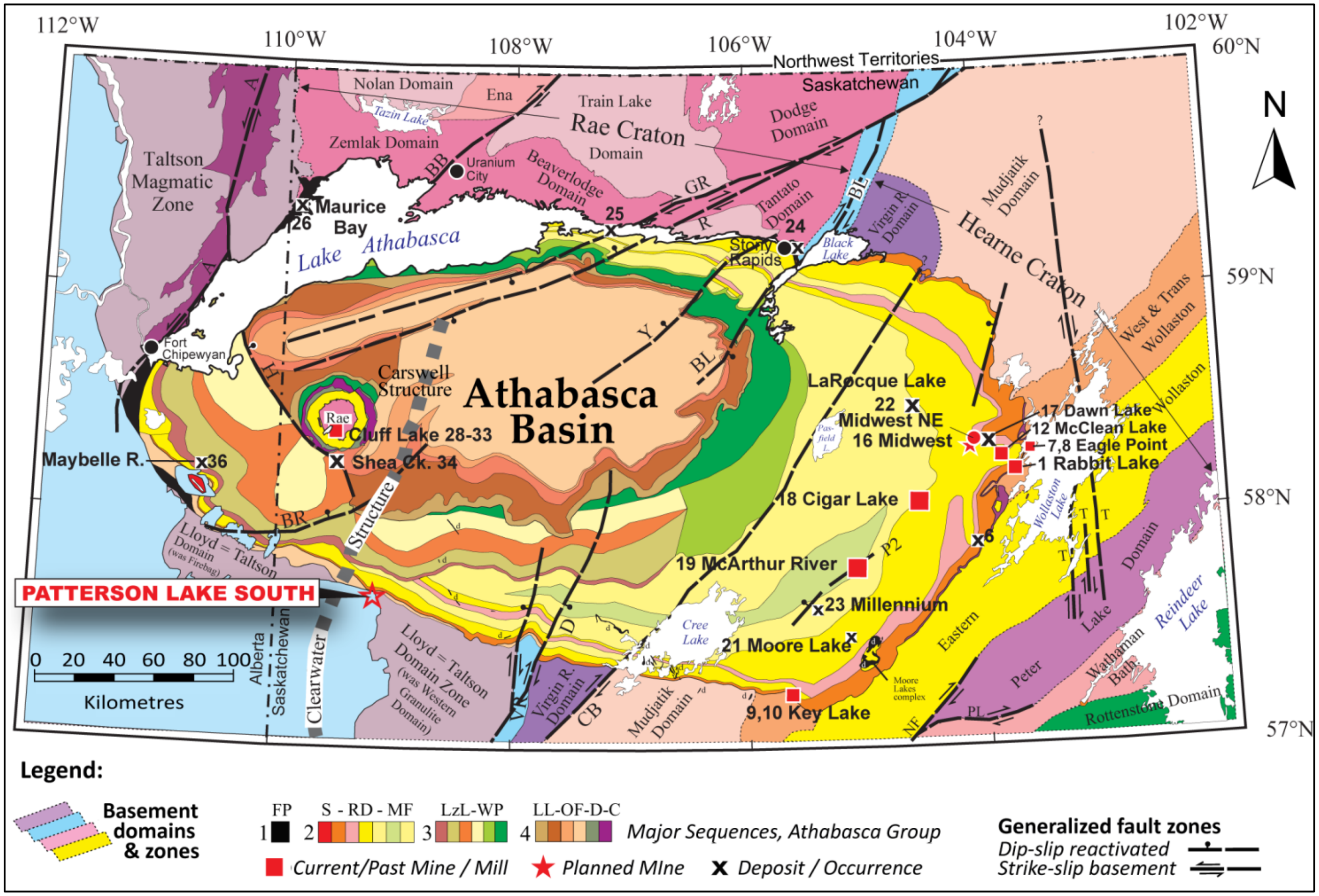
There has been no production from the PLS Property up to the effective date of this Technical Report.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The most significant uranium metallogenic district in Canada is the Athabasca Basin, which covers over 85,000 km^2 in northern Saskatchewan and northeastern Alberta, as illustrated in Figure 7-1. The Athabasca Basin is oval shaped at surface with approximate dimensions of 450 km by 200 km and reaches a maximum thickness of approximately 1,500 m near the centre. The basin itself is a relatively undeformed and unmetamorphosed sequence of Paleoproterozoic to Mesoproterozoic clastic rocks known as the Athabasca Group, lying unconformably on the deformed and metamorphosed rocks of the Western Churchill Province of the Canadian Shield.

The east-west elongate Athabasca Basin lies astride two subdivisions of the Western Churchill Province, the Rae Subprovince (Craton) to the west, and the Hearne Subprovince (Craton) to the east. These are separated by the northeast trending Snowbird Tectonic Zone also known as the Virgin River Shear Zone or Black Lake Shear Zone, south and north of the Athabasca Basin, respectively. The PLS Property is located within the Clearwater and Taltson Domains of the Rae Subprovince near the southwestern edge of the Athabasca Basin. The western portion of the PLS Property overlies the Clearwater Domain and the eastern portion of the PLS Property overlies the Taltson Domain.



Source: after GSC – Mineral Deposits of Canada, Unconformity Associated Uranium Deposits

Figure 7-1: Regional Geology

7.2 Local Geology

The following description of the local geology is revised after Armitage (2013).

The PLS Property lies within the northeastern limits of the Cretaceous Mannville Group (Mannville Group), which covers a large portion of western Saskatchewan, as illustrated in Figure 7-2, Figure 7-3, and Figure 7-4. The Lexicon of Canadian Geologic Units (the Lexicon) describes the Mannville Group as interbedded marine and non-marine sands, shales, and calcareous sediments.

Regionally discontinuous Devonian age Elk Point Group exists beneath the Cretaceous sediments. The Lexicon describes the Elk Point Group as being comprised of nine distinct lithologies consisting primarily of carbonates, evaporites, and clastic rocks.

To date, no Athabasca Group sediments have been intersected on the PLS Property.

Basement rocks of the PLS Property consist of the Clearwater and Taltson Domains. Although not well defined due to limited exposure and mapping, the Clearwater Domain is recognized to consist of gneissic granitoids, anorthosite, monzodiorite, and granites (Card et al. 2014). The Paleoproterozoic Taltson Domain rocks are comprised of granulite facies orthogneisses derived predominantly from diorite, quartz diorite, and quartz monzodiorite, with subordinate tonalite, granodiorite, and granite (Card et al. 2014). Mafic to ultramafic rocks commonly intrude the orthogneisses.

7.3 Property Geology

The following description of the property geology is taken from Mineral Services Canada Inc. (2014a) with revisions after 2018 drilling.

7.3.1 Quaternary Geology

The PLS Property is covered by a thick layer of sandy to gravelly Quaternary glacial material. The Quaternary material ranges in thickness from less than 10 m in the southeast portion of the PLS Property to greater than 100 m directly west of Patterson Lake. No outcrop has been discovered on the PLS Property to date. Eskers, drumlins, and other glacial features show a general north-easterly trend imparted by the most recent glaciation. A roughly north-south orientation is present in the glacial features in the vicinity of the radioactive boulder field west of Patterson Lake, which is interpreted to reflect a glacial outwash plain. Occasional drill holes west of Patterson Lake also intersect apparently thick intervals of glacial diamictite. The diamictite is comprised of dark grey to black silty matrix material with subangular pebble to gravel sized Athabasca sandstone and basement clasts throughout.

7.3.2 Mannville Group

Intermittently on the PLS Property, particularly to the west of Patterson Lake, intervals of dark grey, Cretaceous Mannville Group mudstone have been intersected, interpreted to be the Cantuar Formation. The thickness of the Cantuar Formation appears highly variable, which is likely a result of being washed away during drilling, however, it has been intersected in lengths in excess of 20 m (e.g., PLS12-017). Thin seams of coal are occasionally present within the mudstone.

7.3.3 Elk Point Group

The lowest formation of the Elk Point Group, the Meadow Lake Formation, occurs as a thin intermittent lens on the PLS Property. The greatest proportion of Meadow Lake Formation cored to date occurs in holes drilled to intersect the R00E and R780E mineralized zones. The Meadow Lake Formation is generally medium grained, brownish in colour when fresh, and contains numerous poorly sorted subangular basement and Athabasca sandstone clasts. The matrix around mineral and lithic clasts is well developed and made up of carbonate (MSC 2012). Typical thicknesses of Meadow Lake Formation range widely, from tens of cm to over 10 m. The Meadow Lake Formation is interpreted to be the remaining infill of a basement low over mineralization and has been found to taper off rapidly away from the mineralized zone.

Alteration within the Meadow Lake Formation, when present, is dominated by pervasive chlorite and illite, which turns the sediments whitish green to dark green. Pervasive pink-red hematite alteration also commonly occurs in more competent intervals.

Due to the limited amount of drilling in the Meadow Lake Formation, no significant structures have been noted within this area of the PLS Property to date.

7.3.4 Basement Rocks

The PLS Property covers two geological domains; the western portion covers the Clearwater Domain while the eastern portion covers the Taltson Domain. To date, drilling at the PLS Property has been focused on the basement rocks of the Taltson Domain. In the vicinity of PLS mineralization (i.e., along the PLG-3B EM conductor), the basement rocks are comprised of a northeast trending belt of variably altered and sheared pyroxene bearing orthogneisses bounded to the northwest and southeast by an apparently thick package of quartz-feldspar-biotite-garnet gneiss (QFBG-GN). The pyroxene bearing orthogneisses and QFBG-GN are intruded by numerous sheared, fine-grained granite lenses.

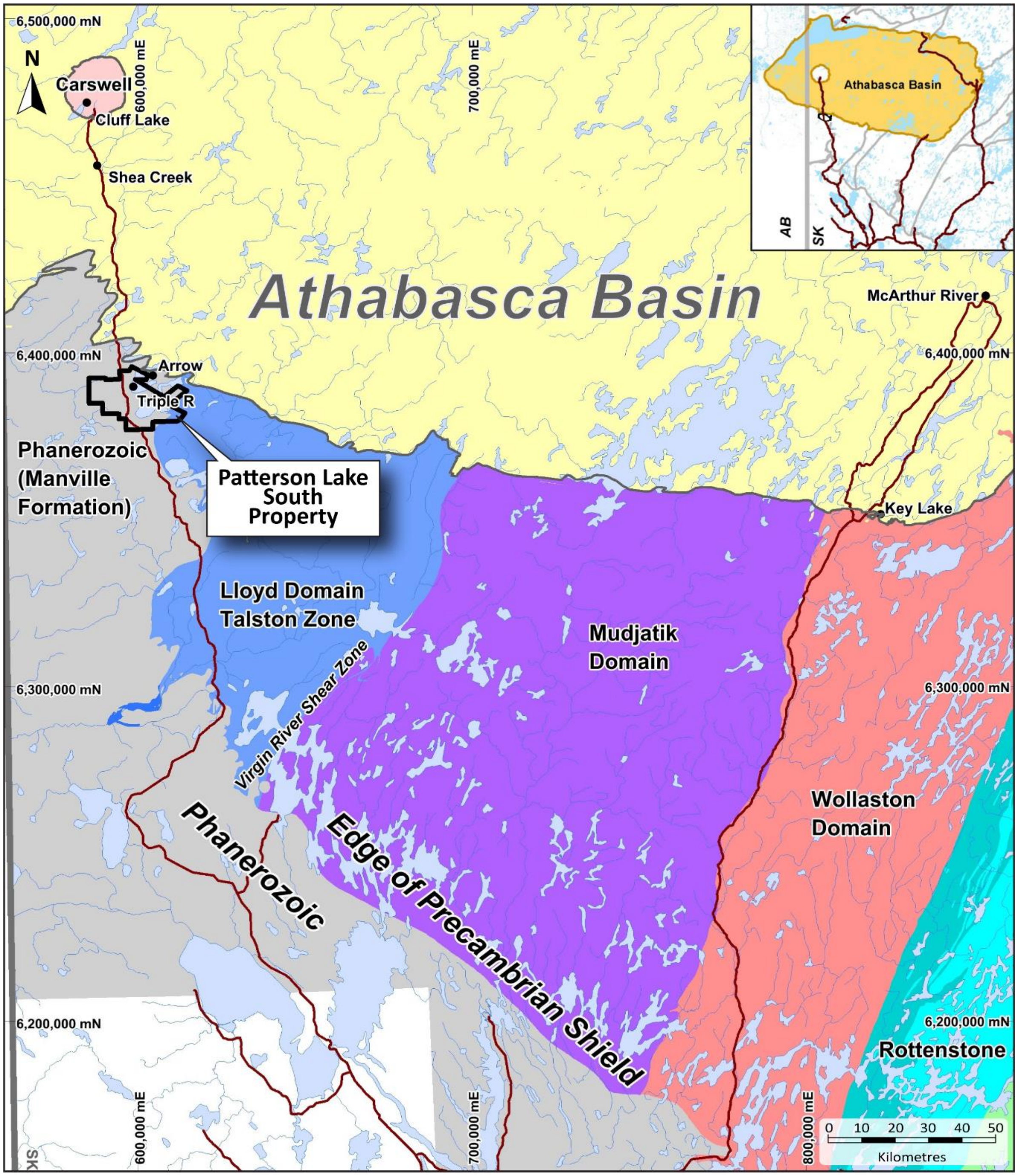
The pyroxene bearing orthogneiss comprises the core of a northeast trending belt of alteration and structural disruption along the PLG-3B EM conductor and dips steeply to the southeast. The orthogneisses are intensely sheared, faulted, and altered into an intercalated sequence of fine-grained ribbony graphite-sulphide gneiss and medium grained garnet porphyroblast gneiss with subordinate garnetite, graphitic mylonite, and cataclasite. The sheared graphitic rocks are collectively termed the Main Shear Zone (MSZ).

The MSZ is constrained to the north and south by QFBG-GN. The QFBG-GN is comprised of approximately 60% quartz and plagioclase, 20% biotite, 15% garnet, and trace pyrite, sillimanite, and graphite.

Lenses of dark green to black, sheared, fine grained granitoid intrude the QFBG-GN and MSZ along the extent of the PLG-3B drilled to date. The sheared granitoids are interpreted to be roughly concordant with the regional geology (i.e., steeply dipping to the southeast) except in the eastern R780E where a thick lens is interpreted to dip shallowly to the east.

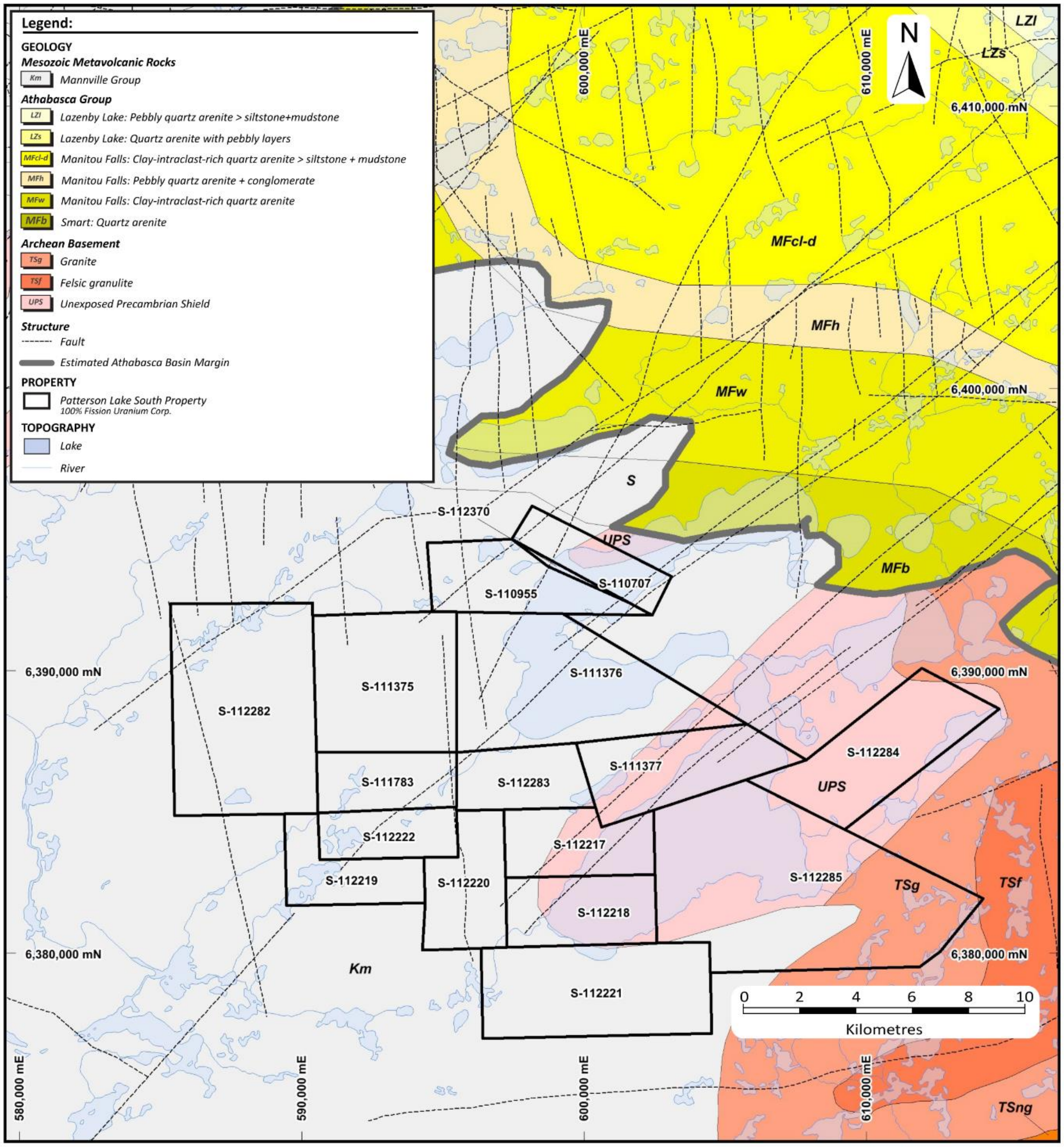
Away from mineralization, the basement rocks immediately in the PLS area are either paleoweathered, weakly altered, or fresh. The paleoweathered rock displays the typical downward gradational profile of a thin bleached and strongly kaolinite altered zone to a hematite dominated and then into a chlorite dominated zone. The paleoweathering profile can extend several m into the basement rock and completely alters the primary mineralogy to secondary clay minerals and quartz. Away from paleoweathered areas, later-stage hydrothermal alteration is common throughout the basement. In particular, a broad zone of alteration occurs around mineralization where fresh basement is rarely encountered. Dark green chlorite alteration of garnet, biotite, and Al-silicates along with fracture infill to disseminated graphite and whitish green clay alteration of feldspar is the most abundant type of basement alteration. Patchy pink to red hematite occurs in the basement lithologies and is often associated with elevated radioactivity. Similarly, patchy, blebby limonite alteration almost entirely occurs with moderate to strong intervals of radioactivity. Along the MSZ and hanging wall QFBG-GN contact, a broad zone of silicification almost completely overprints the QFBG-GN. This silicified unit was initially considered as quartzitic gneiss, but later was reinterpreted as a silicified version of the southern QFBG-GN based on textural observations and the gradational nature of the contact between the southern QFBG-GN and silicified zone.

On a regional scale, the paleotopography in the vicinity of the PLG-3B EM conductor is flat lying. The mineralized zones occur in slight basement topographic lows and are separated by relative highs. In the vicinity of the mineralized zones, the basement surface shows many small-scale offsets, which are interpreted to be caused by a series of stacked reverse faults. Based on processed oriented core data and closely spaced grid drilling, the dominant structural trends along the PLG-3B EM conductor appear to be steeply southeast dipping reverse faults and dextral strike-slip movement. Significant northeast and northwest trending faults interpreted from DC resistivity surveys crosscut the PLG-3B conductor and appear to be associated with broad, strong zones of uranium mineralization. These faults are yet to be positively identified in drill core as they are roughly parallel to the dominant drilling direction. Around zones of intense uranium mineralization microbreccia, dravite filled breccia, and graphitic cataclasite and mylonite occur; however, the intense alteration associated with uranium mineralization often makes these features difficult to identify.



Source: FCU, 2022

Figure 7-2: Local Geology



Source: FCU, 2022

Figure 7-3: Property Geology

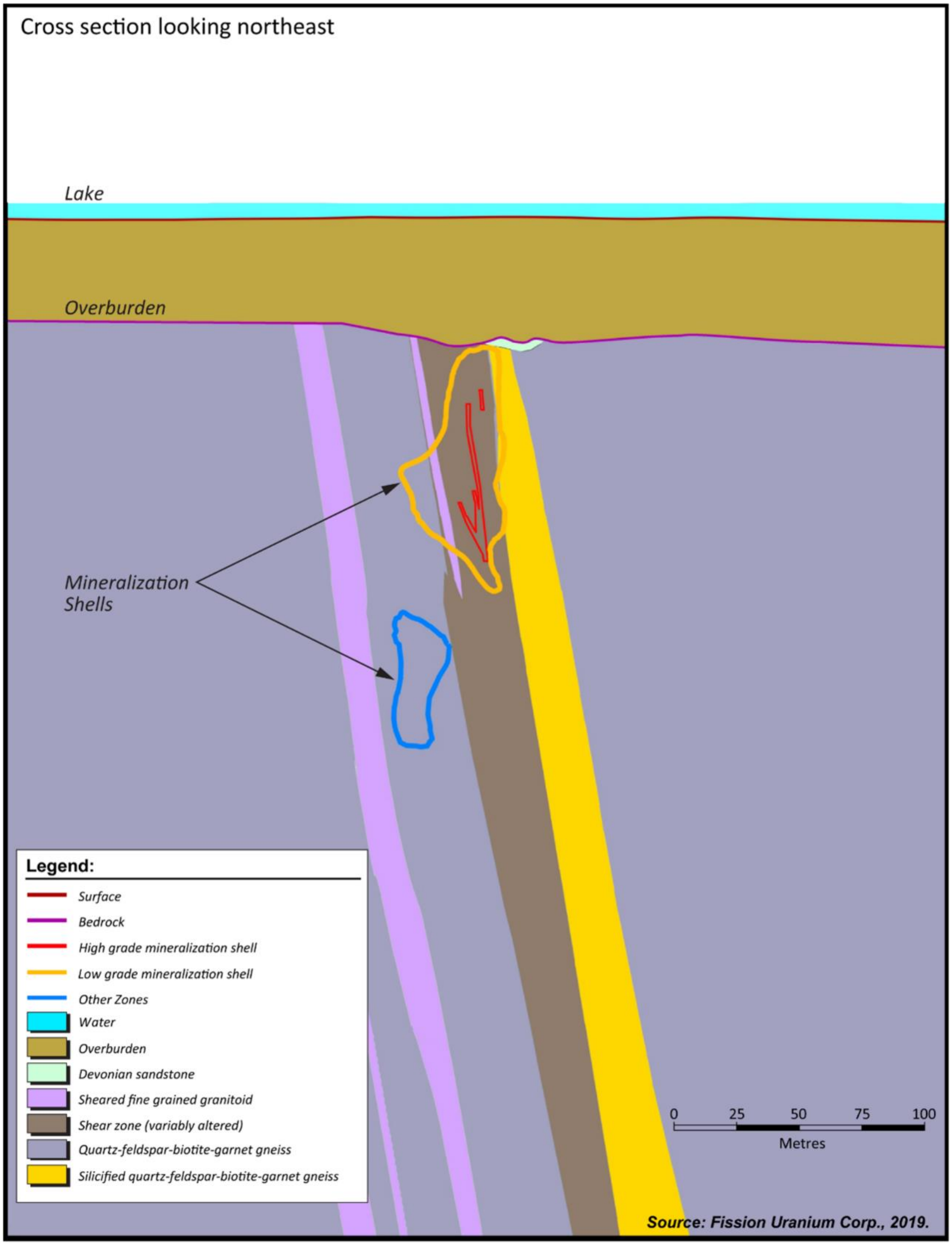


Figure 7-4: Idealized Cross Section

7.4 Mineralization

As of the effective date of this Technical Report, appreciable high-grade mineralization is known to occur on the PLS Property in five zones, which collectively make up the Triple R deposit. From west to east, these zones are: 1) R1515W, 2) R840W, 3) R00E, 4) R780E, and 5) R1620E, the most significant of which is the R780E zone, as depicted in Figure 7-5. Uranium mineralization discovered on the PLS Property to date is hosted primarily in basement lithologies with subordinate amounts intersected in the overlying Devonian sedimentary rocks. Mineralized zones occur within or near to the MSZ over a 3.17 km strike length along the PLG-3B EM conductor. The R1620E zone is currently defined by 23 drill holes and is located on the PLG-3C EM conductor, which is considered to be the eastern extension of the MSZ based on geology.

No significant uranium mineralization has been intersected in exploration drilling away from the PLG-3B and 3C conductors.

Parts of the following description of the mineralization on the PLS Property are taken from Mineral Services Canada Inc. (2014a) and revised after drilling to the end of 2018.

Uranium mineralization at the PLS Property is hosted primarily within metamorphosed basement lithologies and, to a much lesser extent, within overlying Meadow Lake Formation sedimentary rocks.

Mineralization within the Meadow Lake Formation sedimentary rocks typically occurs as fine-grained disseminations, sooty blebs, and rarely semi-massive uranium mineralization. Uranium concentrations within the Meadow Lake Formation are generally low to moderate, however, grades greater than 1.00% by weight (wt%) U_3O_8 have been intersected. When mineralized, the Meadow Lake Formation is typically strongly clay and chlorite altered, though locally can be pervasively hematite stained a deep red. Relative to basement hosted mineralization, only a very small amount of mineralized Meadow Lake Formation has been intersected on the PLS Property to date.

Basement hosted mineralization at the PLS Property occurs in a wide variety of styles, the most common of which appears to be fine-grained disseminated and fracture filling uranium minerals strongly associated with hydrocarbon/carbonaceous matter within the MSZ. Uranium minerals, where visible, appear to be concordant with the regional foliation and dominant structural trends identified through oriented core and fence drilling (i.e., steeply dipping to the southeast). Typically, mineralization within the MSZ is associated with pervasive, strong, grey-green chlorite and clay alteration. The dominant clay species identified through portable infrared mineral analyzer (PIMA) analysis are kaolinite and magnesium-chlorite interpreted to be sudoite. The pervasive clay and chlorite alteration eliminate the primary mineralogy of the host rock with only a weakly defined remnant texture remaining. Locally, intense rusty limonite-hematite alteration in the orthogneisses strongly correlates with high grade uranium mineralization and a “rotten”, wormy texture.

Less common styles of uranium mineralization within the MSZ, which are often associated with very high grade uranium, include: semi-massive and hydrocarbon rich, intensely clay altered (kaolinite) with uranium-hydrocarbon buttons, and massive metallic mineralization. These zones of very high grade mineralization generally occur along the contact of the MSZ and intensely silicified QFBG-GN and comprise a high grade mineralized spine. This spine may represent a zone of intense structural disruption which has been completely overprinted by alteration and mineralization, however, drill holes that undercut the strongly mineralized spine have failed to show signs of significant structural damage. Particularly well mineralized drill holes are often associated with thin swarms of dravite-filled breccia.

Uranium mineralization within the north and south QFBG-GN which bound the MSZ generally occurs as fine grained disseminations and is almost always associated with pervasive whitish-green clay and chlorite alteration with local pervasive hematite. The mineralized zones within the QFBG-GN are interpreted to be stacked structures parallel to the MSZ strike and dip along the PLG-3B conductor.

Results of the detailed mineralogical work at the PLS Property indicate that the dominant uranium mineral present is uraninite, with subordinate amounts of coffinite, possible brannerite and U-Pb oxide/oxyhydroxide. Uranium minerals occur mainly as anhedral grains and polycrystalline aggregates with irregular terminations; irregularly developed veinlets, locally showing extremely complex intergrowths with silicates; micrometric inclusions and dendritic intergrowths with silicates; and very fine-grained dissemination intercalated with clays. In the samples studied, uranium minerals also occur as fine-grained inclusions in carbonaceous matter (hydrocarbon).

7.4.1 R00E Zone

The R00E mineralized zone was the first mineralized zone discovered on the PLS Property and was intersected during the fall 2012 drill program. The sixth drill hole of the campaign, PLS12-022, was a vertical hole drilled from the western shore of Patterson Lake testing for the up-dip extension of the strong alteration and weak mineralization intersected in PLS12-016 (0.07% U_3O_8 over 1.0 m). PLS12-022 intersected a total of 12.5 m of uranium mineralization beginning at the top of bedrock (55.3 m) including a main zone averaging 1.1% U_3O_8 over 8.5 m from 70.5 m to 79.0 m.

The R00E zone is currently defined by 23 drill holes intersecting uranium mineralization over a combined grid east-west strike length of 120 m and a maximum grid north-south width of 50 m. Uranium mineralization at R00E trends north-easterly, in line with the MSZ.

At R00E, uranium mineralization is generally found within several m of the top of bedrock which occurs at a depth of 50 m to 60 m vertically from surface. Several holes (e.g., PLS13-037, PLS13-039) drilled along the southern edge of the mineralization have intersected the down dip uraniferous root over 100 m below the top of bedrock. Uranium mineralization at R00E is hosted within the MSZ, northern QFBG-GN, and Meadow Lake Formation sediments. No uranium mineralization has been intersected to date in the silicified hanging wall or in the southern QFBG-GN.

As the R00E zone had been interpreted to be roughly flat lying at the top of bedrock, vertical holes have dominantly been utilized to delineate mineralization. Vertical holes intersect the mineralized zone roughly perpendicular and therefore provide an approximate true thickness. Table 7-1 lists a selection of significant mineralized drill hole intersections at the R00E zone.

Drilling since the effective date of the previous Mineral Resource estimate did not affect the interpretation of the R00E zone; therefore, the resource model in that area has not changed.

Table 7-1: R00E Zone Significant Intersections

Drill Hole	Interval Length (m)	Average Grade[1] ($\%U_3O_8$)	GT[2] ($\%U_3O_8$ * m)
PLS13-059	20.50	8.36	171.38
PLS13-043	22.00	4.80	105.60
PLS13-079	17.50	5.98	104.65
PLS13-041	25.00	3.02	75.50
PLS13-052	16.49	3.89	64.15
PLS13-027	38.00	1.05	39.90
PLS13-049	17.00	2.10	35.70

Notes:
1. Average grades are based on uncut chemical assay values
2. GT – grade by thickness

7.4.2 R780E Zone

The R780E zone was discovered during the winter 2013 drill program with drill hole PLS13-038. PLS13-038 targeted an intense radon-in-water anomaly occurring along the PLG-3B conductor, approximately 390 m east of the PLS discovery hole. Drill hole PLS13-038 intersected a 34.0 m wide zone of very strong uranium mineralization, beginning at 87.0 m, averaging 4.9% U_3O_8.

The R780E zone is currently defined by 281 drill holes over a grid east-west strike length of 960 m and a maximum grid north-south width of 101 m. Similar to R00E, R780E mineralization trends approximately northeast, in line with the MSZ. Representative sections and plans from the R780E zone are provided in Section 14, Mineral Resources.

As with the R00E zone, R780E uranium mineralization has varying thickness, from tens of cm along the flanks to very wide intervals within the MSZ, as seen in PLS14-248 which intersected a lens of high grade uranium mineralization over 15 m in true thickness. In section view, R780E mineralization generally occurs as sub-vertically and southeast dipping zones, concordant with the regional dip. A very high grade spine of uranium mineralization occurs within the main zone and has been traced as a series of lenses across almost the entire strike length of the R780E zone. The high grade spine occurs adjacent to the contact between the MSZ and silicified QFBG-GN.

At the western R780E zone, uranium mineralization extends to near the top of bedrock. Moving eastward, the top of mineralization appears to be plunging at approximately -7° to the east. In general, the western R780E mineralization morphology is similar to the R00E, spatially restricted to the northern QFBG-GN, MSZ, and Meadow Lake Formation sediments. Moving eastward through the R780E zone, mineralization has been intersected within the MSZ, northern QFBG-GN, and Meadow Lake Formation sediments and, unlike the R00E zone, strong mineralization has been cored in the silicified QFBG-GN and southern QFBG-GN.

Initial drilling at the R780E zone consisted of only vertical holes for three main reasons: testing for subhorizontal mineralization similar to the R00E zone, limitations with the reverse circulation (RC) drill rig used to pre-case holes, and summer barge drilling where angled holes were not technically achievable. From drill hole PLS14-192, which was drilled during the winter 2014 campaign, onwards, the majority of drill holes at R780E were angle holes, mostly drilled south to north in order to best intersect the steeply south dipping mineralized lenses. The Mineral Resource estimate for the R780E zone has been updated with results of the winter 2021 drill program. Table 7-2 lists a selection of significant drill hole intersections at the R780E zone.

Table 7-2: R780E Zone Significant Intersections

Drill Hole	Interval Length (m)	Average Grade[1] (% U_3O_8)	GT[2] (% U_3O_8 * m)
PLS14-248	31.50	28.05	883.58
PLS13-075	60.49	20.26	1225.53
PLS14-129	21.00	28.05	589.05
PLS18-588	26.00	22.62	588.12
PLS14-201	13.05	26.47	345.43
PLS13-051	27.50	27.28	750.20
PLS14-215	30.99	11.49	356.08
PLS14-187	41.50	10.10	419.15
PLS13-053	27.45	22.60	620.37
PLS14-209	42.50	21.97	933.73
PLS13-080	18.48	19.01	351.30
PLS14-290	38.50	32.52	1252.02
PLS18-584	37.90	11.43	433.20

Note:
1. Average grades are based on uncut chemical assay values
2. GT – grade by thickness

7.4.3 R1620E Zone

The R1620E mineralized zone was discovered during the winter 2014 drill program with hole PLS14-196 which was testing a moderate radon-in-water anomaly along the PLG-3C EM conductor, interpreted to be the extension of the PLG-3B EM conductor. PLS14-196 intersected 28.5 m of uranium mineralization beginning at a depth of 100.0 m down hole, which averaged 0.17% U_3O_8.

The R1620E zone is currently defined by 23 drill holes. Uranium mineralization at the R1620E occurs in what is interpreted to be the eastern extension of the MSZ and appears to be associated with the MSZ – silicified QFBG-GN contact. Table 7-3 lists a selection of significant drill hole intersections at the R1620E zone. The R1620E zone was last drilled during the winter 2017 program; additional drilling is recommended.

Table 7-3: R1620E Zone Significant Intersections

Drill Hole	Interval Length (m)	Average Grade[1] (%U_3O_8)	GT[2] (%U_3O_8*m)
PLS16-500	14.00	9.53	133.42
PLS16-460	27.50	3.79	104.23
PLS16-485	10.00	9.75	97.50
PLS16-498	25.50	3.74	95.37
PLS16-464	23.14	6.59	152.49
PLS16-496	15.40	6.45	99.33
PLS16-489	12.00	2.30	27.60
PLS17-518	16.48	1.09	17.96
PLS17-531	9.49	0.74	7.02
PLS16-487	18.50	0.46	8.51
PLS14-196	28.50	0.17	4.85

Notes:
1. Average grades are based on uncut chemical assay values
2. GT – grade by thickness

7.4.4 R840W Zone

The R840W (formerly known as R600W) mineralized zone, located 840 m west of R00E, was discovered during the summer 2013 exploration drill program. PLS13-116 was an angle hole drilled to the north, targeting a radon-in-soil anomaly along the western end of the PLG-3B conductor. The drill hole intersected a thin zone of anomalous radioactivity hosted in the northern QFBG-GN. Follow-up drilling during the 2015 winter program intersected high grade uranium mineralization in drill hole PLS15-352 returning 31.5 m averaging 11.09 wt% U_3O_8.

The R840W zone is currently defined by 91 drill holes with a total grid east-west strike length of 425 m. Similar to the R00E and R780E zones, mineralization trends north-easterly in line with the MSZ. Table 7-4 lists a selection of significant drill hole intersections at the R840W zone. Additional drilling is recommended. Drill holes intersecting the near vertical mineralized zones at shallow angles or nearly parallel the mineralization do not reflect the true thickness of the fractures which range from 10 m to 20 m wide.

Table 7-4: R840W Zone Significant Intersections

Drill Hole	Interval Length (m)	Average Grade[1] (%U_3O_8)	GT[2] (%U_3O_8 * m)
PLS15-439	20.41	15.96	325.74
PLS15-343	26.50	15.30	405.45
PLS16-504	10.50	12.25	128.63
PLS17-517	51.00	1.89	96.39
PLS16-512	54.00	1.39	75.06
PLS17-515	23.00	2.64	60.72
PLS21-613	26.50	3.56	94.34
PLS21-624	46.00	8.01	368.46

Notes:
1. Average grades are based on uncut chemical assay values
2. GT – grade by thickness

7.4.5 R1515W Zone

The R1515W mineralized zone was discovered during the winter 2017 drill program with hole PLS17-539 located 500 m west of R840W. The R1515W zone is currently defined by 25 drill holes with the best mineralized intersection returned in PLS17-564 which cored 14.5 m of uranium mineralization averaging 3.39 wt% U_3O_8. Uranium mineralization at the R1515W occurs in the MSZ and appears to be associated with the MSZ – silicified QFBG-GN contact. Table 7-5 lists a selection of significant drill hole intersections at the R1515W zone. The R1515W zone was last drilled during the winter 2018 program; additional drilling is recommended.

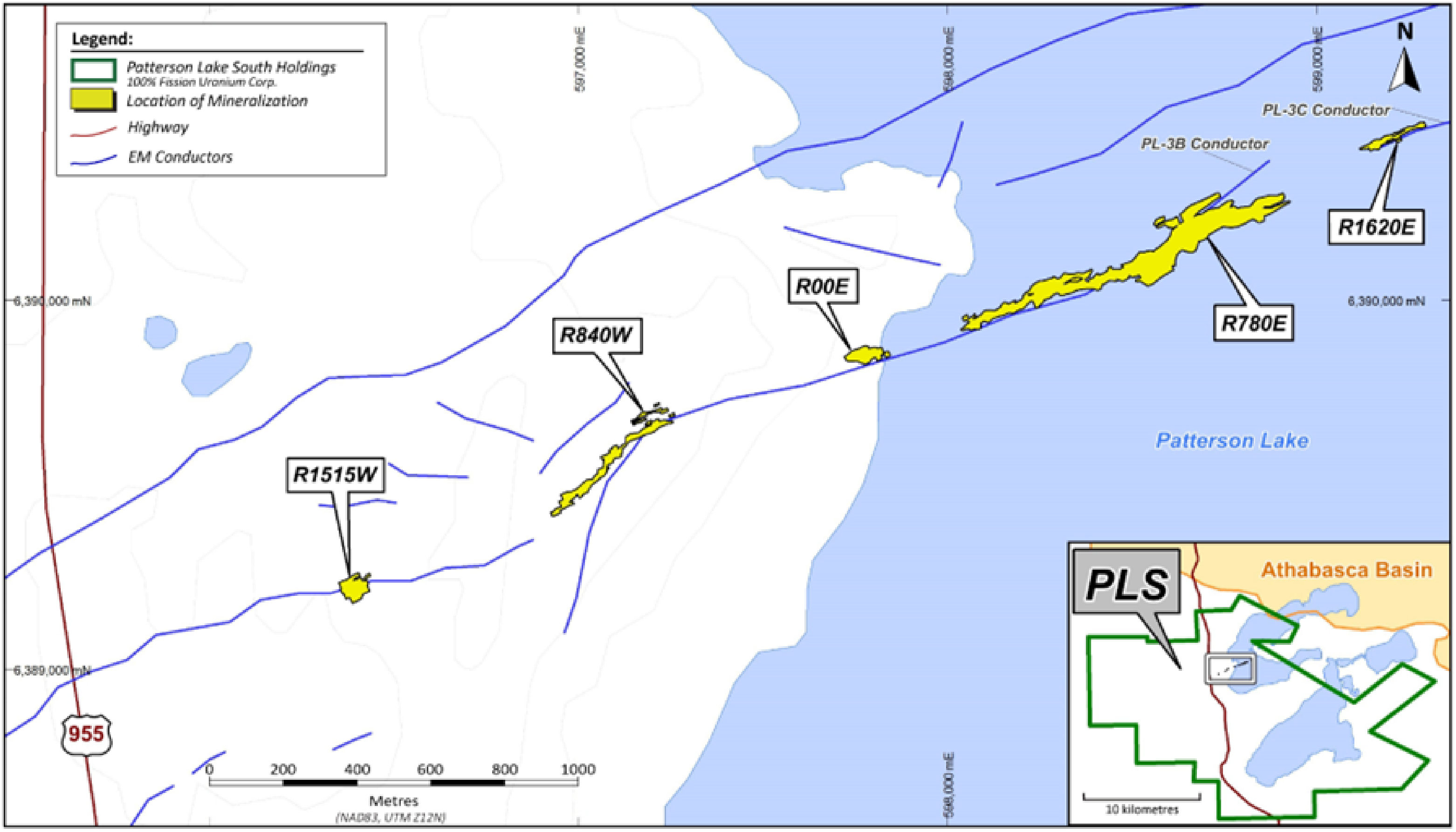
Table 7-5: R1515W Zone Significant Intersections

Drill Hole	Interval Length (m)	Average Grade[1] (%U_3O_8)	GT[2] (%U_3O_8 * m)
PLS17-557	42.95	1.17	50.25
PLS17-553	28.99	1.72	49.86
PLS17-566	49.08	2.06	101.10
PLS18-571	27.98	2.96	82.82
PLS18-572	22.49	1.66	37.33
PLS18-574	17.17	2.72	46.70
PLS18-569	30.47	3.59	109.39
PLS17-561	15.00	1.74	26.10
PLS17-562	27.65	1.90	52.54
PLS17-560	45.99	0.64	29.43

table continues…

Drill Hole	Interval Length (m)	Average Grade[1] ($\%U_3O_8$)	GT[2] ($\%U_3O_8$ * m)
PLS17-563	29.52	1.40	41.33
PLS18-577	15.50	0.73	11.32
PLS18-578A	24.98	0.26	6.49
PLS18-570	35.00	0.71	24.85
PLS17-539	14.53	0.39	5.67

Notes:
1. Average grades are based on uncut chemical assay values
2. GT – grade by thickness



Source: FCU, 2022

Figure 7-5: Location of Target Areas

8.0 DEPOSIT TYPES

The Triple R deposit is considered to be an example of a basement hosted vein-type or fracture-filled uranium deposit.

At numerous locations in Saskatchewan as well as some in Alberta, uranium deposits have been discovered at, above, and below the Athabasca Group unconformity. Mineralization can occur hundreds of m into the basement or can be perched up to 100 m above in the sandstone. At Triple R, relatively minor amounts of uranium have been identified in the overlying Devonian sediments and mineralization has been discovered in the basement at depths ranging from immediately at or just below the unconformity to 400 m below it. Typically, uranium is present as uraninite/pitchblende which occurs as veins and semi-massive to massive replacement bodies. In most cases, mineralization is also spatially associated with steeply dipping, graphitic basement structures that have penetrated into the sandstones and offset the unconformity during successive reactivation events. Such structures are thought to represent both important fluid pathways as well as chemical/structural traps for mineralization through geologic time as reactivation events have likely introduced further uranium into mineralized zones and provided a means for remobilization.

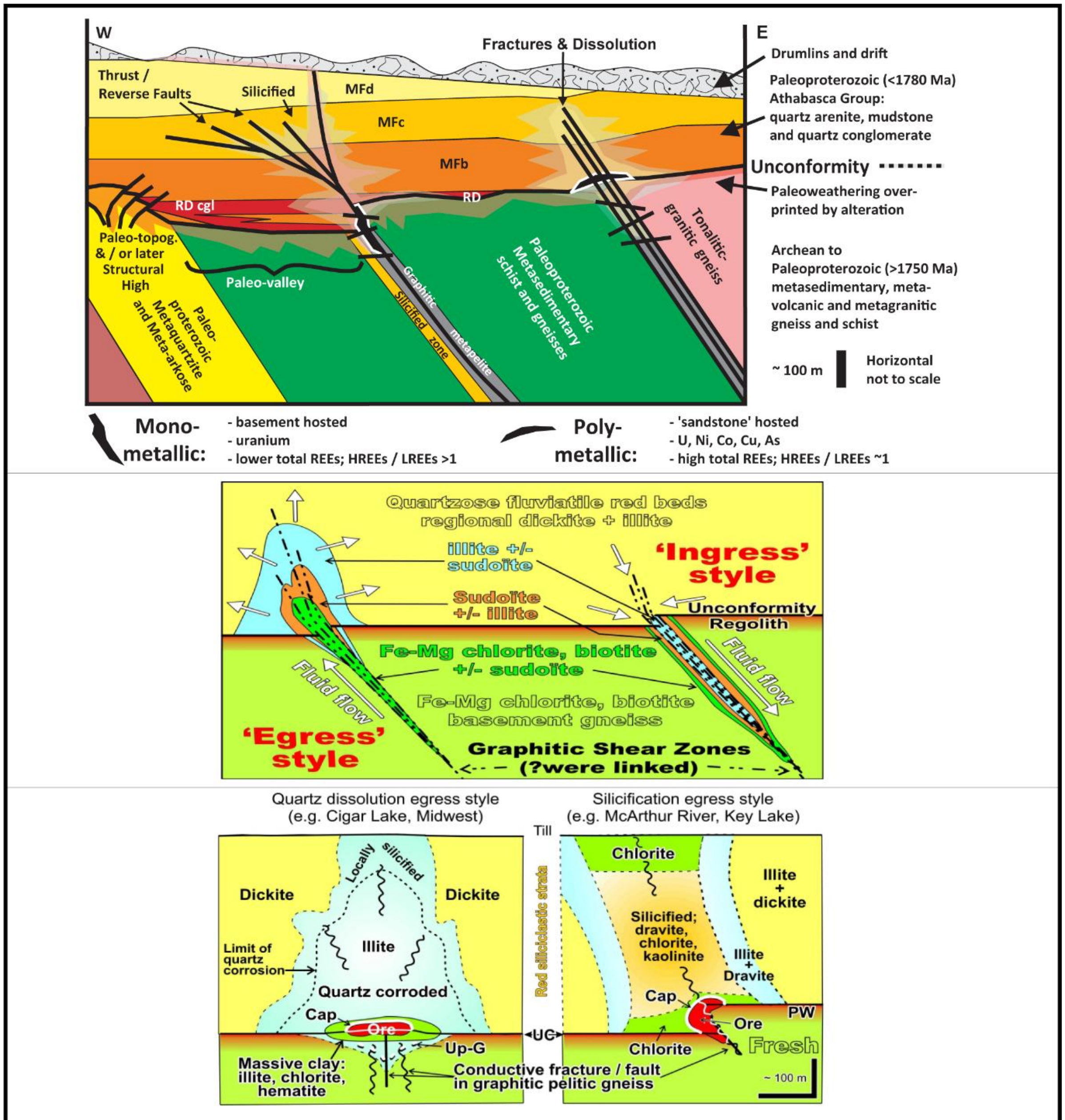
Unconformity-associated uranium deposits are pods, veins, and semi-massive replacements consisting of mainly uraninite, close to basal unconformities, in particular those between Proterozoic conglomeratic sandstone basins and metamorphosed basement rocks. Prospective basins in Canada are filled by thin, relatively flat-lying, and apparently un-metamorphosed but pervasively altered, Proterozoic (~1.8 Ga to <1.55 Ga), mainly fluvial, redbed quartzose conglomerate, sandstone, and mudstone. The basement gneiss was intensely weathered and deeply eroded with variably preserved thicknesses of reddened, clay-altered, hematitic regolith grading down through a green chloritic zone into fresh rock. The basement rocks typically comprise highly metamorphosed interleaved Archean to Paleoproterozoic granitoid and supracrustal gneiss including graphitic metapelite that hosts many of the uranium deposits. The bulk of the U-Pb isochron ages on uraninite are in the range of 1,600 Ma to 1,350 Ma. Mines comprise various proportions of two styles of mineralization. Monometallic, generally basement-hosted uraninite fills veins, breccia fillings, and replacements in fault zones. Polymetallic, commonly subhorizontal, semi-massive replacement uraninite forms lenses just above or straddling the unconformity, with variable amounts of uranium, nickel, cobalt, and arsenic, and traces of gold, platinum-group elements, copper, rare-earth elements, and iron.

Fundamental aspects of the Athabasca unconformity-type uranium deposit model are reactivated basement faults and two distinct hydrothermal fluids. Typically rooted in basement graphitic gneiss, brittle reactivated faults are manifest upward with brittle expression through the overlying sandstones and provide plumbing for the requisite mineralizing system. One of the necessary fluids is reducing, originates in the basement, and is channelled along basement faults.

Two end-members of the deposit model have been defined (Quirt 2003). A sandstone-hosted egress-type (e.g., Midwest A) involved the mixing of oxidized, sandstone brine with relatively reduced fluids issuing from the basement into the sandstone. Basement-hosted, ingress-type (e.g., Triple R, Rabbit Lake) deposits formed by fluid-rock reactions between oxidizing sandstone brine entering basement fault zones and the wall rock. Both types of mineralization and associated host rock alteration occurred at sites of basement-sandstone fluid interaction where a spatially stable redox gradient/front was present. Although either type of deposit can be high grade, with a few percent to 20% U_3O_8, they are not physically large. In plan view, the deposits can be 100 m to 150 m long and a few m to 30 m wide and/or thick. Egress-type deposits tend to be polymetallic (U-Ni-Co-Cu-As) and typically follow the trace of the underlying graphitic gneisses and associated faults, along the unconformity. Ingress-type, essentially monomineralic uranium deposits, can have more irregular geometry.

Unconformity-type uranium deposits are surrounded by extensive alteration envelopes. In the basement, they are relatively narrow but become broader where they extend upwards into the Athabasca Group for tens to even 100 m or more above the unconformity. Hydrothermal alteration is variously marked by chloritization, tourmalinization (high boron, dravite), hematization (several episodes), illitization, silicification/de-silicification, and dolomitization (Hoeve 1984).

Figure 8-1 illustrates various models for unconformity-type uranium deposits of the Athabasca Basin.



Source: From Jefferson et al.,2007.

Figure 8-1: Illustrations of Various Models for Unconformity Type Deposits of the Athabasca Basin

9.0 EXPLORATION

With the exception of drilling, exploration work performed on the PLS Property by Fission Energy, ESO, and their successor companies since 2007 is summarized in this section. Work completed on the PLS Property and its immediate vicinity by other parties prior to 2007 is summarized in Section 6 of this Technical Report. Drilling completed on the PLS Property since 2011 is summarized in Section 10 of this Technical Report.

9.1 Radon and Ground Radiometric Surveys

9.1.1 2008 Radon and Radiometric Surveys

From early to mid-October 2008, a preliminary electret ion chamber (EIC) radon detection survey consisting of 280 sample locations on the northernmost portion of the PLS Property was completed by RadonEx Ltd. (RadonEx). A radiometric gamma survey was done concurrently with the radon survey. Sample locations were spaced 200 m apart along four east-west running lines. Locations were 100 m apart along Highway 955 and both branching four-wheel drive roads. Up to five tightly spaced sample locations were completed for each CanOxy alphameter anomaly on the PLS Property. Step out and confirmation sample locations were completed as time allowed. Radon sampling was not conducted during or within 24 hours of a precipitation event.

Radon and radiometric values were generally low across the PLS Property (Armitage 2013).

9.1.2 2011 Radon and Radiometric Surveys

Throughout June 2011, a radon survey consisting of 462 sample locations on two grids was completed. A radiometric total count gamma-ray survey was carried out concurrently with the radon survey. Sample locations were spaced at 100 m intervals along north-south oriented lines, which were spaced 200 m apart. Grids 1 and 2 are located west and east of Highway 955, respectively. Radon sampling was not conducted during or within 24 hours of a precipitation event.

Radon values show strong anomalies related to the historical CanOxy alphameter anomalies and the 2009 airborne radioactive hotspots on Grid 1. Strong radon anomalies are associated with historical CanOxy EM conductors on Grid 2.

Three sample locations of interest are located in the northwest corner of Grid 1, away from the bulk of coincident radon and radiometric anomalies found in the south half of Grid 1.

The southeast corner of Grid 2 shows radon and radiometric anomalies south of the EM conductors. There are five radiometrically anomalous sample locations (PR11-404 to 408) in a column with only one of these locations (PR11-407) having strongly anomalous radon values. East of this anomalous radiometric column, sample location PR11-420 shows anomalous radon (1.65 pCi/m^2/sec) with a low radiometric value (50 cps) (Ainsworth 2011b).

9.1.3 2013 Radon and Ground Radiometric Surveys

During January and February 2013, RadonEx conducted an EIC radon in lake water (radon-in-water) and radon in lake sediment (radon-in-sediment) survey on the PLS Property (Charlton, Owen, and Charlton 2013a). Time-domain EM (TDEM) and versatile time-domain (VTEM) conductors with coincident resistivity lows located along strike of the discovery hole PLS12-022 were targeted. Station spacing was 20 m on 60 m north-south oriented lines within four main areas across Patterson Lake. A total of 186 radon-in-water and 167 radon-in-sediment samples were collected.

In Areas 1 and 2, the western side of the survey, an east-west to east-northeast–west-southwest (ENE-WSW) trend appears in both sets of data. In Areas 3 and 4, the eastern side of the survey, the correlation between sediment and water results is less evident, and results in these areas were generally lower than in the western section of the lake.

During April 2013, RadonEx conducted additional EIC radon-in-water and radon-in-sediment surveying on Patterson Lake (Charlton, Owen, and Charlton 2013b). Station spacing was generally 20 m and line spacing was generally 60 m. This survey was intended to infill areas from a previous radon-in-water and sediment survey, and to extend the coverage. A total of 151 sediment samples and 220 water samples were collected in and around the R780E zone.

Most of the sediments collected were fine sand with small pebbles and small amounts of organic matter. Two areas were characterized by sediments with high iron content and pebbles with iron nodules, namely, the southwest portion of the survey area, where the highest concentration of anomalous radon readings is located, and the northeast portion of the survey area, where a few moderately anomalous readings were collected during the February 2013 radon survey. Iron enrichment in the northeast portion of the survey area is much less prominent than in the southwest portion of the grid.

A clear ENE-WSW trend in the radon-in-water results is coincident with the strong VTEM conductor and with the Triple R deposit. The trend also appears in the radon-in-sediment results to a lesser degree.

During August 2013, an EIC radon detection survey consisting of 434 sample locations was completed by RadonEx. A radiometric gamma survey was performed concurrently with the radon survey. Samples were located at 10 m intervals. Survey lines were from 100 m to 450 m in length and spaced from 10 m to 40 m.

The survey area extended approximately 700 m westward from discovery diamond drill hole PLS12-022 on the west shore of Patterson Lake and was conducted to locate any additional mineralization down-ice and westward of the known mineralized zone.

Results suggested generally moderate variations in radon flux measurements across the survey area. Measurements appeared to increase towards the north end of the two north-reaching extension lines.

9.1.4 2014 Radon Surveys

From January to March 2014, RadonEx conducted additional EIC radon-in-water and radon-in-sediment surveying on the PLS Property (Charlton, Owen, and Charlton 2014). The surveys covered four separate areas: three on Patterson Lake and one on nearby Forrest Lake. In total, the surveys consisted of 2,610 radon-in-water sample stations and 266 radon-in-sediment sample stations. Station spacing was generally 20 m and line spacing was generally 60 m, with some line spacing at 30 m. The survey was intended to locate radon anomalous zones and trends along previously located geophysical conductor corridors interpreted from TDEM and VTEM surveys.

At Area A, covering the area of the mineralized zone and the primary conductive corridor, a series of discontinuous radon trends is evident, and 11 radon-in-water anomalies and trends were chosen for potential drill testing. The top ten Area A radon-in-water results compare well with the R780E Zone radon-in-water results from 2013. A discordant set of radon anomalies is suggestive of east-southeast striking cross-faulting.

At Area B, in the northeastern section of Patterson Lake, two parallel radon trends are recognized, of which the north one is very strong and appears to correspond to a conductor axis. Radon trends are suggestive of north trending cross-faulting through the grid area.

The Area C radon coverage in the southwest part of Patterson Lake reveals two anomalous parallel radon trends, which partially correlate to conductors. Area C radon-in-water results compare very favourably with the 2013 R780E results. A north-trending fault is interpreted to displace and reorient the radon trends.

Area D is a large irregular grid covering northern parts of Forrest Lake. Water depths are much greater here, particularly in the D-2 area (>70 m), where the bottom is covered with a thick layer of organics. Radon signatures are masked and muted in this part of the lake and no radon targets are identified at D-2.

In the D-1 area to the northeast, where the lake is shallower, five very high radon-in-water anomalies were found, including some of the highest radon-in-water results yet recorded on the PLS Property.

During August 2014, Remote Exploration Services (Pty) Ltd. (RES) conducted a RadonX™ radon cup survey over the R600W Zone (now part of R840W) at the PLS Property (RES 2014a). In total, 580 cups were deployed in a grid with 20 m line spacing and 10 m cup spacing along line. The total area of the grid was 0.11 km^2. The survey was conducted in order to compare and confirm results from the 2013 RadonX™ radon cup surveying over the same grid area.

The survey results confirmed zones of anomalous and highly anomalous radon flux values (RnV) that, in general, are centred on or slightly to the north of the main ENE-WSW trending EM conductor that is associated with the mineralization. The orientation of this EM conductor parallels the interpreted strike of major fault structures in the area. Faults are known conduits for radon gas emanating from uraniferous mineralized bodies.

The western zone of anomalous RnV correlates with a delineated mineralized zone defined from drilling. Additionally, there is a northwest trend of slightly anomalous to anomalous RnV that intersects the north-northeast trend and could represent subordinate structures in this direction.

During October 2014, RES conducted a radon cup survey over three separate areas east of Forrest Lake, approximately 10 km southeast of the Triple R deposit (RES 2014b). In total, 867 cups were deployed. The grids consisted of 30 m line spacing and 20 m cup spacing along each line. The total area of the three grids encompassed 0.481 km^2.

The three grids targeted high priority conductors identified by airborne VTEM surveying and/or ground TDEM surveying, namely the PLV-68A conductor (Grid S1), the PLV-63D conductor (Grid S3), and the PLV-63C conductor (Grid S4). Areas and trends of anomalous radon flux measurements were observed on each of the three grids.

A helium-hydrogen-neon soil gas survey consisting of 110 stations was conducted by Petro-Find Geochem Ltd. in October 2014. The survey provided coverage along trend to the east and over top of the R600W zone, and was also designed to duplicate previous radon-in-soil measurement locations. Helium anomalies coincided with the R600W zone mineralization and with at least one prominent radon gas anomaly to the north.

9.2 Airborne Surveys

9.2.1 2007 MEGATEM Magnetic and Electromagnetic Survey

During November 2007, prior to the execution of the PLS joint venture between Fission Energy and ESO, Fission Energy and ESO completed a fixed wing combined electromagnetic (MEGATEM) and magnetic airborne survey over their respective mineral claims: S-110954 and S-110955 (Fission Energy) and S-110707 and S-110723 (ESO). The results of the survey were of very low resolution (Armitage 2013).

9.2.2 2009 Airborne Magnetic and Radiometric Survey

In mid-October 2009, Special Projects Inc. (SPI) completed a combined fixed wing light detection and ranging (LiDAR), radiometric, and high resolution airborne magnetic geophysical survey over the northern portion of the PLS Property totalling approximately 3,342 line-km. Flight lines were oriented at 135° and were spaced at 50 m intervals. The aeromagnetic survey successfully delineated different basement lithologies. A structural interpretation was completed which identified the traces of surface and basement faults, shear zones, and areas of structural complexity (McElroy and Jeffrey 2010). The airborne radiometric spectrometer survey outlined a number of uraniferous hot-spots within a 3.9 km long by 1.4 km wide area, which was subsequently found to be the result of a radioactive boulder field that contained boulders composed of massive or semi-massive uranium oxide minerals. This radioactive area extended south of claim S-111375, which led to the staking of claim S-111783 in April 2010.

9.2.3 2012 Geotech Magnetic and Electromagnetic Survey

In mid-February 2012, Geotech Ltd. completed a detailed, combined helicopter-borne versatile time-domain electromagnetic (VTEMplus) survey with Z and X component measurements and a horizontal magnetic gradiometer survey over the entirety of the PLS Property. Flight lines totalling 1,711.3 line-km and oriented at 135° were flown at 200 m line spacing.

The survey was instrumental in defining conductive packages over the PLS Property. Figure 9-1 illustrates the results of the survey.

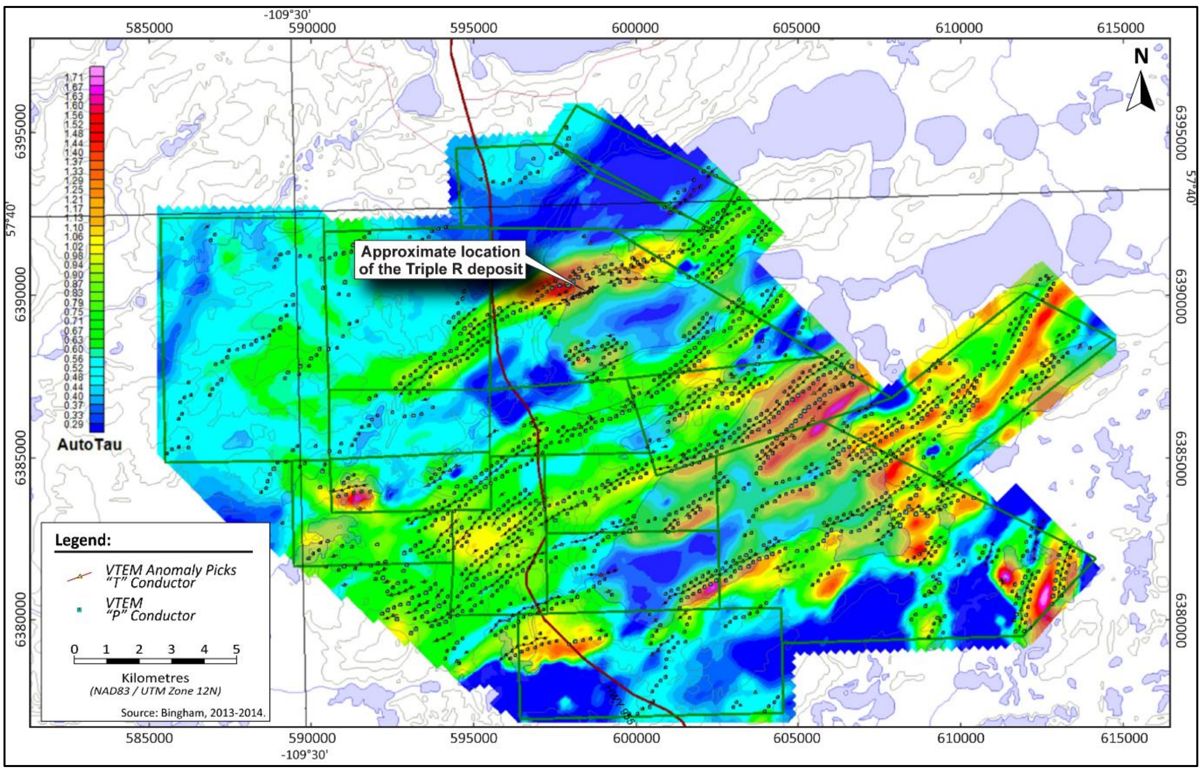


Figure 9-1: 2012 VTEM Interpretation

9.2.4 2012 Airborne Radiometrics and Magnetic Survey

From mid- to late September 2012, SPI completed a combined fixed wing LiDAR, radiometric, and magnetic survey over the southern portion of the PLS Property totalling 5,611.5 line-km, of which 5,147.3 line-km were flown within the PLS Property boundary. The flight lines were oriented at 126° and were spaced at 50 m intervals.

The data was merged with the previous 2009 SPI high resolution survey to create a seamless magnetic grid over the PLS Property area.

From the analysis of the field data, it was apparent that the geological setting of the PLS Property area is complicated and that there are numerous lineaments related to contacts and structures between basement units.

The PLS Property area has several predominant trends. The survey area is divided into three magnetic zones: a central zone (A) of relatively low magnetism characterized as predominantly northeast magnetic trends (conforming to the general domain orientation of the Athabasca Basin), a western zone (B) of relatively high magnetism with predominant northwest magnetic trends, and an eastern zone (C) of low magnetism with predominant north-northeast trends (Bingham 2012).

Figure 9-2 illustrates the results of the merged, processed magnetic data and the three magnetic zones as interpreted by Bingham (2012).

In April 2014, SPI was commissioned to survey two blocks over the Triple R deposit and over part of the Forrest Lake conductor trend. The blocks were flown with orthogonal line directions and 50 m line spacing. The purpose of the survey was to provide a more detailed magnetic grid for better definition of structures, lithology, and magnetite depletion. Total survey coverage was 2,136 line-km.

During October 2014, Eagle Mapping Ltd. was contracted to obtain high resolution airborne LiDAR survey data from a 154 km^2 area encompassing the known mineralization.

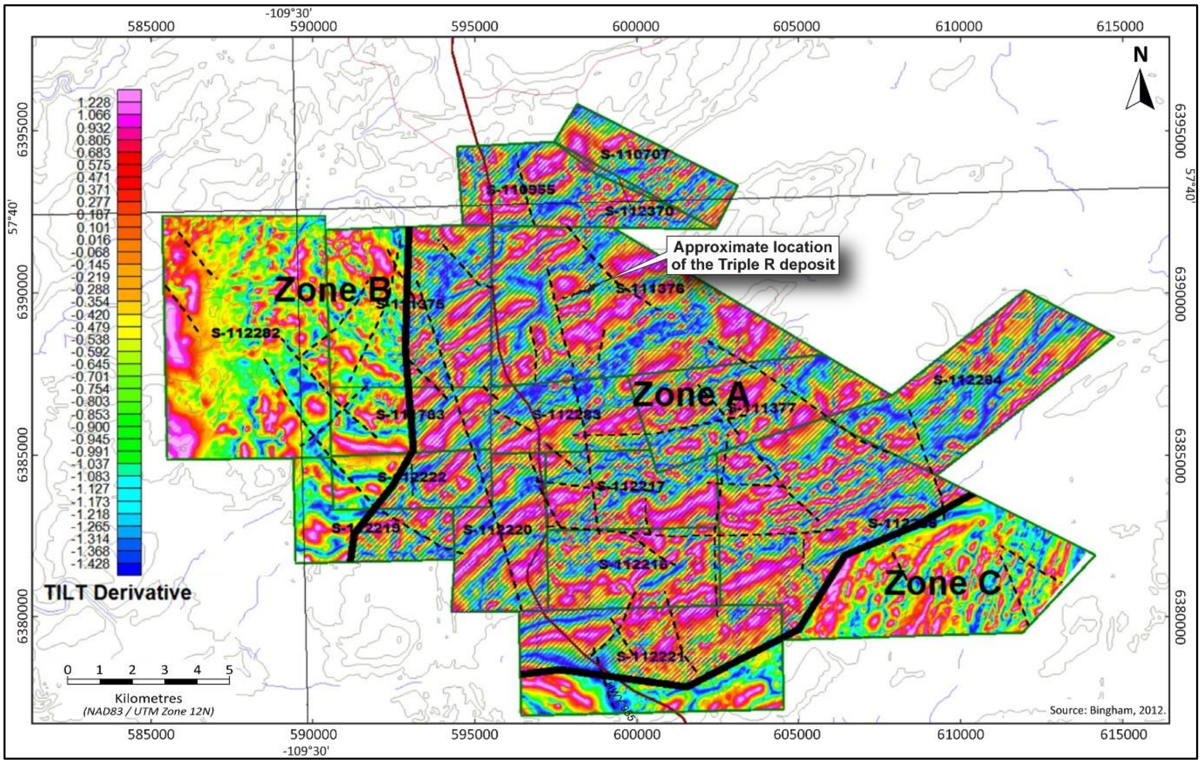


Figure 9-2: Interpreted Major Structures and Tilt Derivative Magnetics

9.2.5 2016 HELISAM Airborne Survey

Discovery Int'l Geophysics Inc. (Discovery) in partnership with Gap Geophysics Australia Pty Limited (GAP) was commissioned by FCU to conduct geophysical surveys using GAP's proprietary sub-audio magnetics (SAM) technique utilizing a helicopter receiver platform (HeliSAM) and a ground receiver platform (GroundSAM). The surveys took place between February 26 and March 18, 2016 (Discovery and GAP 2016).

The purpose of the survey was designed to assess the impact of the use of a high power geophysical source in increasing the geophysical signal and thus increase the signal to noise ratio of the response for MMC surveying. Additionally, it aimed to assess the high-resolution advantages of using the SAM total field B-field sensor with low base frequency excitation for resistivity surveying (equivalent MMR or MMC) in the Athabasca basin and evaluate the performance of the HeliSAM system as an alternative to more expensive ground DC resistivity surveys for the detailed resolution of low-resistivity alteration zones associated with uranium mineralization.

The initial scope specified that GAP was to utilize its SAM technology to survey one large grid laid out along the Patterson Lake conductive corridor. Two additional survey grids were later added to the scope, with smaller dipole separation and alternate dipole placement to focus current flow.

The total coverage collected via HeliSAM equates to 393.4 line-km, while the GroundSAM method totalled 45.1 line-km. Approximately 2.0 km of overlapping lines were collected between the E_ext grid HeliSAM and GroundSAM portions to tie the grid together.

The deliverables for the project were as follows:

- Located total field electromagnetic (TFEM) data extracted from transmitter off-time.

- Magnetometric Conductivity (MMC) data extracted from transmitter on-time.
- Located Total Magnetic Intensity data.

9.3 Trenching and Boulder Surveys

Several trenching and boulder surveys have been carried out on the PLS Property since 2011. Results are compiled in Figure 9-3.

9.3.1 June 2011 Boulder Prospecting

In June 2011, 89 radioactive hotspots from the 2009 airborne radiometric survey were investigated on the ground. The radioactive hotspots were spread out over an area of approximately 3.9 km long by up to 1.4 km wide that trended north-northeast to south-southwest.

Eight soil samples were also taken (PS11-01 to PS11-08), with only one of these samples having off-scale radioactivity.

Based on this small sample set, the strong pathfinder elements for the high grade uranium oxide include Au, B, Co, Cr, Cu, Li, Mo, Pb, Sb, Sr, Th, W, Zr, and most rare earth elements (REE). Nickel was not found to be a strong pathfinder element (Ainsworth 2011b).

9.3.2 October 2011 Trenching and Boulder Prospecting

From mid- to late October 2011, a program consisting of trenching and boulder prospecting was completed on mineral claims S-111375, S-111376, and S-111783.

A total of 18 trenches were excavated to assess the uraniferous boulder field that had been discovered in June 2011. The uraniferous boulders lie between two major terminal moraines of the Cree Lake Moraine. The trenches were located on three lines traversing the terrain in the up-ice direction. These trenches covered the region from the westernmost moraine to the northeast where surficial material bearing uraniferous boulders is overlain by non-radioactive overburden. The trenches were located on the ground using a handheld Garmin GPS unit.

A total of 25 soil samples and 21 boulder samples were recovered from the trenches.

The magnetic susceptibility of the materials was measured in trenches using an Exploranium KT-9 Kappameter. In general, the magnetic susceptibility of the surficial materials is much lower, less than 0.5 x 10^{-3} SI units, than in rock.

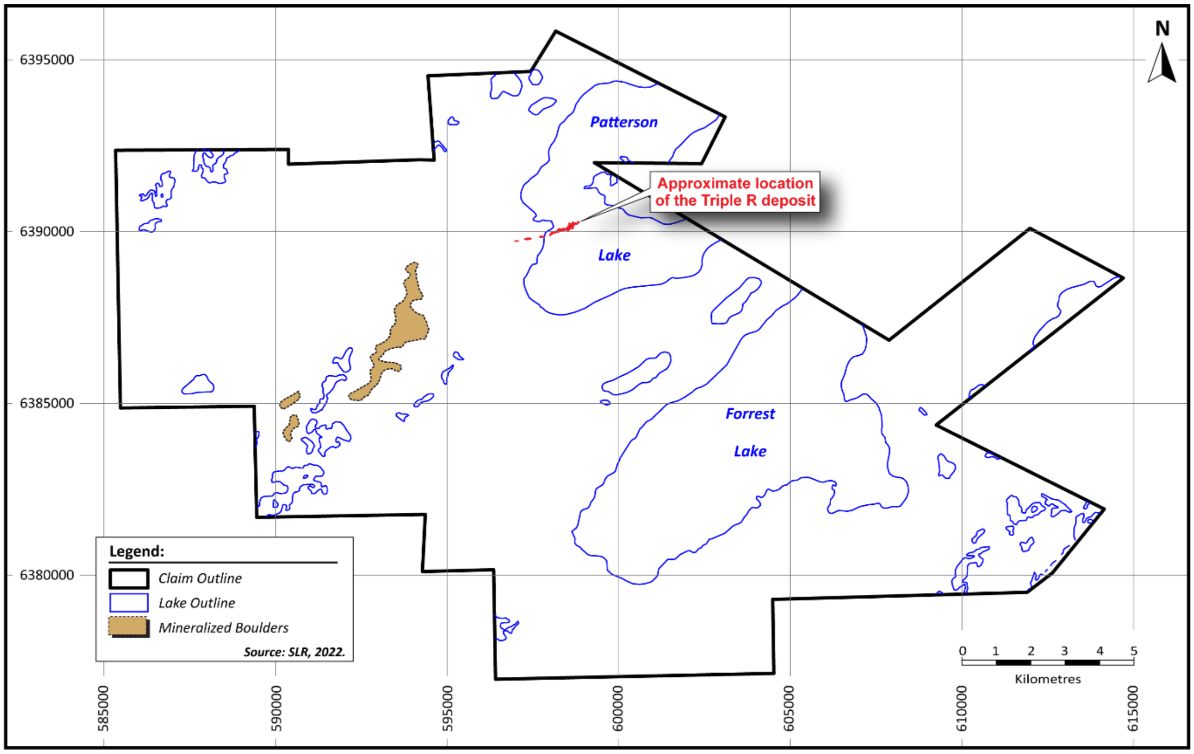


Figure 9-3: Location of Mineralized Boulders

An Exploranium GR-110 scintillometer was used to measure radioactivity. If a strongly radioactive area was found near the profile, the profile readings were located away from that area or otherwise recorded in the notes. In general, the radioactivity reflected the stratigraphy more strongly than the magnetic susceptibility, however, this may be a result of the values occurring over a wider range.

A total of 25 soil samples were recovered from trenches PT11-01 to PT11-16. Maximum radiometric values of the in-situ soil samples ranged from 80 cps to 2,418 cps. Uranium-in-soil values ranged from below detection limits (less than 2 ppm U) to 336 ppm. All samples identified as non-radioactive assayed below detection limits, and all soils identified as radioactive assayed above detection limits, indicating a correlation between radioactivity and uranium values.

Eight boulders were found in trench PT11-08, three were found in trench PT11-06, two were found in each of trenches PT11-03, PT11-05, PT11-10, and PT11-11, and one was found in each of trenches PT11-12 and PT11-14. A total of 21 uraniferous boulders were recovered from the trenches (Ainsworth and Thomas 2012).

In mid- to late October 2011, the boulder survey consisted of prospecting with an Exploranium GR-110 handheld scintillometer while trenches were being excavated or backfilled, and while traversing between trenches. The survey resulted in the discovery of many uraniferous boulders. Where radiometric readings were elevated, hand-dug test pits were excavated until a uranium mineralized boulder was found or no obvious radioactive source was located.

Forty-nine of the boulder samples (PB11-67 to PB11-115) were recovered within claims S-111375 and S-111783. All 49 uranium oxide mineralized boulders were found within the limits of the June 2011 boulder field over an area of approximately 4.9 km long by up to 0.9 km wide. These were composed of massive or semi-massive uranium oxide minerals or were basement rocks that contained blebs and/or finely disseminated uranium oxide minerals. The boulder samples ranged from gravel sized up to 25 cm x 30 cm x 40 cm. Radioactivity of these boulders ranged from 701 cps to greater than 9,999 cps (off-scale), and assays ranged from 0.07% U_3O_8 to 31.4% U_3O_8 (Ainsworth and Thomas 2012).

9.3.3 October 2012 Boulder Prospecting

From early to mid-October 2012, radioactive hotspots in two separate areas identified by the September 2012 SPI airborne survey were investigated on the ground.

Boulder surveying in the Patterson Lake area recovered 40 radioactive boulders of which 17 had off-scale radioactivity (greater than 9,999 cps). Thirty-six of these 40 boulder samples were composed of massive or semi-massive uranium oxide minerals or were basement rocks that contained visible blebs and/or finely disseminated uranium oxide minerals. The boulder samples ranged from gravel sized to 30 cm in the longest dimension and assayed from 9 ppm U to 40.0% U_3O_8. These additional boulder samples increased the size of the Patterson Lake boulder field to approximately 7.35 km long by up to 1.0 km wide.

The strong pathfinder elements for the high grade uranium oxide are consistent with previous surveys, namely: Au, B, Co, Cr, Cu, Li, Mo, Pb, Sb, Sr, Th, W, Zr, and most REE.

Boulder prospecting in the Forrest Lake area recovered eight radioactive boulders with radioactivity ranging from 139 cps to 1,060 cps. No visible uranium mineralization was observed in any of the basement boulders that comprised lithologies of quartz-feldspar gneiss, schist, and quartz-feldspar-mafic granite and pegmatite. These boulders ranged from cobble sized to over 80 cm in the longest dimension. The boulders assayed from 6 ppm U to 84 ppm U (Ainsworth 2012b).

9.4 Ground Geophysical Surveys

9.4.1 2008 Self-Potential Survey

In early October 2008, a preliminary self-potential (SP) survey consisting of three lines totalling 8.7 km was completed. SP stations were spaced at 20 m intervals along the lines. Negative values represent most SP anomalies. Lithologic conditions targeted in this survey were clay altered zones, which were conductive and exhibited a negative SP anomaly.

The SP survey values ranged from -339 mV to +124 mV. Four anomalies were delineated (Ainsworth and Beckett 2008).

9.4.2 2011 and 2012 DC Resistivity, HLEM, and SQUID-EM Surveys

Geophysics carried out during November and December 2011 and February through April 2012 consisted of DC resistivity, MaxMin HLEM, and very small moving loop Super-conducting Quantum Interference Device (SQUID)-EM surveys. The ground geophysics was carried out on the PLS main grid area as a follow-up over a radioactive uraniferous boulder field located 5 km to the southwest that had been discovered in June 2011. A total of 30.58 km of MaxMin HLEM, 83.60 km of resistivity, and 14.40 km of SQUID-EM surveys were completed.

The DC resistivity was successful in defining a number of potential targets based on conductivity, changes in the width of conductive packages, and more subtle features indicating possible cross structures. The resistivity and HLEM were initially used for drill targeting with a limited amount of ground SQUID-EM used to follow up some VTEM targets (Bingham 2012).

9.4.3 2012 and 2013 Resistivity and SQUID-EM Surveys

Geophysics carried out during 2012 and 2013 consisted of DC resistivity and SQUID-EM surveys on the PLS west grid area, and SQUID-EM surveys and small moving loop Transient EM survey coverage on the PLS main grid area. A total of 24.6 line-km of resistivity and 30.9 line km of EM surveys was completed.

The extended resistivity data of both the PLS main grid and PLS west grid appeared to be more effective in mapping the expected conductive Cretaceous sediments in this area.

Three conductors were outlined with the ground SQUID-EM survey on the PLS west grid. The south conductor is the most prospective due to strike length, conductivity, and an association with an enhanced basement resistivity low in the vicinity of the conductor on lines 2400E and 2600E. Line 2400E shows a marked increase in amplitude and conductivity. The west end of the central conductor may have a structural association. The north conductor is of low priority mostly due to its apparent shallow dip.

On the PLS main grid, the SQUID-EM surveys infilled and located the south (mineralized), central, and north conductors along the main conductor trends. The amplitude of the south (mineralized) “B” conductor is very weak and flat lying on lines 7200E and 7400E. The south (mineralized) “B” conductor is interpreted as much deeper and weaker on the east extent (Lines 7000, 7200, and 7400) (Bingham 2013).

9.4.4 2013 and 2014 Resistivity and SQUID-EM Surveys

Geophysics carried out during late 2013 and early 2014 consisted of DC resistivity and very small moving loop SQUID-EM surveys conducted by Discovery. During the periods July to August 2013 and September to October 2013, pole-dipole resistivity surveys were completed over the Verm and Far East Grids. During December 2013, pole-dipole resistivity surveys were carried out over the Area B and Forrest Lake grids. During December 2013 to February 2014, Discovery carried out HT SQUID small moving loop TDEM surveys over the Area B, Far East, Forrest Lake, and Verm grids. A total of 93.9 km of pole-dipole DC resistivity and 43.7 km of small moving loop EM surveys were conducted.

The 2013-2014 geophysical surveys were successful in defining priority ground targets based on a combination of resistivity and EM surveys over priority areas based on previous VTEM surveys. Follow-up drilling was conducted on the identified targets in 2015 but no significant mineralization was encountered.

9.4.5 2014 and 2015 Lake Bottom Spectrometer Survey

A proprietary lake bottom spectrometer survey system developed by SPI was operated during April to May 2014 at Area A, covering the area of known mineralization and the primary conductive corridor, and at Area B in the northeastern section of Patterson Lake. The system consisted of a 150 in three sodium-iodide crystal with digitizing electronics for remote data acquisition and control, housed in a temperature controlled casing. The survey was carried out from lake ice utilizing snowmobile/sled and a Novatel L1-L2 Glonass GPS. A total of 1,185 measurements were collected at 20 m stations along 50 m spaced lines that were designed to run parallel to the EM conductor trend in the target areas.

Analysis of the results indicate that the system detected uranium mineralization at 585E and 1080E, and elsewhere anomalous uranium values generally coincided with RadonEx EIC radon-in-water values.

During the same timeframe as the lake bottom spectrometer survey, SPI utilized a proprietary four channel ground penetrating radar (GPR) system towed behind a tracked vehicle to complete approximately 180,000 water depth measurements in the central and northeast areas of Patterson Lake. The water depths matched up well with depths from diamond drilling and earlier radon-in-water surveys

9.4.6 2016 Moving Loop TEM Survey

A small moving loop TEM profiling survey was conducted between May 2 and May 25, 2016, by Discovery, utilizing a Geonics 3D-3 coil sensor (Geonics 3D-3) with a 250 m transmitter-receiver offset measured between the receiver and the centre of the transmitter loop (Discovery 2016).

Data was collected along a total of 26.25 km on 21 grid lines, 19 of which lines covered an area west of the 840W Zone out to 2600W, and two of the lines tested the mineralized conductor trend on the east side of Patterson Lake.

The survey was successful in extending the mineralized conductor trend past the 840W Zone out to the radioactive boulder field and outlined a 200 m long conductivity 'bright spot' centred at 1550W. The Geonics 3D-3 sensor coil allowed the detection of weaker conductive anomalies than a SQUID type sensor, which had been used in previous surveys along the mineralized trend.

9.4.7 2016 Acoustic Profiles (Marine) Survey

SPI was contracted to perform a multi-channel marine acoustic survey to provide a detail subsurface image over the mineralized zones beneath Patterson Lake. There were two objectives:

- Geotechnical – lithology prediction of the overburden to ensure the overall stability of the proposed slurry wall and open pit
- Exploration – a better understanding of the geology of the deposit and surrounding area to evaluate the potential for further exploration (SPI, 2016a and 2016b).

To achieve this objective, a marine-type seismic system was used with a 24-channel streamer and a bubble gun source. A 50 m by 50 m orthogonal grid was followed using a shot interval of 1 m, a recording length of 0.5 s to 1.0 s, depending on water depth, a sample rate of 0.25 ms, and a boat speed of 0.75 m/s.

Between August 31 and September 16, 2016, 88 acoustic profiles covering 127.4 km were acquired. Four hydro-geotechnical wells were used to provide information on the sediments above the basement rock and to correlate with reflectors in the acoustic profiles, while resource drilling was used to correlate basement reflectors.

Interpretation by SPI showed basement faults in east-west and north-south directions, with low vertical displacement. The conclusion reached by SPI was that the entire area appears to be a sinistral east-west strike-slip zone noting indirect evidence of horizontal movement in three areas.

Glacial till was interpreted to consist of a series of ridges oriented in two main directions: an earlier almost east-west and later one north-south orientation. The north-south trending ridges are on top of the east-west ridges. East-west ridges are present in the eastern part of the survey. Smaller ones are 50 m to 60 m wide and 10 m to 20 m high. A better developed east-west ridge (100 m to 120 m wide and 20 m to 30 m high) can be observed on top of the uranium deposit for a significant length in its western part. An interesting aspect of these ridges is that some of them are on top of a basement fault.

Above the glacial till is outwash sand over the whole area. Towards the centre south of the survey and moving to the centre of the lake there are very recent lake sediments.

The survey revealed overburden formations and their thicknesses in the proposed pit area as well as through the entire survey area, and a structural model of the basement lithologies is available to guide future exploration.

9.4.8 2017 Moving Loop TEM Survey

A small moving loop TEM profiling survey was conducted between January 7 to February 2, 2017, by Discovery, utilizing a Geonics 3D-3 with a 250 m transmitter-receiver offset measured between the receiver and the centre of the transmitter loop. Both the transmit loop and receiver were moved in 25 m increments (Discovery 2017).

Data was collected along a total of 24.35 km 11 grid lines. These lines were intended to test various airborne EM conductors within the Property, with a single line established over the strongest part of the airborne conductor, in preparation to select exploration drill targets. A number of profiles were done to determine if weak VTEM P-type responses were sub-vertical basement conductors. Basement conductors were successfully located for the P, Q, R, and S trends. However, the VTEM P-type targets were generally not basement conductors on the Carter Trend (north) and west of the Boulder field. A weak conductor was located south of the boulder field on Line 250W.

9.4.9 2017 Acoustic Profile (Land) Survey

SPI was contracted to perform a test multi-channel acoustic survey on widely spaced lines between the R00E Zone and Highway 955 to the west (SPI 2017). The objective of this test was to obtain the parameters necessary to carry out a future production acoustic survey on land to extend the acoustic coverage from the R780E Zone westward. Several acoustic signal sources were tested to determine if the layers in the sediments could be resolved and the top of the basement rocks mapped, as was achieved with the Marine Acoustic System over the R780E Zone. With a successful test, it would then be possible to obtain acoustic data on land from R00E to the mineralized boulder field to aid exploration and to map the stratigraphy of the sediments for geotechnical applications.

Between May 20 and June 1, 2017, a total of 4.158 km of data was collected along five lines of various orientations, with shot intervals of 2.5 m, a receiver interval of 5 m, and a vibrator source emitted at 30 Hz to 290 Hz with recovered frequencies at 200 Hz.

The land acoustic survey was able to map only the top of mudstone and the basement horizons due to lower resolution of the land acoustics (wide line spacings) compared to the previous marine acoustic survey. Also, where the overburden was thicker early refraction and noise at near offsets made the imaging of the mudstone problematic.

9.4.10 2018 Vertical Seismic Profiling Survey

SPI was commissioned to undertake vertical seismic profiling (VSP) measurements in lake drilled holes for the PFS part of the winter 2018 drilling campaign (SPI 2018). VSP surveying involves recording the responses of geophones in a borehole or well for sources at the earth's surface. The objective of the VSP survey was direct ‘S’ and ‘P’ velocity measurements and reflection information. The ‘S’ velocities were required to determine liquefaction potential of the overburden, and the ‘P’ velocities were needed to calibrate the velocity model used for depth conversion of the marine 2D acoustic profiling dataset.

The VSP survey was carried out from February 16 to 20, 2018, on 11 drill holes. A marine source (bubble gun) was used for the first four holes, an impact source (propane gun) was used for the single land hole, and a P and S impact source was used for the final six holes.

The quality of the data is controlled by the coupling between the casing and the geological formations. All holes surveyed were not cemented, therefore the coupling was not good. Moving the source away from the hole (typically

20 m) improved the quality of the data, however, no reflection data could be acquired. Also, improvements in data quality were noticed when the mud used was thicker than usual.

The ‘P’ velocities were reliable with mitigation of the tube-wave and casing arrivals by operating the source with offset. ‘S’ velocity data was acceptable with improvements noticed in holes where coupling to formation was good through actual contact or addition of heavy drilling fluid (mud).

9.4.11 2019 Vertical Seismic Profiling Survey

SPI was commissioned to undertake VSP, density, and gamma ray measurements in lake drilled holes for the PFS part of the winter 2019 drilling campaign (SPI 2019). VSP surveying involves recording the responses of geophones in a borehole or well for sources at the earth's surface.

The VSP survey was carried out from February 16 to March 13, 2019, on 11 drill holes.

The quality of the data is controlled by the coupling between the casing and the geological formations. All drill-holes surveyed were not cemented with the exception of the last drill-hole RD-35, therefore the coupling was not good. Based on the results of the 2018 survey to mitigate this phenomenon the distance between source and surveyed hole (offset) was increased to 20 m. Despite the 20 m offset, some drill-holes surveyed had casing arrivals (tube waves that travel along steel casing). Even with increased offset, no useful reflection data was recorded.

The ‘P’ velocities were reliable with mitigation of the tube-wave and casing arrivals. ‘S’ velocity data was acceptable. Density data resolution was very good and repeatable (down-hole and up-hole runs).

10.0 DRILLING

After initial targeting using geophysical surveys, diamond drilling on the PLS Property is the principal method of exploration and delineation of uranium mineralization. Drilling can generally be conducted year-round on the PLS Property.

To date, FCU and its predecessors have completed a total of 844 drill holes totalling 227,775 m across the PLS Property, as provided in Table 10-1. Drilling includes exploration, geotechnical, metallurgical, water wells, and hydrogeology drill holes.

Nine holes that were drilled prior to 2011 are considered historical and have minimal information available, and were thus excluded from the resource evaluation. In addition to the historical holes, 105 holes (totalling 11,913 m) used for geotechnical assessment and hydrogeologic characterization were also excluded from the Mineral Resource estimate, however, the three pit wall geotechnical holes drilled in 2019 were included in the Mineral Resource estimate.

Table 10-1 lists the holes by drilling program and hole purpose. Figure 10-1 illustrates the collar locations of the drill holes.

Since the September 19, 2019, Mineral Resource estimate, FCU has completed an additional 181 drill holes totalling 27,392 m over the PLS Property, as listed in Table 10-2, primarily focused at the R780E and R840W deposits. Drilling has been exclusively diamond core drilling (DD) and includes both infill exploration and geotechnical drill programs. The geotechnical drilling has included water level monitoring wells, pump test wells, surface foundations and underground rock mechanics condemnation drilling for proposed facilities (decline and ventilation shafts), and geotechnical characterization for the proposed TMF. Some geotechnical drilling included sonic drilling for overburden coring, or for drill holes that do not extend into the bedrock. Sonic drilling has also been used for the drilling of pumping wells.

Sample acquisition, preparation, security, and analysis were essentially the same for all drill programs and are described in Section 11.

Table 10-1: Drilling Programs

Year	Drilling Program	Hole Purpose	Drill Type	Number of Holes	Total Drill Depth (m)
1978	H1978	Historic	NA; minimal info available	2	240
1979	H1979	Historic	NA; minimal info available	5	549
1980	H1980	Historic	NA; minimal info available	2	217
2011	2011W	Exploration	DD	7	838
2012	2012W	Exploration	DD	16	2,175
	2012S	Exploration	DD	9	1,659
			Dual Rotary	12	1,548
2013	2013W	Exploration	DD	46	9,942
	2013S	Exploration	DD	53	15,564
2014	2014W	Exploration	DD	92	34,252
	2014S	Exploration	DD	82	28,329
2015	2015W	Exploration	DD	88	28,297
	2015S	Exploration	DD	61	21,776
			RC	1	262
2016	2016W	Exploration	DD	38	11,679
			RC	5	1,308
	2016S	Exploration	DD	30	10,044
			RC	4	1,077
		Geotech - Ring Dyke	DD	4	290
		Water Well	DD	9	542
2017	2017W	Exploration	DD	57	17,605
	2017S	Exploration	DD	8	2,626
		Metallurgical	DD	3	811
		Geotech - Pit Wall	DD	4	614
		Water Well	DD	1	305
2018	2018W	Exploration	DD	15	4,906
		Geotech - Pit Wall	Sonic/DD	3	703
		Geotech - Ring Dyke	Sonic/DD	16	1,028
	2018S	Exploration	DD	9	2,928
		TMF	Sonic/DD	5	628

table continues…

Year	Drilling Program	Hole Purpose	Drill Type	Number of Holes	Total Drill Depth (m)
2019	2019W	Geotech - Pit Wall	Sonic/DD	3	1,034
		Geotech - Ring Dyke	Sonic/DD	16	1,056
		Monitoring Well	DD	2	366
		Pumping Test	DD	2	280
		TMF	Sonic/DD	10	1,125
2021	2021W	Exploration	DD	21	7,147
	2021S	Exploration	DD	25	6,124
		Metallurgical	DD	4	722
		Monitoring Well	DD	3	140
		Pumping Well	Sonic/DD	2	217
		TMF	Sonic/DD	21	1,153
		Camp	Sonic/DD	1	31
		Decline	Sonic/DD	25	2,707
		Process Plant Facility	Sonic/DD	3	124
		Geotech	Sonic/DD	3	774
		Ventilation Shaft	Sonic/DD	4	423
		Waste Rock	Sonic/DD	6	567
2022	2022W	Ventilation Shaft	SONIC/DD	1	202
		Crown Pillar	SONIC/DD	5	843
Grand Total	--	--	--	**844**	**227,775**

Notes:

1. “W” refers to Winter, and “S” refers to Summer (i.e., 2018S refers to the 2018 Summer drill program)
2. “DD” refers to diamond drill.

Table 10-2: Drilling Programs Completed Since October 2018

Year	Drilling Program	Hole Purpose	Drill Type	Number of Holes	Total Drill Depth (m)
2018	2018W	Geotech - Pit Wall	DD	3	703
		Geotech - Ring Dyke	DD	16	1,028
	2018S	TMF	DD	5	628
2018 Total				24	2,359
2019	2019W	Geotech - Pit Wall	DD	3	1,034
		Geotech - Ring Dyke	DD	16	1,056
		Monitoring Well	DD	2	366
		Pumping Test	DD	2	280
		TMF	DD	10	1,125
2019 Total	--	--	--	33	3,861
2021	2021W	Exploration	DD	21	7,147
	2021S	Exploration	DD	25	6,124
		Metallurgical	DD	4	722
		Monitoring Well	DD	3	140
		Pumping Well	DD	2	217
		TMF	DD	21	1,153
		Camp	DD	1	31
		Decline	DD	25	2,707
		Maintenance Facility	DD	3	124
		Geotech	DD	3	774
		Ventilation Shaft	DD	4	423
		Waste Rock	DD	6	567
2021 Total	--	--	--	118	20,128
2022	2022W	Ventilation Shaft	SONIC/DD	1	202
		Crown Pillar	SONIC/DD	5	843
2022 Total	--	--	--	6	1,044
Grand Total	**--**	**--**	**--**	**181**	**27,392**

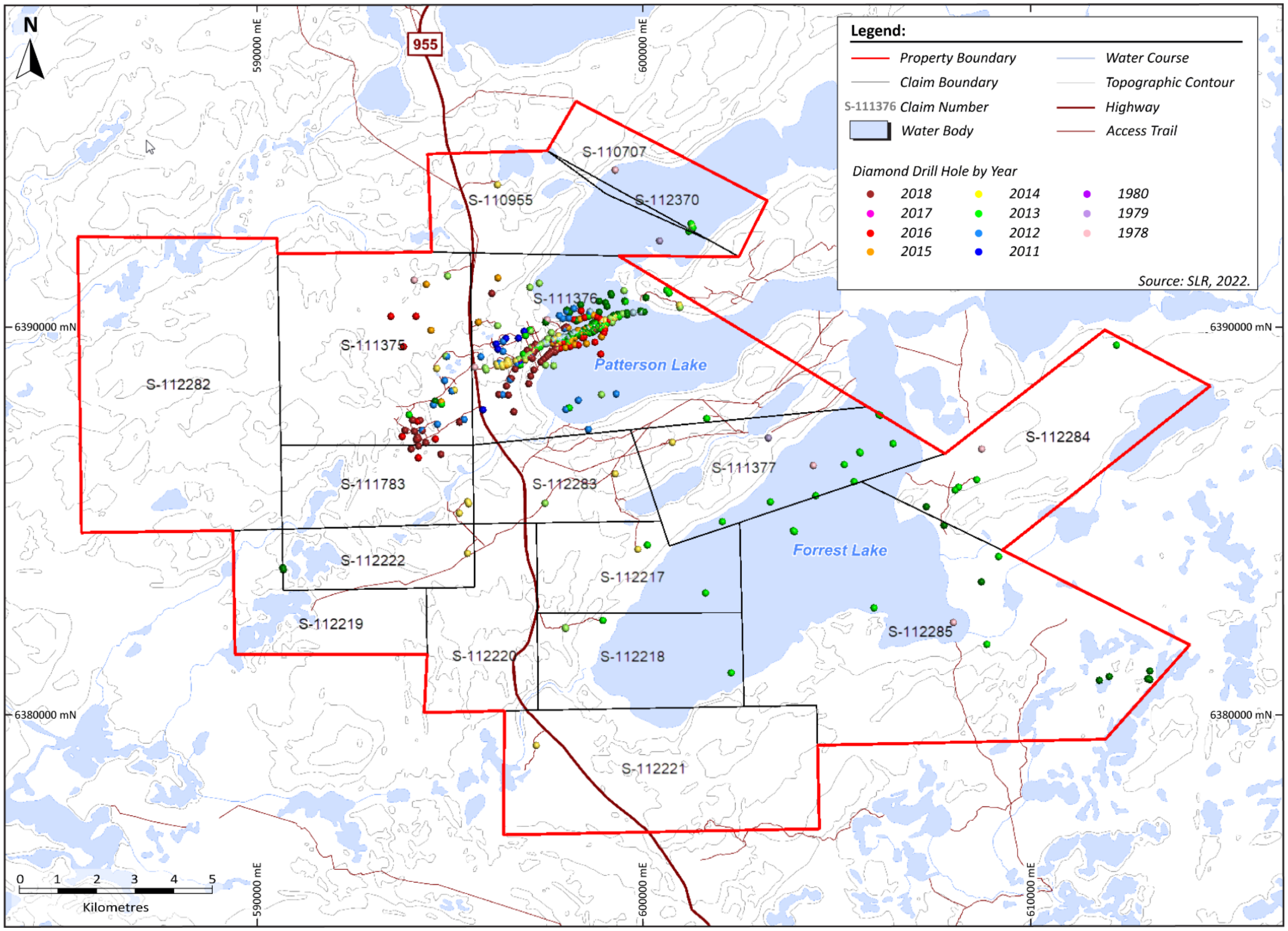


Figure 10-1: Drill Hole Location Plan

10.1 Diamond Drilling

From November 2011 to September 2015, 142,832 m of drilling was completed in 454 diamond drill holes on the PLS Property. During the winter 2015 drill program, an initial Inferred Mineral Resource estimate for the Triple R deposit was published (RPA 2015a). Following the spring 2015 drill program, RPA completed a PEA on the Triple R deposit (RPA 2015b).

From January 2016 to December 2018, FCU continued to conduct both delineation and step out drilling programs along strike of the Triple R deposit by completing 52,983 m of drilling in 169 holes. Drill holes were primarily designed to both infill in support of an Indicated Mineral Resource classification in the R780E high grade (HG) and R780E MZ domain and materially expand the footprint of Inferred mineralization in the R00E and R780E areas. Step out regional drilling during this time was also successful in identifying two significant new areas of mineralization (R1515W and R1620E) and expanding mineralization at R840W. The goal of the summer 2018 program, which consisted of nine holes totalling 2,928 m drilled, was to drill key areas of the R780E HG zone that were classified in 2015 as "Inferred" and upgrade them to "Indicated". To that extent, the nine drill holes intersected width and strength of mineralization where expected and allowed for upgrading the classification in these areas. Following the summer 2018 drill program, RPA along with Clifton and Wood completed a PFS on the PLS property (RPA, Wood, and Clifton 2019) based on a total of 197,651 m of drilling in 636 drill holes.

As part of advanced stage studies for the Triple R deposit, FCU resumed infill drilling programs at R780E and R840W with the intention of upgrading certain high priority areas from Inferred Mineral Resources to Indicated Mineral Resources. Of the 175 drill holes completed since the 2019 estimate, 46 infill exploration and 3 geotechnical drill holes, totalling 14,304 m, targeted the HG and main low grade (LG) domains in the R780E and R840W zones (24 drill holes totalling 8,180 m and 25 drill holes totalling 6,124 m, respectively) with the objective to upgrade Inferred Mineral Resources to the Indicated classification, and improve the geotechnical understanding of the zones. The drilling programs are listed in Table 10-3 and illustrated in Figure 10-2, Figure 10-3, and Figure 10-4.

Table 10-3: Drilling Programs Completed to Update the R780E (September 2021) and R840W (April 2022) Mineral Resource Estimates Since 2019

Year	Zone	Drilling Program	Hole Purpose	Drill Type	Number of Holes	Total Drill Depth (m)
2019	R780E	2019W	Exploration / Geotech - Pit Wall	DD	3	1,034
				2019 Total	**3**	**1,034**
2021	R780E	2021W	Exploration	DD	21	7,147
	R840W	2021S	Exploration	DD	25	6,124
				2021 Total	**46**	**13,271**
				Grand Total	**49**	**14,305**

The initial drill program in 2011 was contracted to Aggressive Drilling Ltd. from Saskatoon, Saskatchewan, which used a skid-mounted Boart Longyear LF-70 drill. From February 2012 to April 2013, the drilling was contracted to Hardrock Diamond Drilling Ltd. from Penticton, British Columbia, which used Atlas Copco CS-10 and CS-1000 skid-mounted drills. From July 2013 onwards, drilling was carried out by Bryson Drilling Ltd. from Archerwill, Saskatchewan, using Zinex Mining Corp. A5 diamond drills.

Unless the hole was pre-cased using an RC drill, the usual procedure was to drill through the overburden with NW (88.9 mm diameter) equipment and ream HWT (114.3 mm) casing until refusal or bedrock was reached. If the HWT rods became stuck, casing was completed using NW equipment until competent bedrock was reached at which time coring with NQ (69.9 mm) commenced. A select number of drill holes were directionally drilled by International Directional Services (IDS) in order to facilitate accurate drill targeting during infill drilling.

A Boart Longyear LS 600 track mounted sonic drill was used to case diamond drill holes starting with the winter 2021 drill program; the sonic would advance 6” sonic casing to bedrock, and then install HW diamond drill casing through the sonic casing, before retrieving their sonic casing and moving on to the next location. The advantage of utilizing a sonic drill for casing is the speed of casing as well as increased hole straightness required for accurate targeting of tight infill drillholes.

10.2 Dual Rotary Drilling

From October to November 2012, twelve 4.5 in (11.43 cm) diameter dual rotary drill holes totalling 1,548 m were completed by J.R. Drilling Ltd. of Cranbrook, British Columbia, using a Foremost DR-12 drill. These drill holes were not used in the resource evaluation but designed to penetrate the glacial sediments overlying bedrock so that the specific (and more radioactive) till sheet hosting uranium mineralized boulders could be traced back to bedrock source by gamma probing the overburden. Additionally, some rotary drill hole collars were planned to also test bedrock VTEM and TDEM conductors by drilling approximately 20 m into solid bedrock. The overburden and basement material were collected on site in sampling buckets at 1 m intervals. Each bucket was measured using an Exploranium GR-110G total count gamma-ray scintillometer, and a 1 kg to 3 kg sub-sample was removed for logging using a scoop from a 5-gal bucket.

Each drill hole was logged using a Mount Sopris 2PGA-1000 gamma probe. Additionally, holes PLSDR12-001 and PLS12-009 through PLSDR12-012 were surveyed using a custom downhole spectrometer probe, built, and operated by SPI. A Trimble GeoXH handheld GPS instrument and a Trimble 5800 base station for differential corrections were utilized to locate all dual rotary drill hole locations.

According to Ainsworth (2012b), accurate and precise sample collection for geochemical analysis was challenging due to several factors. Sample volume returned through the cyclone was at times overwhelming and was further complicated by the large influx of groundwater. The drilling itself introduced sample bias especially in terms of size fraction and relative abundance. It was found that fine materials were prone to be either washed or blown away. Since the maximum size of returned samples was approximately 2 cm to 3 cm, it can be presumed that material larger than small pebbles was either pushed out of the way or crushed by the advancing drill bit and casing.

The current working depth of each rotary hole was determined by marking the casing every m. The inaccuracies of this method were confirmed by comparing the determined final depth to the gamma probe wire line measured final depth; discrepancies of several m were common.

Caving of material around the casing and subsequent transport to surface introduced sample contamination, especially in thick sand units beneath the water table.

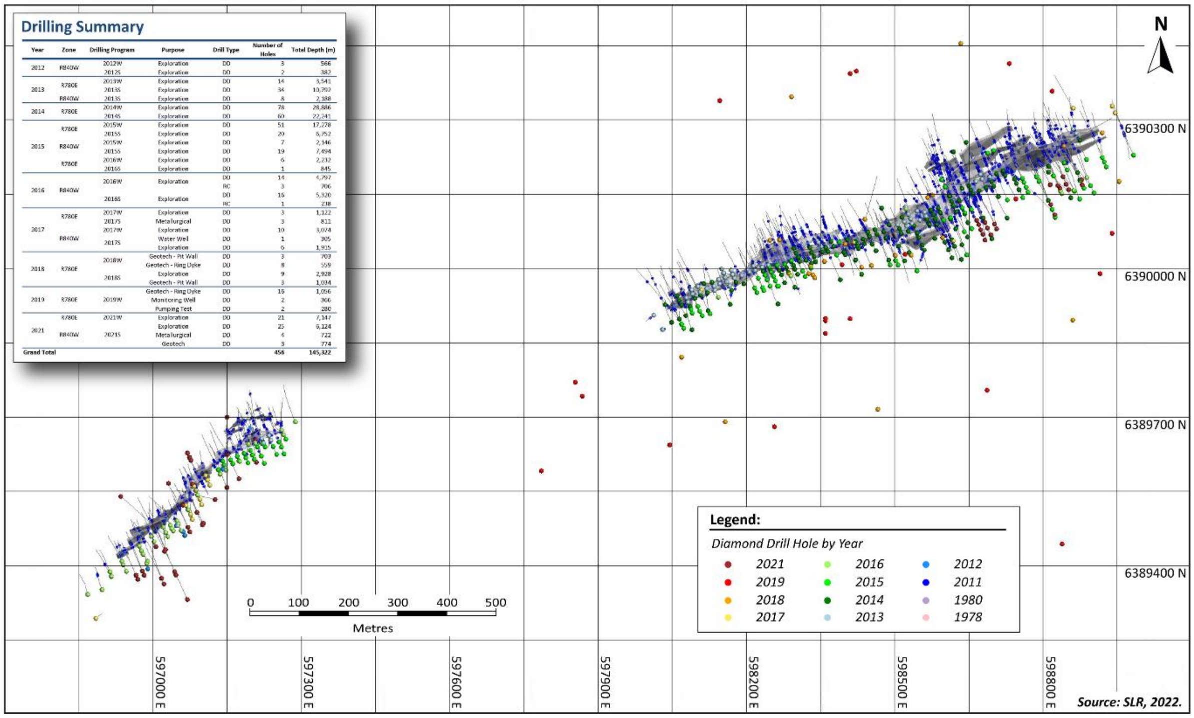


Figure 10-2: R780E and R840E Drill Hole Location Map

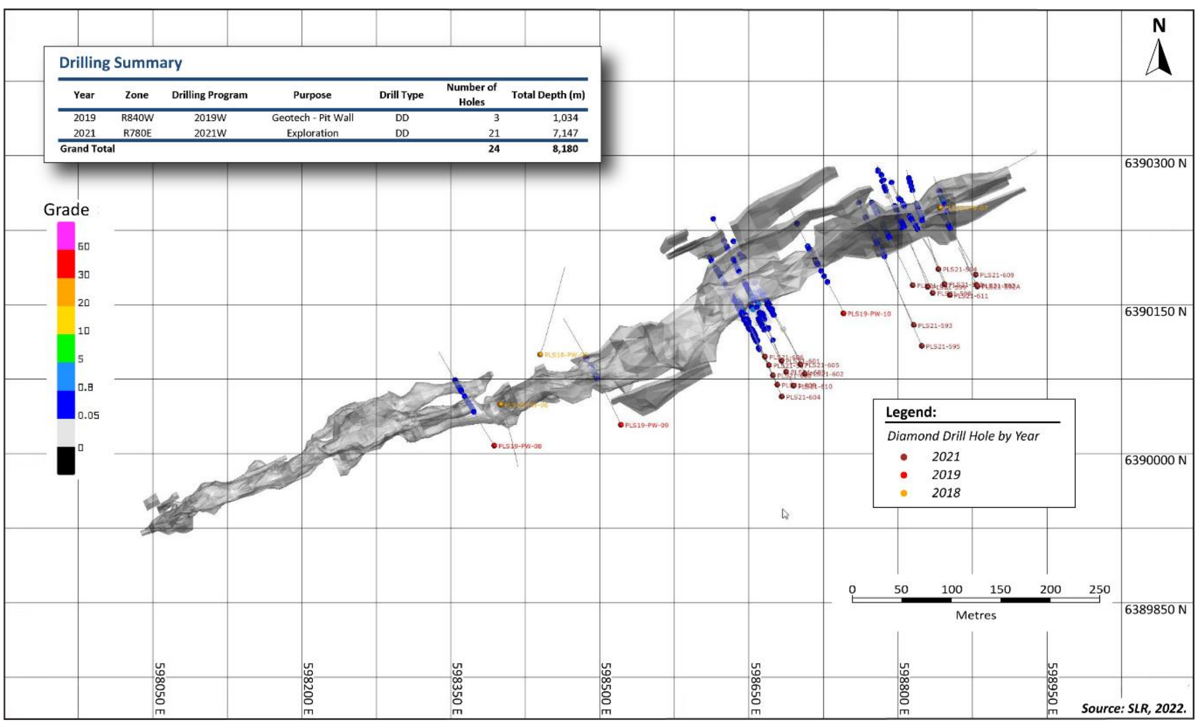


Figure 10-3: R780E Exploration Infill Drill Hole Location Map Completed 2019–2021

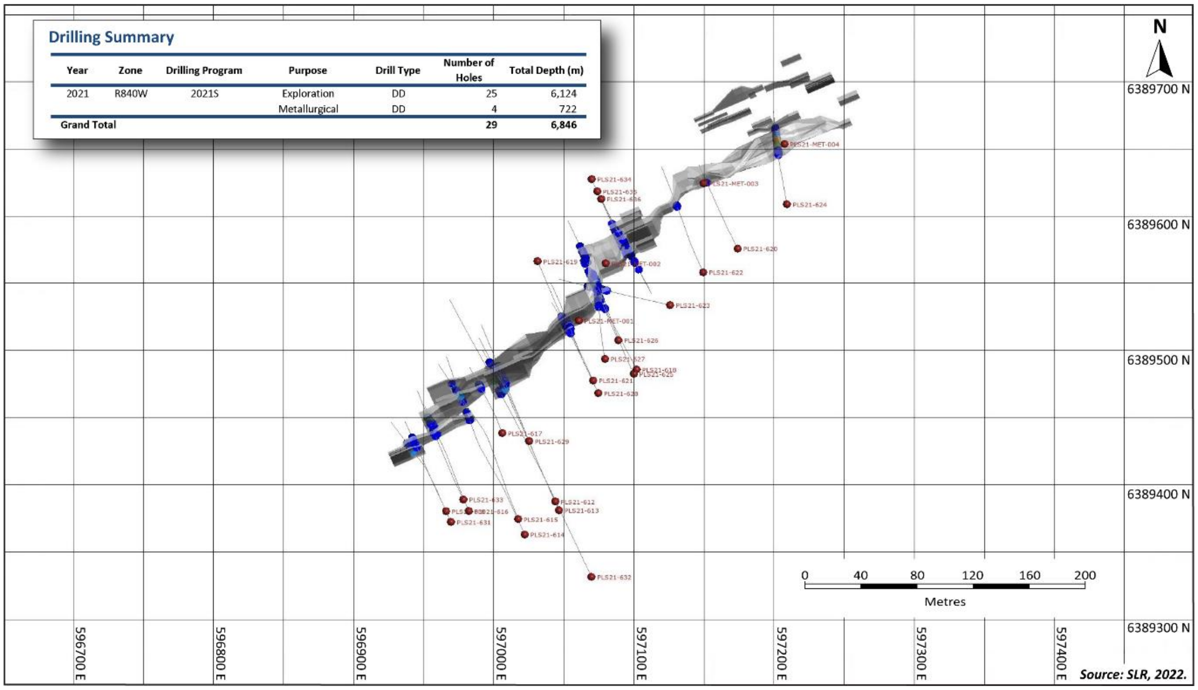


Figure 10-4: R840WE Exploration Infill Drill Hole Location Map Completed 2021

10.3 Reverse Circulation Drilling

In January 2013, the process of pre-drilling the casings of most holes was initiated. Northspan Explorations Ltd. (Northspan) was contracted to set the casing to a targeted depth of 1 to 2 m above bedrock. Northspan used either a Hornet XL or Attacus RC drill to sink the HW (117.65 mm) casing. No samples were recovered during the RC drilling. A Trimble GeoXH handheld GPS instrument and Trimble 5800 base station for differential corrections were utilized to locate all drill collar locations during the winter 2013 program. From the summer 2013 drill program onwards, all holes were located using a Trimble R10 GNSS real time kinematic (RTK) system.

10.4 Drill Hole Surveying

The collars of the 2011 and winter 2012 program holes were located using a handheld Garmin GPSMAP 60CSx instrument. During the winter 2013 program, drilled holes were located using a Trimble GeoXH handheld GPS instrument and a Trimble 5800 base station for differential correction. From the summer 2013 drill program onwards, all holes were located using a Trimble R10 GNSS RTK system. All drill hole positions from the 2012 fall program onwards were surveyed again upon completion of the hole to account for moving of the drill, due to either ground conditions or drilling difficulty. All roads and traverses travelled were located with a handheld Garmin GPSMAP 60CSx or Trimble instrument noted above.

10.5 Downhole Orientation Surveying

Until the summer of 2014, all holes drilled from the lake were oriented vertically. Holes drilled during the 2011 and winter 2012 drilling programs were tested for dip deviation with acid tests. The fall 2012 drilling program holes were either acid tested or surveyed with a Reflex EZ-Shot instrument. Upon completion, all holes drilled in 2013 were surveyed using an Icefields gyro survey tool. The Icefields gyro was replaced in 2014 by a Stockholm Precision Tools north seeking gyro. In 2016 a Reflex Smart Gyro was also used. Additionally, an Axis Champ Gyro was also used sporadically during the 2015 and 2018 drill programs. For the winter 2015 drill program, an Icefields gyro shot instrument was used to survey all drill holes. Gyro selection depended on rental availability for primary and backup units. All gyros are routinely tested in a test bore hole which FCU has maintained for gyro data quality control purposes.

10.6 Drill Core Handling and Logging Procedures

All holes were systematically probed within the rods using a Mount Sopris 500 m (4MXA-1000) or 1,000 m (4MXC-1000) winch, Matrix logging console, and either a 2PGA-1000 or 2GHF-1000 total gamma count probe upon completion of the hole.

Core recovery is generally very good, allowing for representative samples to be taken and accurate analyses to be performed.

The drill core was placed sequentially in wooden core boxes at the drill by the drillers. Twice daily, the core boxes were transported by FCU personnel to the core logging and sampling facility where depth markers were checked, and the core was carefully reconstructed. The core was logged geotechnically on a run by run basis including the number of naturally occurring fractures, core recovery, rock quality designation (RQD), and range of radiometric counts per second. The core was scanned using a handheld Exploranium GR-110G total count gamma-ray scintillometer until the winter 2014 program, after which Radiation Solutions RS-121 scintillometers were used. Between the 2015 winter program and 2017, clay mineralogy was identified in the field using an ASD Inc. TerraSpec Halo near infrared mineral analyzer.

The core was descriptively logged utilizing a Panasonic Tough Book laptop computer by an FCU geologist paying particular attention to major and minor lithologies, alteration, structure, and mineralization. Logging and sampling information was entered into a spreadsheet based template which was integrated into the Project digital database.

All drill core was photographed wet with a digital camera before splitting.

FCU's sampling protocol calls for representative samples to be taken of both sandstone and basement lithologies. At least one representative sample of sandstone (Devonian or Athabasca) was taken when intersected. In thicker zones of sandstone (more than 5 m), representative samples were taken at 2.5 m intervals. Representative samples of basement lithologies consisting of 50 cm of split core (halved) were taken every 10 m within the basement, starting immediately in bedrock.

In addition to the representative samples, point samples were taken in both sandstone and basement lithologies.

All sandstone and basement intervals with handheld scintillometer readings greater than 300 cps, or containing significant faults and associated alteration, were continuously sampled with a series of 50 cm split core samples. In areas of strong to intense alteration, evenly spaced 50 cm split core samples were taken from the start of the alteration. The spacing of the samples varied with the width of the alteration zone as follows: 1 m spacing for alteration zones less than or equal to 5 m long, 2 m spacing for alteration zones between 5 m and 30 m long, and 5 m spacing for alteration zones more than 30 m long.

Samples for density measurements were taken in both sandstone and basement lithologies. Because of the limited thickness of sandstone intersected on the PLS Property, Sarioglu (2014) recommended that at least one sandstone sample be taken for density measurement per hole, where possible. Density samples in mineralized basement or sandstone giving handheld scintillometer readings greater than 300 cps were taken at 2.5 m intervals. No density samples were taken in barren sandstone from the 2014 summer drill program onwards. Basement samples for density outside the mineralized zone were taken at 20 m intervals until the winter 2014 drill program, after which no barren basement density samples were taken.

Core marked for sampling was split in half using a manual core splitter. Half the core was returned to the core box and the other half was placed in plastic sample bags and secured with an impulse sealer.

Split core samples were tracked using three part ticket booklets. One tag was stapled into the core box at the start of the appropriate sample interval, one tag was placed into the sample bag, and the final tag was retained in the sample booklet for future reference. For each sample, the date, drill hole number, project name, and sample interval depths were noted in the sample booklet. The data were transcribed to a Microsoft Excel spreadsheet and stored on the FCU data server. Sample summary files were checked for accuracy against the original sample booklets after the completion of each drill program. The digital sample files also contain alteration and lithology information.

Core trays were marked with aluminum tags.

The plastic sample bags were put into 5-gallon sample pails and sealed, and were held in a secure area until they were ready for transportation. The samples were picked up on site by Marsh Expediting and transported by road to La Ronge before transhipment to Saskatchewan Research Council (SRC) in Saskatoon. SRC operates in accordance with International Organization for Standardization / International Electrotechnical Commission (ISO/IEC) 170:2005 (CAAN-P-4E) General Requirements of Mineral Testing and Calibration Laboratories and is also compliant with CAN-P-1579, Guidelines for Mineral Analysis Testing Laboratories.

At SRC, sandstone and basement samples were prepared in separate areas of the laboratory to minimize the potential for contamination. Sample preparation in the laboratory involved drying the samples and sorting them according to radioactivity before jaw crushing.

In the SLR QP's opinion, the logging and sampling procedures meet or exceed industry standards and are adequate for the purpose of Mineral Resource estimation.

10.7 Drill Core Storage

The core from the first drilling programs was stored at the Big Bear Lodge on Grygar Lake, but since August 2013, all the core has been stored at a purpose-built storage facility located west of Patterson Lake.

11.0 SAMPLE PREPARATION, ANALYSIS, AND SECURITY

11.1 Sample Preparation and Analysis

11.1.1 Drill Core Geochemical Analysis

All geochemistry core samples were analyzed by the ICP1 package offered by SRC, which includes 62 elements determined by inductively coupled plasma optical emission spectrometry (ICP-OES). All samples were also analyzed for boron until the end of the winter 2012 drill program and uranium by fluorimetry (partial digestion). Uranium by fluorimetry was replaced at SRC in late 2012 by inductively coupled plasma mass spectrometry (ICP-MS) analysis, which was discontinued on FCU's samples after the winter 2013 drill program.

For partial digestion analysis, samples were crushed to 60% passing -2 mm and a 100 g to 200 g sub-sample was split out using a riffler. The sub-sample was pulverized to 90% passing 106 µm using a standard puck and ring grinding mill. The sample was then transferred to a plastic snap top vial. An aliquot of pulp was digested in a mixture of HNO_3:HCl in a hot water bath for an hour before being diluted by 15 mL of de-ionized water. The samples were then analyzed using a Perkin Elmer ICP-OES instrument (models DV5300 or DV8300). For total digestion analysis, an aliquot of pulp was digested to dryness in a hot block digester system using a mixture of concentrated HF:HNO_3:$HClO_4$. The residue was then dissolved in 15 mL of dilute HNO_3 and analyzed using the same instrument(s) as above.

Select samples with low concentrations of uranium (less than 100 ppm) identified by the partial and/or total ICP-OES analysis were also analyzed by fluorimetry (2012) and ICP-MS (winter 2013). After being analyzed by ICP-OES, an aliquot of digested solution was pipetted into a 90% Pt – 10% Rh dish and evaporated. A NaF/LiF pellet was placed on the dish and fused on a special propane rotary burner then cooled to room temperature. The uranium concentration of the sample was then read using a spectrofluorometer. Uranium by fluorimetry has a detection limit of 0.1 ppm (total) or 0.02 ppm (partial). In the fall of 2012 uranium analysis by fluorimetry was replaced at SRC with uranium by ICP-MS. For ICP-MS partial digestions, an aliquot of sample pulp is digested in a mixture of concentrated nitric hydrochloric acid (HNO_3:HCl) in a test tube in a hot water bath, then diluted using de-ionized water.

For boron analysis, an aliquot of pulp was fused in a mixture of NaO_2/$NaCO_3$ in a muffle oven. The fused melt was dissolved in de-ionized water and analyzed by ICP-OES.

11.1.2 Drill Core Assay

Drill core samples from mineralized zones were sent to SRC for uranium assay. The laboratory offers an ISO/IEC 17025:2005 accredited method for the determination of U_3O_8 in geological samples. The detection limit is 0.001% U_3O_8. Samples were crushed to 60% -2 mm and a 100 g to 200 g sub-sample was split out using a riffle splitter. The sub-sample was pulverized to 90% passing 106 µm using a standard puck and ring grinding mill. An aliquot of pulp was digested in a concentrated mixture of HNO_3:HCl in a hot water bath for an hour before being diluted by de-ionized water. Samples were then analyzed by a Perkin Elmer ICP-OES instrument (models DV4300 or DV5300).

In addition to uranium assaying, all samples from mineralized zones were also assayed by SRC for gold and, until mid-summer 2014, platinum group elements (Pt, Pd). Samples were prepared using the same method as described above. An aliquot of sample pulp was mixed with fire assay flux in a clay crucible and a silver ingot was added prior to fusion. The mixture was fused at 1,200°C for 90 minutes. After the mixture had fused, the slag was poured into a form which was cooled. The lead bead was recovered and chipped until only the precious metal bead remains. The bead was then parted in diluted HNO_3. The precious metals were dissolved in aqua regia and then diluted for analysis by ICP-OES and/or atomic absorption spectrometry (AAS). The analysis has a detection limit of 2 ppb for all three elements. SRC participates in the Canadian Centre for Mineral and Energy Technology (CANMET) Canadian Certified Reference Materials Project (CCRMP) Proficiency Testing Program for Mineral Analysis Laboratories (PTP-MAL) for elements assayed using this method.

11.1.3 Drill Core PIMA Analysis

Core chip samples for clay analysis were sent to Rekasa Rocks Inc., a private facility in Saskatoon, for analysis on a PIMA spectrometer using short wave infrared spectroscopy. Samples were air or oven dried prior to analysis in order to remove any excess moisture. Reflective spectra for the various clay minerals present in the sample were compared to the spectral results from Athabasca samples for which the clay mineral proportions have been determined in order to obtain a semi-quantitative clay estimate for each sample.

11.1.4 Drill Core Petrographic Analysis

Samples collected for petrography were sent to Vancouver Petrographics Ltd., located in Langley, British Columbia, for the preparation of thin sections and polished slabs. Petrographic analysis was performed in the offices of Mineral Services Canada Inc. (MSC) using a Nikon Eclipse E400 microscope equipped with transmitted and reflected light. The results of that work are in two internal reports prepared by MSC for FCU: MSC12/018R-Patterson Lake and MSC14/012R_PLS (MSC 2012, MSC 2014c) rock types.

11.1.5 Drill Core Bulk Density Analysis

Drill core samples collected for bulk density measurements were sent to SRC. Samples were first weighed as received and then submerged in de-ionized water and re-weighed. The samples were then dried until a constant weight was obtained. The sample was then coated with an impermeable layer of wax and weighed again while submersed in de-ionized water. Weights were entered into a database and the bulk density of each sample was calculated. Water temperature at the time of weighing was also recorded and used in the bulk density calculation.

Results were used to convert volumes to tonnages when estimating Mineral Resources. The method and results are described in Section 14, Mineral Resource Estimate.

11.2 Quality Assurance and Quality Control

Quality assurance/quality control (QA/QC) programs provide confidence in the geochemical results and help ensure that the database is reliable to estimate Mineral Resources. FCU's QA/QC program at Triple R includes the following components:

1. Determination of accuracy – achieved by regular insertion of standards or certified reference materials (CRM) of known grade and composition.
2. Determination of precision – achieved by regular insertion of duplicates for each stage of the process where a sample is taken or split.
3. Checks for contamination – by insertion of blanks.

A summary by type of the CRM, blanks, and duplicates inserted into the sample stream is presented in Table 11-1 while the frequency is summarized in Table 11-2. Prior to the winter 2012 drill program, the only QA/QC procedures implemented on samples from the PLS Property were those performed internally by SRC as discussed below.

Table 11-1: Summary of QA/QC Source and Type by Year

QA/QC Group	Sample Source	QA/QC type	Sample Descriptions	2011	2012		2013		2014		2015	
				Fall	Winter	Fall	Winter	Summer	Winter	Summer	Winter	Summer
1	FU	InHouse (FU) blank samples (pulp sample)	Blanks	N	N	N	Y	N	N	N	N	N
	FU	InHouse (FU) blank samples (rock sample)	Blanks	N	N	N	N	Y	Y	Y	Y	Y
2	FU	InHouse (FU) U_3O_8 wt.% CRM - from Waterbury	*LGR/MGR/HGR	N	N	N	Y	N	N	N	N	N
	CANMET & FU	FU inserted CANMET U_3O_8 wt.% CRM	#UTS-3(LGR), DH-1A, RL-1(MGR), BL-5(HGR), ***CUP-2(VHGR)	N	N	N	N	Y	Y	Y	Y	Y
	CANMET & FU	FU inserted CANMET Au CRM	CH-4	N	N	N	N	N	N	N	N	N
3	FU & SRC	Partial and total (ppm) duplicates (1/4 split)	Field Duplicate (FiD), Prep & Pulp Duplicates	N	Y	Y	Y	Y	Y	N	N	N
	FU & SRC	Partial and total (ppm) duplicates (1/2 split)	Field Duplicate (FiD), Prep & Pulp Duplicates	N	N	N	N	N	Y	Y	Y	Y
	FU & SRC	U_3O_8 test report wt.% duplicates (1/4 split)	Field Duplicate (FiD), Prep & Pulp Duplicates	N	N	Y	Y	Y	Y	N	N	N
	FU & SRC	U_3O_8 test report wt.% duplicates (1/2 split)	Field Duplicate (FiD), Prep & Pulp Duplicates	N	N	N	N	N	Y	Y	Y	Y
4	CANMET & SRC	SRC inserted CANMET U_3O_8 wt.% CRM standards	BL2A, BL3, BL4A, BL5, SRCU2	N	Y	Y	Y	Y	Y	Y	Y	Y
	SRC	SRC inserted internal ICP standards w/ varying Boron	CAR110/BL, CAR110/BSM, ****CAR110/BSH, ASR109/BL, CAR218/BSL/BSM	Y	Y	Y	Y	Y	Y	Y	Y	Y
	SRC	Au standards	OXG83, OXL75, OXL78, SJ10	N	Y	Y	Y	N	N	N	N	N
5	FU/SRC	ICP repeat analysis of same sample	Basement/BasementRA/Sandstone and Repeats	Y	Y	Y	Y	Y	Y	Y	Y	Y
	FU/SRC	U_3O_8 test report wt.% repeats	Basement/BasementRA/Sandstone and Repeats	N	N	Y	Y	Y	Y	Y	Y	Y
	FU/SRC	Au repeats	Basement/BasementRA/Sandstone and Repeats	N	Y	Y	Y	Y	Y	Y	Y	Y

QA/QC Group	Sample Source	QA/QC Type	Sample Descriptions	2016		2017		2018		2019	2021	
				Winter	Summer	Winter	Summer	Winter	Summer	Winter	Winter	Summer
1	FU	InHouse (FU) blank samples (pulp sample)	Blanks	N	N	N	N	N	N	N	N	N
	FU	InHouse (FU) blank samples (rock sample)	Blanks	Y	Y	Y	Y	Y	Y	Y	Y	Y
2	FU	InHouse (FU) U_3O_8 wt.% CRM – from Waterbury	*LGR/MGR/HGR	N	N	N	N	N	N	N	N	N
	CANMET & FU	FU inserted CANMET U_3O_8 wt.% CRM	#UTS-3(LGR), DH-1A, RL-1(MGR), BL-5(HGR), ***CUP-2(VHGR)	Y	Y	Y	Y	Y	Y	Y	Y	Y
	CANMET & FU	FU inserted CANMET Au CRM	CH-4	N	N	N	N	N	Y	Y	Y	Y

table continues…

QA/QC Group	Sample Source	QA/QC Type	Sample Descriptions	2016		2017		2018		2019	2021	
				Winter	Summer	Winter	Summer	Winter	Summer	Winter	Winter	Summer
3	FU & SRC	Partial and total (ppm) duplicates (1/4 split)	Field Duplicate (FiD), Prep & Pulp Duplicates	N	N	N	N	N	N	N	N	N
	FU & SRC	Partial and total (ppm) duplicates (1/2 split)	Field Duplicate (FiD), Prep & Pulp Duplicates	Y	Y	Y	Y	Y	Y	Y	Y	Y
	FU & SRC	U_3O_8 TEST REPORT wt.% duplicates (1/4 split)	Field Duplicate (FiD), Prep & Pulp Duplicates	N	N	N	N	N	N	N	N	N
	FU & SRC	U_3O_8 TEST REPORT wt.% duplicates (1/2 split)	Field Duplicate (FiD), Prep & Pulp Duplicates	Y	Y	Y	Y	Y	Y	Y	Y	Y
4	CANMET & SRC	SRC inserted CANMET U_3O_8 wt.% CRM standards	BL2A, BL3, BL4A, BL5, SRCU2	Y	Y	Y	Y	Y	Y	Y	Y	Y
	SRC	SRC inserted internal ICP standards w/ varying Boron	CAR110/BL, CAR110/BSM, ****CAR110/BSH, ASR109/BL, CAR218/BSL/BSM	Y	Y	Y	Y	Y	Y	Y	Y	Y
	SRC	Au standards	OXG83, OXL75, OXL78, SJ10	N	N	N	N	N	N	N	N	N
5	FU/SRC	ICP repeat analysis of same sample	Basement/BasementRA/Sandstone and Repeats	Y	Y	Y	Y	Y	Y	Y	Y	Y
	FU/SRC	U_3O_8 TEST REPORT wt.% repeats	Basement/BasementRA/Sandstone and Repeats	Y	Y	Y	Y	Y	Y	Y	Y	Y
	FU/SRC	Au repeats	Basement/BasementRA/Sandstone and Repeats	Y	Y	Y	Y	Y	Y	Y	Y	Y

Table 11-2: Summary of QA/QC Sampling Insertions by Year

Sample Type	2011 Fall	2012 Winter	2012 Fall	2013 Winter	2013 Summer	2014 Winter	2014 Summer	2015 Winter	2015 Summer
Drill holes	7	16	9	46	53	92	82	88	61
Total Original Geochem Sample Count[1]	**49**	**530**	**518**	**4,791**	**9,058**	**26,732**	**17,045**	**15,039**	**8,142**
w/ U_3O_8 assays (+/- Au assays)	0	5	249	2,852	7083	21,269	12,922	10,013	4,475
w/ U_3O_8 assays (+ Au assays)	0	4	246	2,696	4046	15,225	12,921	9,773	4,475
w/ U_3O_8 assays (no Au assays)	0	1	3	156	3037	6,044	1	240	0
w/ Au assays (no U_3O_8 assays)	0	79	8	26	0	1	2	0	0
ICP-OES Only	49	446	261	1,913	1975	5,462	4,123	5,026	3,667
Total QA/QC Material Frequency	**5**	**240**	**324**	**1,938**	**4726**	**12252**	**7,319**	**6,897**	**2,990**
Blanks	0	0	0	39	49	114	74	64	41
Field Duplicates	0	54	42	151	425	1,269	800	660	331
Coarse Rejects (Prep Duplicates)	0	54	42	151	425	1,269	800	660	331
Pulp Duplicates	0	54	42	151	425	1,269	800	660	331
In House Reference Materials	0	0	0	119	151	273	203	199	132
External Reference Materials	3	48	129	782	1502	3,964	2,468	2,096	1,035
Repeats of U_3O_8 assays, Au Assays, and ICP Analyses	2	30	69	545	1749	4,094	2,174	2,258	689
Umpire Laboratory Repeat Analyses	0	0	2	24	56	122	96	100	100
Total Samples	**54**	**770**	**842**	**6,729**	**13,784**	**38,984**	**24,364**	**21,936**	**11,132**

Sample Type	2016 Winter	2016 Summer	2017 Winter	2017 Summer	2018 Winter	2018 Summer	2019 Winter	2021 Winter	2021 Summer	Total
Drill Holes	37	30	47	15	34	9	3	20	24	673
Total Original Geochem Sample Count[1]	**4,862**	**4,486**	**6,764**	**3,139**	**3,916**	**2,912**	**545**	**5,000**	**2,603**	**116,131**
w/ U_3O_8 assays (+/- Au assays)	2,829	1,906	2,337	2,219	3,164	2,748	482	4,726	2,396	81,675
w/ U_3O_8 assays (+ Au assays)	2,456	1,906	2,336	2,219	3,110	2,719	480	4,725	2,396	64,612
w/ U_3O_8 assays (no Au assays)	373	0	1	0	54	2,710	0	1	0	12,620
w/ Au assays (no U_3O_8 assays)	0	0	0	0	0	9	0	0	0	125
ICP-OES Only	2,033	2,580	4,427	920	683	155	63	274	207	34,264
Total QA/QC material frequency	**2,071**	**1,510**	**2,027**	**1,333**	**2,040**	**1,409**	**334**	**1,935**	**1,145**	**50,495**
Blanks	25	19	26	10	14	9	3	20	25	532
Field Duplicates	229	161	204	122	155	135	30	221	120	5109
Coarse rejects (Prep Duplicates)	229	161	204	122	155	135	30	221	120	5109
Pulp Duplicates	229	161	204	122	155	135	30	221	120	5109
In House Reference Materials	103	63	114	48	58	47	3	87	91	1,691
External Reference Materials	424	511	693	446	554	421	114	780	459	16,429
Repeats of U_3O_8 assays, Au assays, and ICP analyses	732	334	482	363	789	416	115	781	440	16,062
Umpire Laboratory repeat analyses	100	100	100	100	160	120	0	0	0	1,180
Total Samples	**6,933**	**5,996**	**8,791**	**4,472**	**5,956**	**4,321**	**879**	**6,935**	**3,748**	**166,626**

Sample Type	% of Total Original Samples	% of Resource Samples (i.e., U_3O_8 Assayed Samples)	% of all Samples (Original + QA/QC)	% of Duplicate Samples
Drill holes	--	--	--	--
Total Original Geochem Sample Count[1]	**100%**	**N/A**	**70%**	**N/A**
w/ U_3O_8 assays (+/- Au assays)	70%	100%	49%	N/A
w/ U_3O_8 assays (+ Au assays)	56%	79%	39%	N/A
w/ U_3O_8 assays (no Au assays)	11%	15%	8%	N/A
w/ Au assays (no U_3O_8 assays)	0.1%	0.2%	0.1%	N/A
ICP-OES Only	30%	42.0%	21%	N/A
Total QA/QC Material Frequency	**43%**		**30%**	**N/A**
Blanks	0.5%	1%	0.3%	N/A
Field Duplicates	4%	6%	3%	N/A
Coarse rejects (Prep Duplicates)	4%	6%	3%	N/A
Pulp Duplicates	4%	6%	3%	N/A
In House Reference Materials	1%	2%	1%	N/A

table continues…

Sample Type	% of Total Original Samples	% of Resource Samples (i.e., U_3O_8 Assayed Samples)	% of all Samples (Original + QA/QC)	% of Duplicate Samples
External Reference Materials	14%	20%	10%	N/A
Repeats of U_3O_8 Assays, Au Assays, and ICP Analyses	14%	20%	10%	N/A
Umpire Laboratory Repeat Analyses	1.0%	1.4%	0.7%	12%
Total Samples	--	--	--	--

Note:

1. Counts are for the entire PLS Property excluding seven drill holes from the summer 2021 program, for which geochemical and assay data had not been received at the effective date of this resource estimate update

Results from the QA/QC samples were continually tracked by MSC as certificates for each sample batch were received through 2018, at which time FCU geologists assumed tracking of certificates. If QA/QC samples of a sample batch pass within acceptable limits, the results of the sample batch are imported into the master database.

Results from the QA/QC program are documented in various reports by MSC and FCU. SLR relied on these reports in addition to independent verifications and review of QA/QC data. SLR considers the QA/QC protocols in place at Triple R to be acceptable and in line with standard industry practice and is of the opinion that the resource database is suitable to estimate Mineral Resources for the Triple R deposit.

11.2.1 Certified Reference Material

During the winter 2016 drilling campaign, FCU started a changeover to a new medium uranium (U_3O_8) grade CRM (DH-1A) and implemented a very high grade CRM (CUP-2). The 2018 winter campaign also started to use a gold (Au) CRM (CH-4) for analysis.

CRMs were obtained from CANMET. These include UTS-3 (0.06% U_3O_8), RL-1 (0.237% U_3O_8), DH-1A (0.310% U_3O_8), BL5 (8.36% U_3O_8), and CUP-2 (88.94% U_3O_8) which represent a low, medium, medium, high, and very high grade CRM for uranium, respectively. CRM CH-4 (0.88 g/t Au) is used as the gold grade CRM.

One of each CRM was inserted into the sample batch for each drill hole that intersected mineralization. CRM containers were shaken prior to use to ensure homogeneity and 15 g of material was required per sample. Samples were taken with clearly marked plastic spoons to avoid cross contamination between containers. For holes that did not intersect mineralization, only the low grade reference sample was inserted.

A total of 1,691 CRM samples were submitted by FCU for analysis at SRC. The precision and performance over time of the laboratory is displayed graphically in Figure 11-1 to Figure 11-5 for uranium and Figure 11-6 for gold. The variation from the CRM's mean value in standard deviations (SD) defines the QA/QC variance and is used to determine acceptability of the CRM sample assay. Results within +/- two standard deviations (±2SD) are considered acceptable. Failure criteria for CRM samples are met when either (a) two consecutive samples return values outside two standard deviations from the mean, on the same side of the mean, or (b) any sample returns a value outside three standard deviations (±3SD) from the mean.

On average, less than 1.3% of 1,501 uranium samples were outside the precision limits. Seven samples from UTS-3, five samples from RL-1, four samples from BL-5, and three samples from CUP-2 returned values in excess of 3SD from the respective mean.

The SLR QP is of the opinion that the results of the CRM samples from 2013 to 2021 support the use of samples assayed at the SRC laboratory during this period in Mineral Resource estimation.

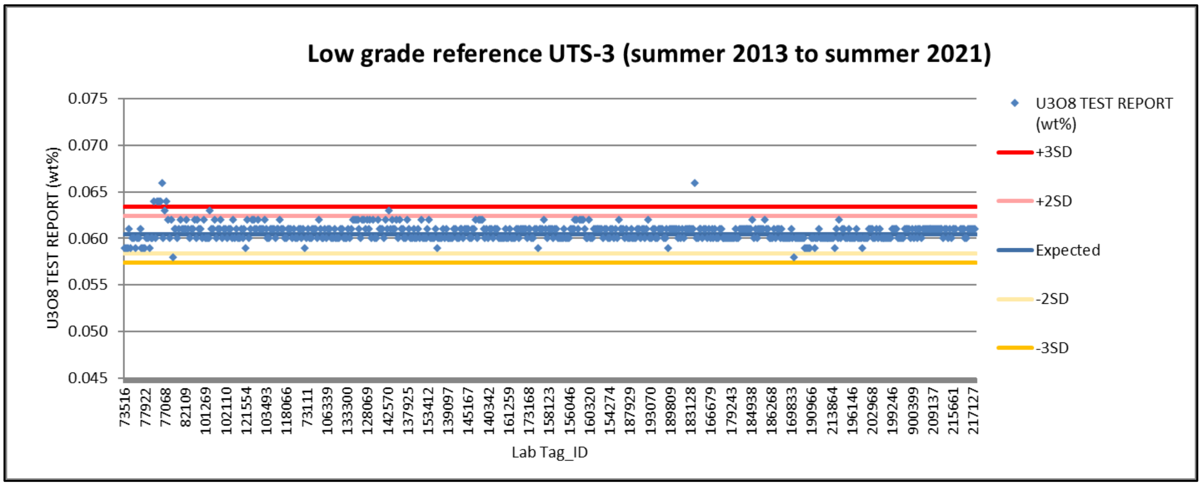


Figure 11-1: CRM Control Chart – UTS-3 (U_3O_8 Low Grade Standard)

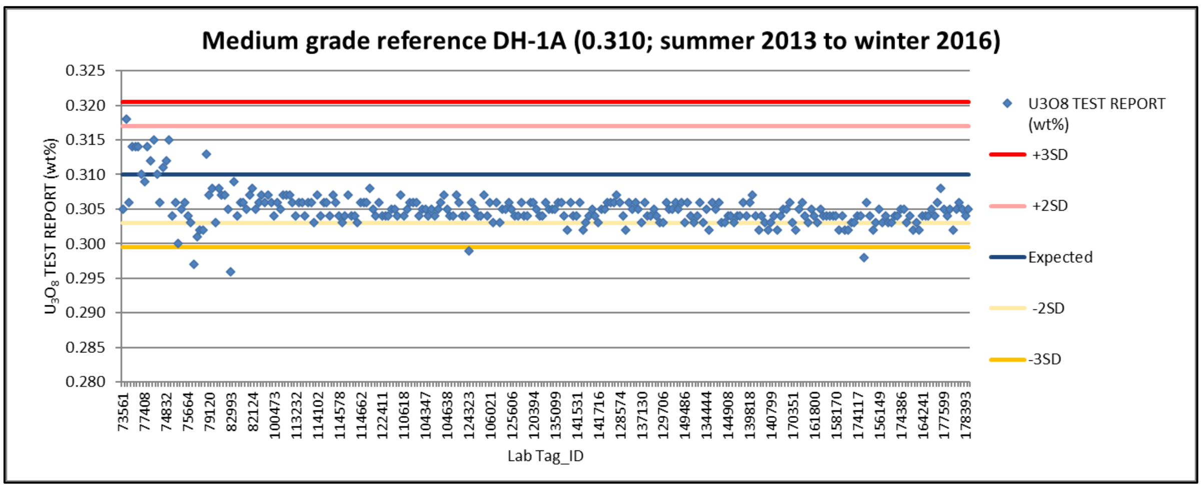


Figure 11-2: CRM Control Chart – DH-1A (U_3O_8 Medium Grade Standard)

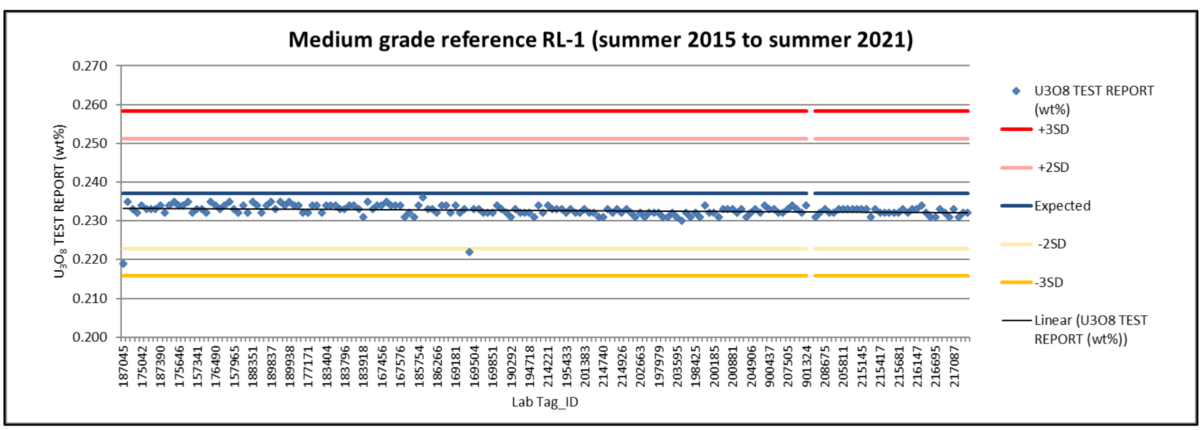


Figure 11-3: CRM Control Chart – RL-1 (U_3O_8 Medium Grade Standard)

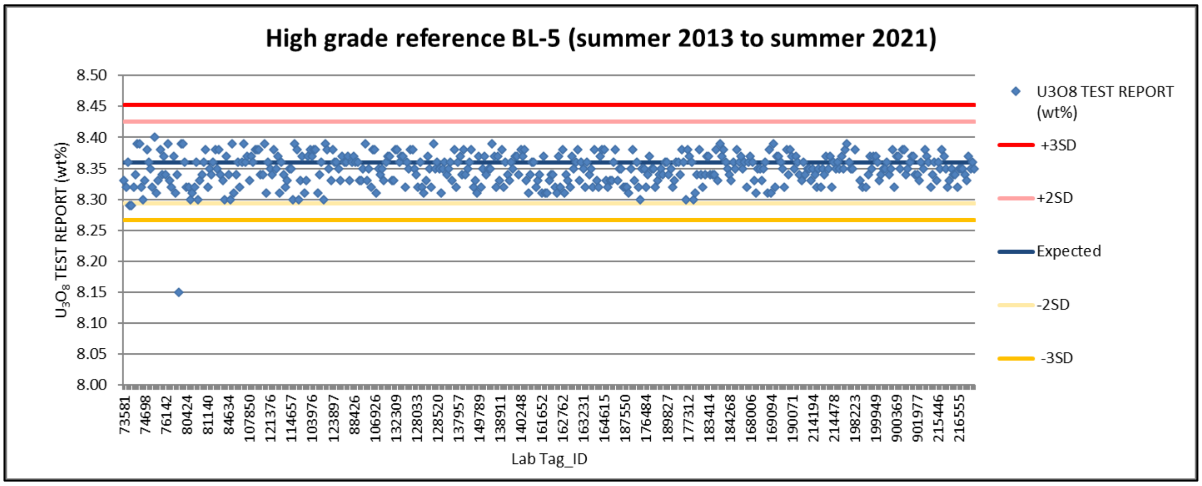


Figure 11-4: CRM Control Chart – BL-5 (U_3O_8 High Grade Standard)

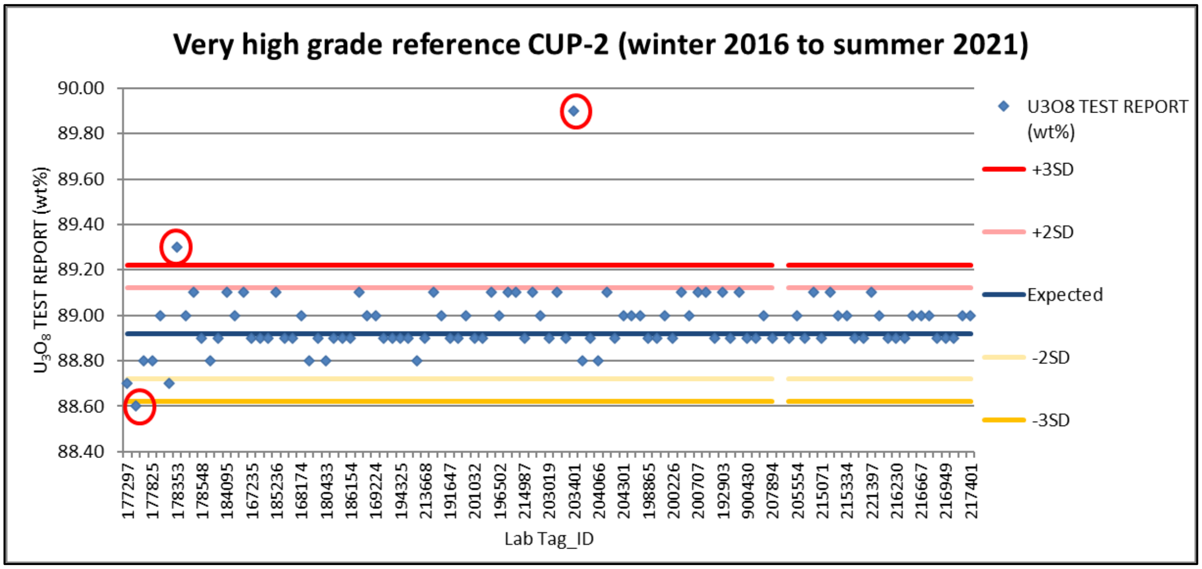


Figure 11-5: CRM Control Chart – CUP-2 (U_3O_8 Very High Grade Standard)

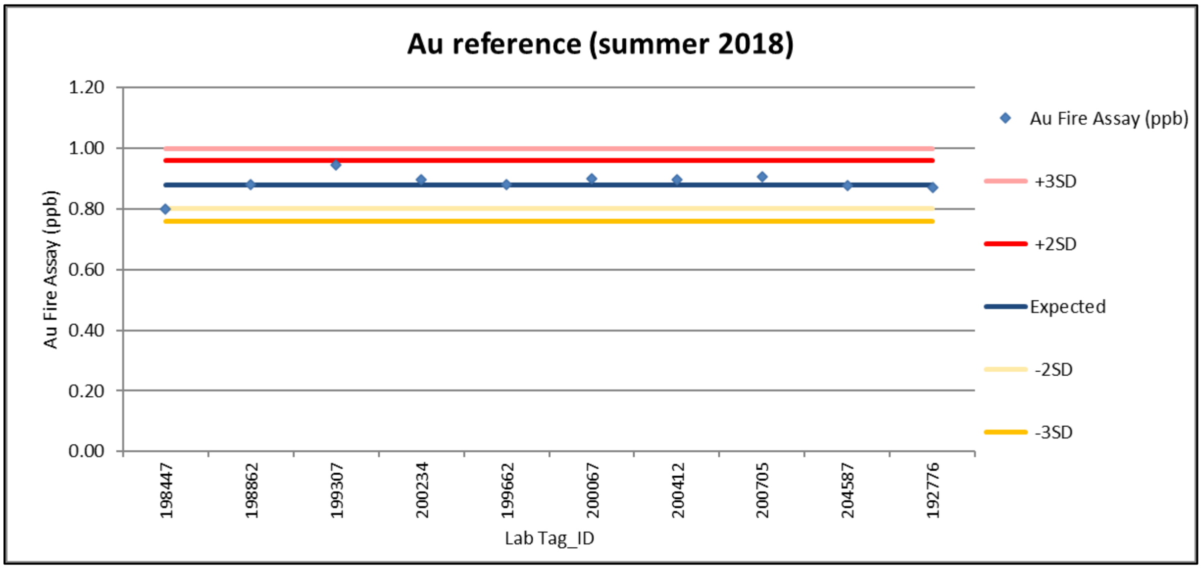
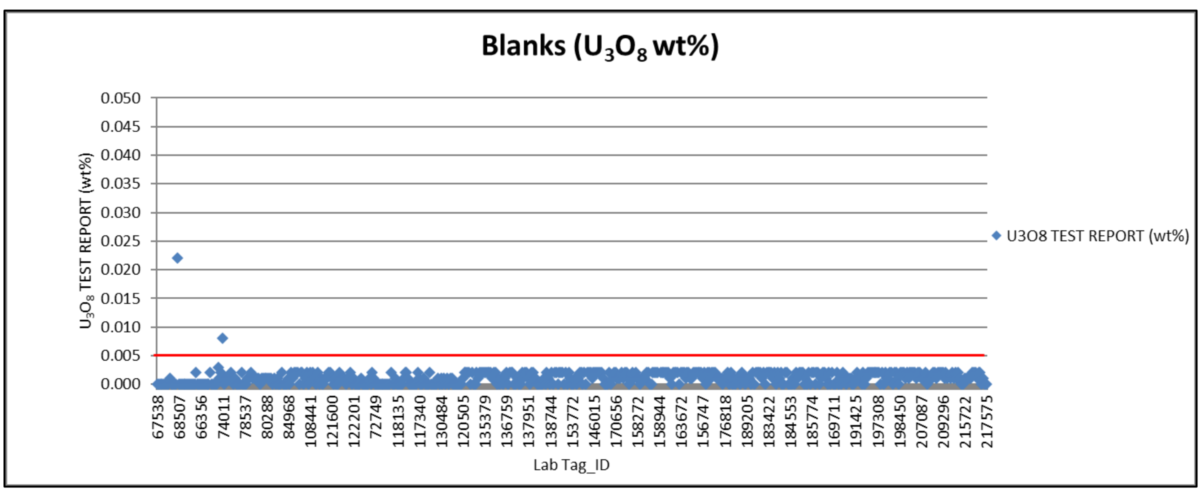


Figure 11-6: CRM Control Chart – CH-4 (Au Very High Grade Standard)

11.2.2 Blanks

Blank material was sourced from the remaining half split core of previously analyzed samples that returned uranium concentrations below detection limits for the 2013 drill program and massive quartz veins intersected on the PLS Property during the 2014 program. One blank sample was inserted for each drill hole that intersects mineralization. Blank reference samples were not submitted for holes that did not intersect mineralization.

Figure 11-7 showing the results of 485 blank samples sorted by increasing sample analysis date. A failure criterion for blank samples is met when a sample returns greater than 0.005% U_3O_8, which is a concentration five times greater than the detection limit of the instrument (0.001% U_3O_8). Two sample failures occurred in 2013 with a maximum of 0.022% U_3O_8. FCU chose not to take corrective steps after reviewing the grades, failure rate, and other QA/QC results from these two batches.



Note: Blank failure if > 5x detection limit (marked by red line)

Figure 11-7: Blank Material Control Chart

11.2.3 Duplicate Samples

Duplicate QC samples measure the precision of the sample preparation through to the analytical stage of chemical analysis. Four types of duplicate samples are submitted:

1. Field duplicates: These are core duplicates split in FCU's core facility. Until 2014, quarter split samples were used (with the original sample representing half of the core, and the filed duplicated representing one quarter of the core, with the last quarter retained in the core box; after 2014, FCU switched to half split, with the original sample comprising of half of the core, and the field duplicated comprising the other half. This was done to improve the statistic by reducing the mean relative differences between the original and duplicate samples. The field duplicate contains all levels of error: core splitting, sample size reduction, sub-sampling of the pulp, and the analytical error. One duplicate is to be inserted for every 20 regular samples. For mineralized drill holes, at least two field duplicate samples should be taken, one from the mineralized zone and one from unmineralized basement. In thicker mineralized zones (more than 20 m), a field duplicate should be taken every 20 samples. For each drill hole, the field duplicates should be retained and inserted into the batch at the end of the hole and assigned sample numbers following on from the last sample in the hole.

2. Preparation duplicates: These are sample splits taken after the coarse crush but before pulverizing. A preparation duplicate should be inserted for each field duplicate submitted. The preparation duplicates are taken by the laboratory. To facilitate this, during sampling, an empty sample bag with an FCU sample tag is inserted into the batch after each field duplicate with instructions for the laboratory to prepare and insert a preparation duplicate of the previous sample.

3. Pulp duplicate: This is a split of the pulp material that is weighed and analyzed separately. Similar to the preparation duplicate, the pulp duplicates are inserted for each field duplicate by inserting an empty bag with an FCU sample tag and instructions for the laboratory to prepare and insert a duplicate of the pulp from the previous sample.

4. Umpire pulp duplicates: Umpire pulp duplicates are submitted to a third-party laboratory to make an additional assessment of laboratory bias. FCU arranged the consignment of 250 preparation and 410 pulp duplicates from the 2015 summer through the 2018 summer drill programs to be analyzed at SGS in Lakefield, Ontario. The sample preparation and analytical methods were similar to those at SRC.

Results from the field, preparation, and pulp duplicate programs are plotted in Figure 11-8 to Figure 11-10. FCU's protocols call for reject and pulp duplicates to be taken from the field duplicate; therefore, reject and pulp results are plotted against the field duplicate results in Figure 11-9 and Figure 11-10. Results are as expected, with better repeatability for the pulps and preparation duplicates.

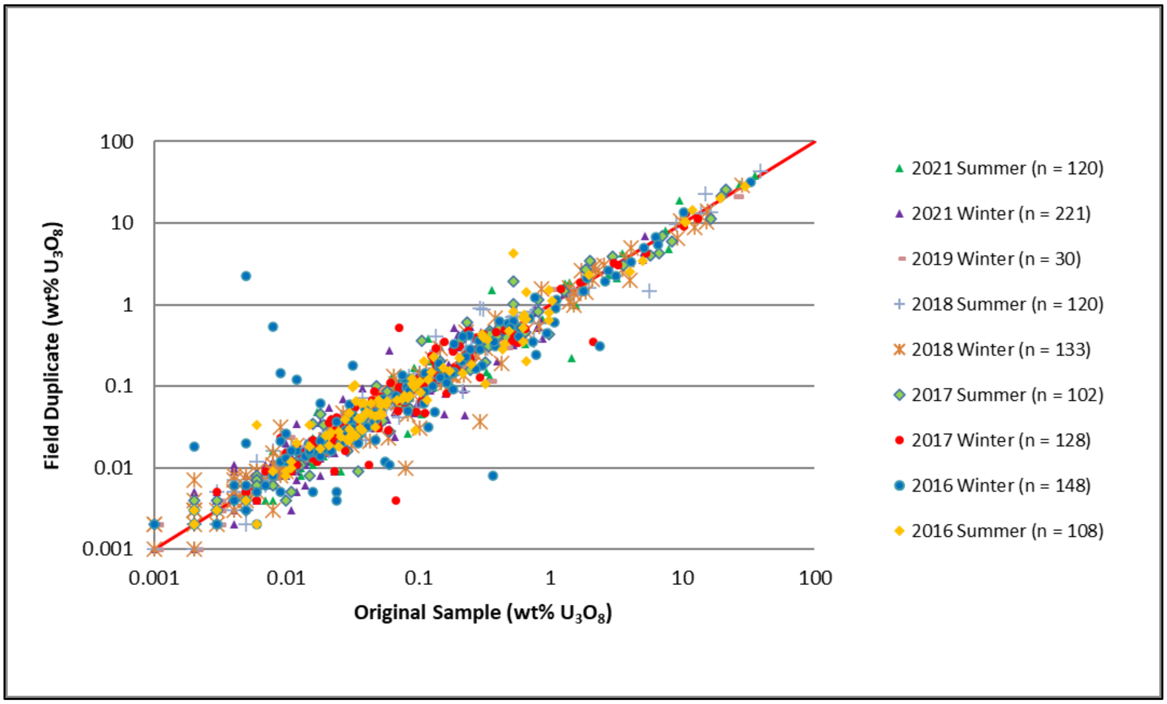


Figure 11-8: Field Duplicate Control Chart (2016 to 2021)

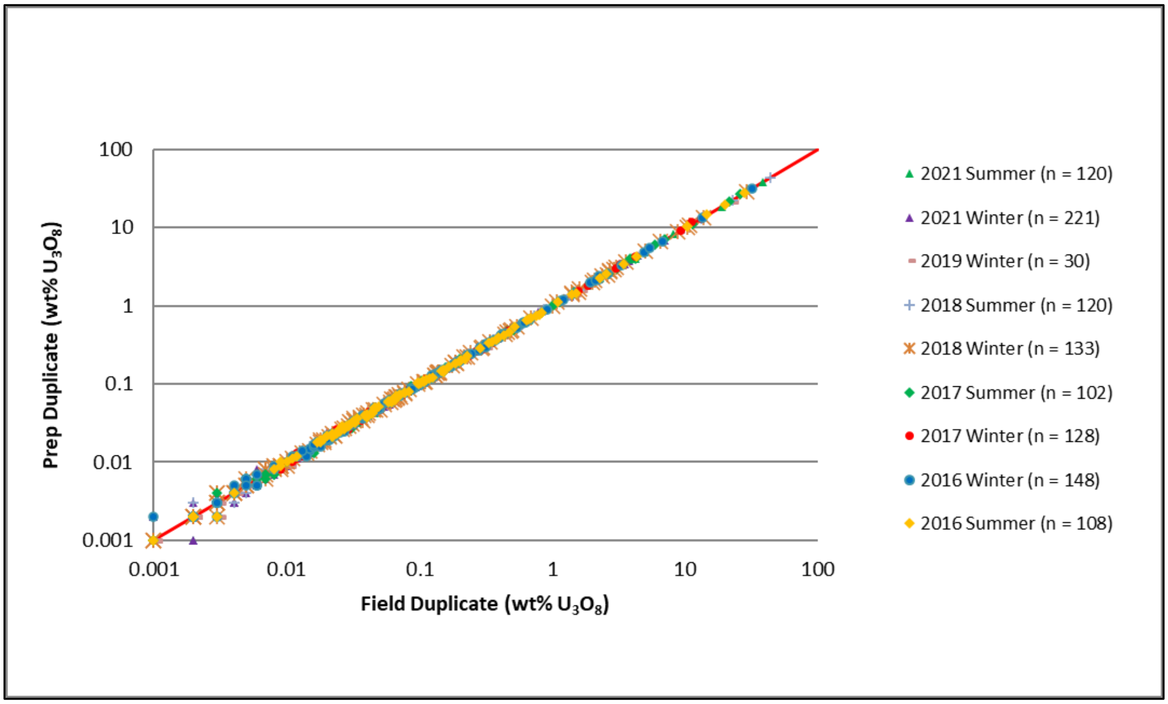


Figure 11-9: Coarse Reject Duplicate Results (2017 to 2021)

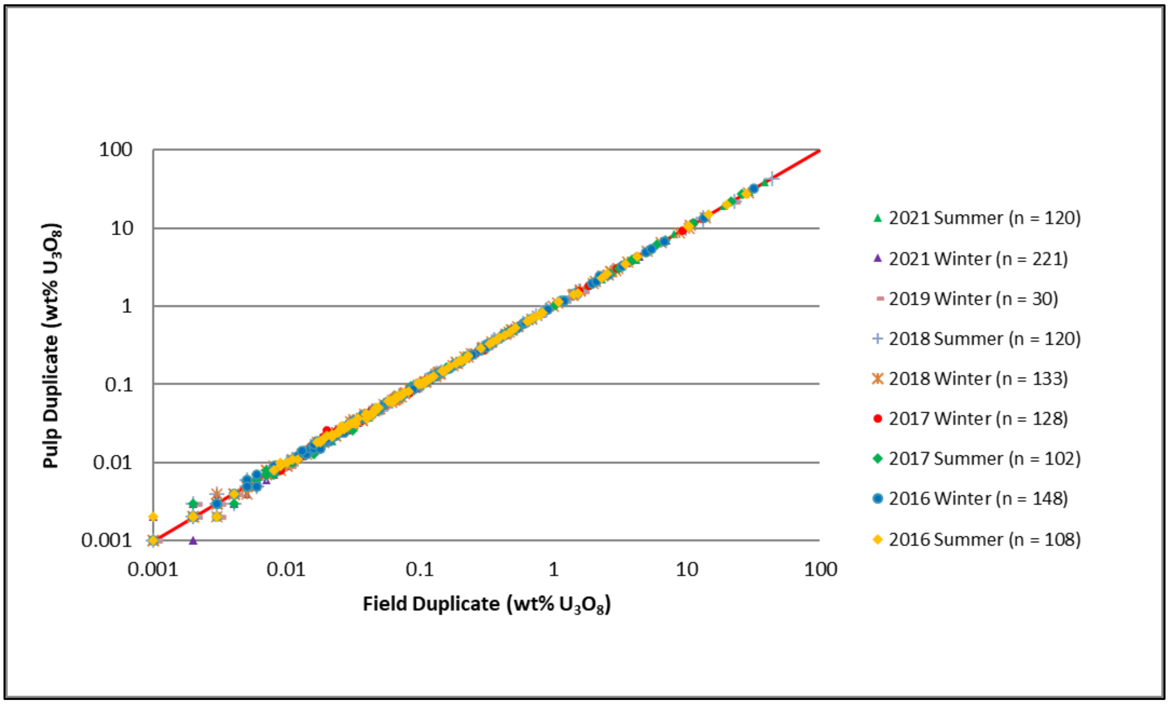


Figure 11-10:Pulp Duplicate Results (2016 to 2021)

11.2.4 SRC Internal QA/QC Program

Quality control was maintained by all instruments at SRC being calibrated with certified materials. Independent of FCU's QA/QC samples, standards were inserted into sample batches at regular intervals by SRC. Within each batch of 40 samples, one to two quality control samples were inserted. All quality control results must be within specified limits otherwise corrective action was taken. If for any reason there was a failure in an analysis, the subgroup affected was reanalyzed.

Five U_3O_8 reference standards were used: BLA2, BL3, BL4A, BL5, and SRCUO2, which have known concentrations of 0.502% U_3O_8, 1.21% U_3O_8, 0.147% U_3O_8, 8.36% U_3O_8, and 1.58% U_3O_8, respectively. Four gold standards were also used by SRC for the Project: OXG83, OXL75, OXL78, and SJ10, which have gold concentrations of 1,002 ppb, 5,876 ppb, 5,876 ppb, and 2,643 ppb, respectively. With the exception of SRCUO2 (produced in-house at SRC), all reference materials are certified and provided by CANMET.

SRC has developed and implemented a laboratory management system which operates in accordance with ISO/IEC 17025:2005 (CAN-P-4E), General Requirements for the Competence of Mineral Testing and Calibration Laboratories. The laboratory also participates in a certified interlaboratory testing program, CCRMP/PTP-MAL, for gold using lead fusion fire assay with an AAS finish.

All processes performed at the laboratory are subject to a strict audit program, which is performed by approved trained professionals. SRC is independent of FCU.

Based on the data validation and the results of the standard, blank, and duplicate analyses, the SLR QP is of the opinion that the assay and bulk density databases are of sufficient quality for Mineral Resource estimation at the Triple R deposit.

11.2.5 Secondary Laboratory Check

Between the fall of 2011 and the summer of 2018, a total of 1,180 samples were sent to SGS laboratory to measure the accuracy of the results from SRC. SLR reviewed the results and found high degrees of correlation and relative bias within acceptance limits (Figure 11-11).

The mean relative difference for all duplicate samples is 5.10%. One duplicate sample with an original result of 34.6% U_3O_8 from SRC returned 28.4% U_3O_8 from SGS. MSC suggests that this difference may be due to analytical error or slight differences in the analytical methodology combined with complications arising from the carbonaceous material during the digestion stage.

No samples since the summer of 2018 have been sent to the SGS laboratory.

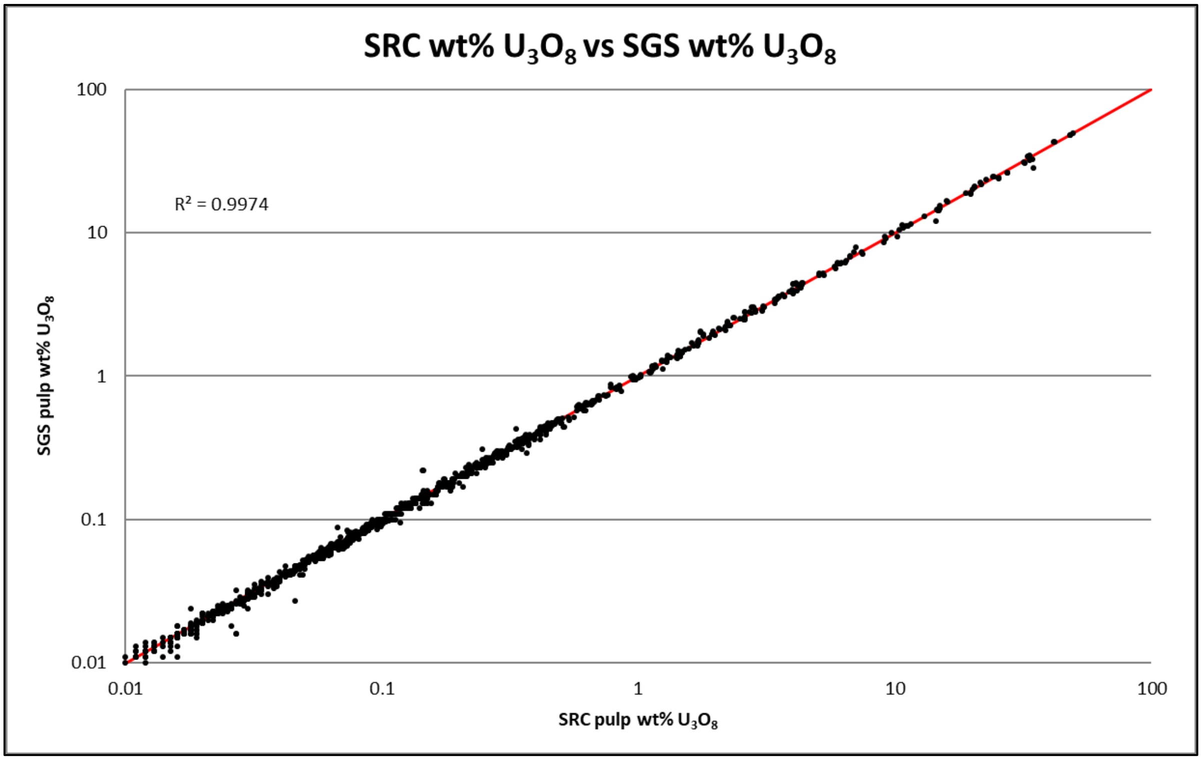


Figure 11-11: SRC vs. SGS Duplicate Results

The SLR QP is of the opinion that the secondary laboratory checks are of sufficient quality for Mineral Resource estimation at the Triple R deposit. SLR also recommends resuming the use of a secondary laboratory check with any future exploration drilling. Umpire laboratory duplicates have never failed QA/QC and triggered a repeat analysis at either laboratory.

11.3 Sample Security and Confidentiality

Drill core was delivered directly to FCU’s core handling facility located on the PLS Property. After logging, splitting, and bagging, core samples for analysis were stored in a secured shipping container at the same facility. The samples were picked up on site by Marsh Expediting and transported by road to La Ronge before transhipment to

SRC in Saskatoon. The shipping container was kept locked or under direct supervision of FCU personnel. A sample transmittal form was prepared that identified each batch of samples.

SRC considers customer confidentially and security of utmost importance and takes appropriate steps to protect the integrity of sample processing at all stages from sample storage and handling to transmission of results. All electronic information is password protected and backed up on a daily basis. Electronic results are transmitted with additional security features. Access to SRC's premises is restricted by an electronic security system. The facilities at the main laboratory are regularly patrolled by security guards 24 h/d.

Official results are provided as a series of Adobe PDF files. A Microsoft Excel spreadsheet file containing only the analytical results is also provided. These files are sent using a secured password protected compressed file.

In the SLR QP's opinion, the sampling methods, chain of custody procedures, and analytical techniques are appropriate and meet acceptable industry standards, and results are appropriate to estimate Mineral Resources.

12.0 DATA VERIFICATION

SLR reviewed and verified the resource database including: a review of the QA/QC methods and results, verifying assay certificates against the database assay table, standard database validation tests, and three site visits including drill core review. No limitations were placed on SLR's data verification process. The review of the QA/QC program and results is presented in Section 11, Sample Preparation, Analyses, and Security.

SLR considers the resource database to be reliable and appropriate to prepare a Mineral Resource estimate.

12.1 Site Visit and Core Review

Mr. Mark B. Mathisen, CPG, visited the PLS Property on August 6 to 8, 2018, during the summer drill programs in connection with the 2019 Triple R Mineral Resource estimate. During the visit, Mr. Mathisen visited barge-based drills and reviewed all core handling, logging, sampling, and storage procedures.

SLR examined core from several drill holes and compared observations with assay results and descriptive log records made by FCU geologists. As part of the review, SLR verified the occurrences of mineralization visually and by way of a handheld scintillometer. Holes reviewed included but were not limited to: PLS13-64, PLS13-75, PLS14-129, PLS14-183, and PLS14-186. There are no known outcrops of significance on the PLS Property to visit.

12.2 Database Validation

SLR performed the following digital queries. No significant issues were identified.

- Header table: Searched for incorrect or duplicate collar coordinates and duplicate hole IDs.
- Survey table: Searched for duplicate entries, survey points past the specified maximum depth in the collar table, and abnormal dips and azimuths.
- Core recovery table: Searched for core recoveries greater than 100% or less than 80%, overlapping intervals, missing collar data, negative lengths, and data points past the specified maximum depth in the collar table.
- Lithology: Searched for duplicate entries, intervals past the specified maximum depth in the collar table, overlapping intervals, negative lengths, missing collar data, missing intervals, and incorrect logging codes.
- Geochemical and assay table: Searched for duplicate entries, sample intervals past the specified maximum depth, negative lengths, overlapping intervals, sampling lengths exceeding tolerance levels, missing collar data, missing intervals, and duplicated sample IDs.

No significant issues were identified.

12.3 Independent Verification of Assay Table

For 2018, the geochemical table contained 5,733 records. SLR verified approximately 3,702 records representing 53% of the data for gold and uranium values against 52 different laboratory certificates. No discrepancies were found.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

This section summarizes the key metallurgical test work undertaken on core samples extracted from the PLS Uranium Property. There were three separate test work campaigns supporting the three phases of project advancement, namely:

- Scoping Study: 2013 to 2014
- PFS: 2017 to 2018
- FS: 2021 to 2022

13.1 Scoping Study: 2013 to 2014

FCU commissioned MSC to manage a metallurgy and mineralogy study of the PLS uranium deposit in 2013. The objectives of this study were:

- To investigate the metallurgical characteristics of the deposit and their relationships with ore and gangue mineralogy, and
- To assess how these characteristics and relationships vary spatially and between different rock types.

The analytical work was performed by the SRC, and the mineralogical work was undertaken by SGS.

13.1.1 Sample Selection

MSC selected 41 samples spatially representative of the PLS basement lithologies, uranium grades, and mineralogy as of December 2013. Five composite samples representative of the mineralized areas were produced. Figure 13-1 is a cross-section showing the distribution of the samples and composites used in the scoping study (section looking northwest). The sample dots are colour-coded based on their U_3O_8 % by mass. The red line represents the mineralization outline, and the light brown indicates high proportions of carbonaceous matter. The uranium and gold analyses of the five composites and combined master composite are presented in Table 13-1.

Table 13-1: Uranium and Gold Concentration in the Five Composites and Master Composite

Parameter	Composite 1	Composite 2	Composite 3	Composite 4	Composite 5	Master Composite
Mass (kg)	4.2	5.3	5.4	5.4	6.0	9.0
U_3O_8 (ppm)	27,600	7,330	23,600	34,200	12,400	20,900
Au (ppm)	0.044	0.017	1.840	2.330	0.609	1.100

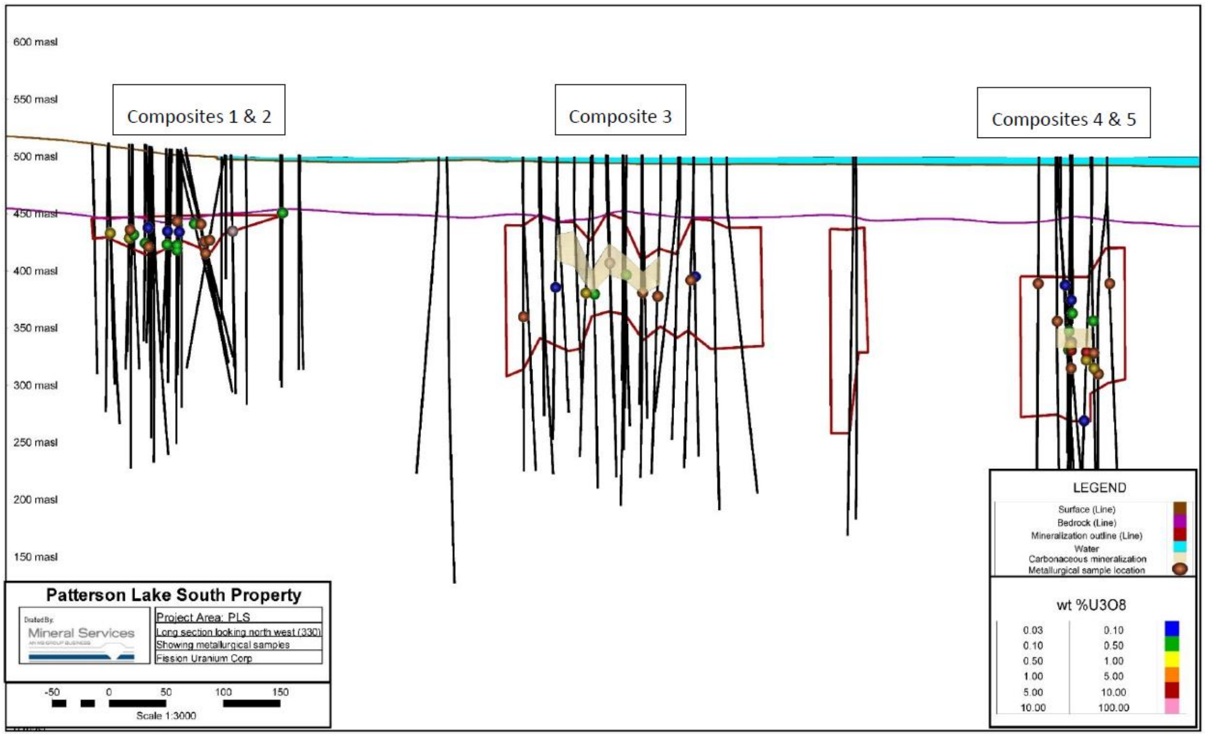


Figure 13-1: Distribution of Samples and Composites

13.1.2 Key Results

- Uranium occurs in all the samples, mainly as uraninite (UO_2)/uranophane ($Ca(UO_2)_2(SiO_3OH)_2 \cdot 5H_2O$), with lesser coffinite ($U(SiO_4)_{1-x}(OH)_{4x}$), brannerite [$(U,Ca,Ce)(Ti,Fe)_2O_6$], and U-Pb minerals; fourmarierite ($Pb(UO_2)_4O_3(OH)_4 \cdot 4H_2O$), metaschoepite ($UO_3 \cdot <2H_2O$), umohoite [$(UO_2)(MoO_4) \cdot 2H_2O$], and vandendriesscheite ($PbU_7O_{22} \cdot 12H_2O$). Other (U, Pb)-oxides are possibly present.

- Free and liberated U-minerals account for 49% to 60% of all U-minerals in composite samples. The relative abundance of exposed U-minerals (i.e., unlocked U-minerals, strictly surrounded by <100 % gangue minerals) varies from 97.0% to 98.9%.

- Non-liberated U-minerals typically occur as complex intergrowths with silicates, carbonates, or “soft” silicates (clays/chlorite/micas).

- All composite samples contain carbon, most of which is accounted for by the presence of graphite and/or carbonate.

- The uranium head grade of the composites ranged from 0.7% U_3O_8 to 3.4% U_3O_8, with an average grade of 1.78% U.

- Uranium extraction ranged from 95% to 99.4%, with an average of 98.1%, when leached at a grind size of P_{85} = 250 µm, a temperature of 45 to 55°C, free acid levels of 25 g/L, a final ORP of 450 mV, and a leach retention time of six hours.

- Regarding lithological variation, pelitic composite samples are characterized by higher uranium grades, higher carbonaceous matter content, and similar or lower uranium recovery compared to the semi-pelitic and quarzitic composite samples.

13.1.3 Leach Tests on Master Composite

One master composite sample was prepared using the five sub-composite samples according to the ratio of 1:1:2:1:1. The master composite sample contains 2.09% U_3O_8.

- A uranium extraction of 98.4% was achieved in six hours at 50°C and atmospheric pressure with 54 kg acid addition per t of leach feed.
- Leach efficiency was similar for the grind size (85% passing, P_{85}), ranging between 75 µm and 250 µm, indicating that uranium minerals can be liberated at coarse grind size.
- It was found that leaching at a temperature higher than 50°C did not improve uranium leach extraction but can lead to high reagent consumption and gangue dissolution. Leaching at a temperature lower than 50°C slightly slowed the uranium extraction kinetics.
- It was found that a free acid level of 25 g/L is necessary to maximize the uranium leach extraction. The acid consumption was about 30 kg/t of leach feed when 54 kg acid per t of leach feed was added, equivalent to 1.5 kg of acid consumed per kg of U_3O_8 leached.
- There was no significant difference in uranium leaching when the ORP was controlled in the 450 to 550 mV range. Sodium chlorate ($NaClO_3$), hydrogen peroxide (H_2O_2), and ferric sulphate ($Fe_2(SO_4)_3$) are all effective oxidants and can be used to achieve the ORP target. Oxidant consumption was 7.2 kg/t of leach head for $NaClO_3$ and 2.9 kg/t for H_2O_2.

13.2 Prefeasibility Study: 2017 to 2018

In 2017, FCU commissioned Melis Engineering Ltd. (Melis) to manage the metallurgical test work program as a component of the PFS on the PLS Project. The test work was conducted by the SGS laboratory at Lakefield, Ontario. The objective of the test program was to provide the process design data for the PFS.

The test program encompassed test composites preparation and analysis, comminution testing, leaching, liquid-solid separation, SX, precipitation of yellowcake, tailings and effluent treatment, physical and chemical characterization of tailings, and environmental testing of prepared tailings.

13.2.1 Sample Selection

In 2017, three purpose-drilled HQ metallurgical test holes were completed in the central part of FCU's Triple R uranium deposit. The positions and intersections of the three holes are illustrated in Figure 13-2. A total of 219 mineralized samples and 45 unmineralized gangue samples were collected.

Twenty-six sub-composite samples were assembled as follows:

- 12 composites spatially representing different lithologies, ranging from 0.51% to 25.9% U_3O_8, with an average of 5.13% U_3O_8.
- Two composites were prepared representative of shallow open pit accessible material and deeper underground mining accessible material.
- Three composites were prepared representative of conceptual mine schedule U_3O_8 grades:
  - Year 1: 2.84%
  - Year 2: 2.08%

- Year 6: 1.60%

- One composite near COG of 0.33% U_3O_8.
- Four comminution test composites, one per mineralized lithology.
- Four gangue composites, one per laterally spatial domain.

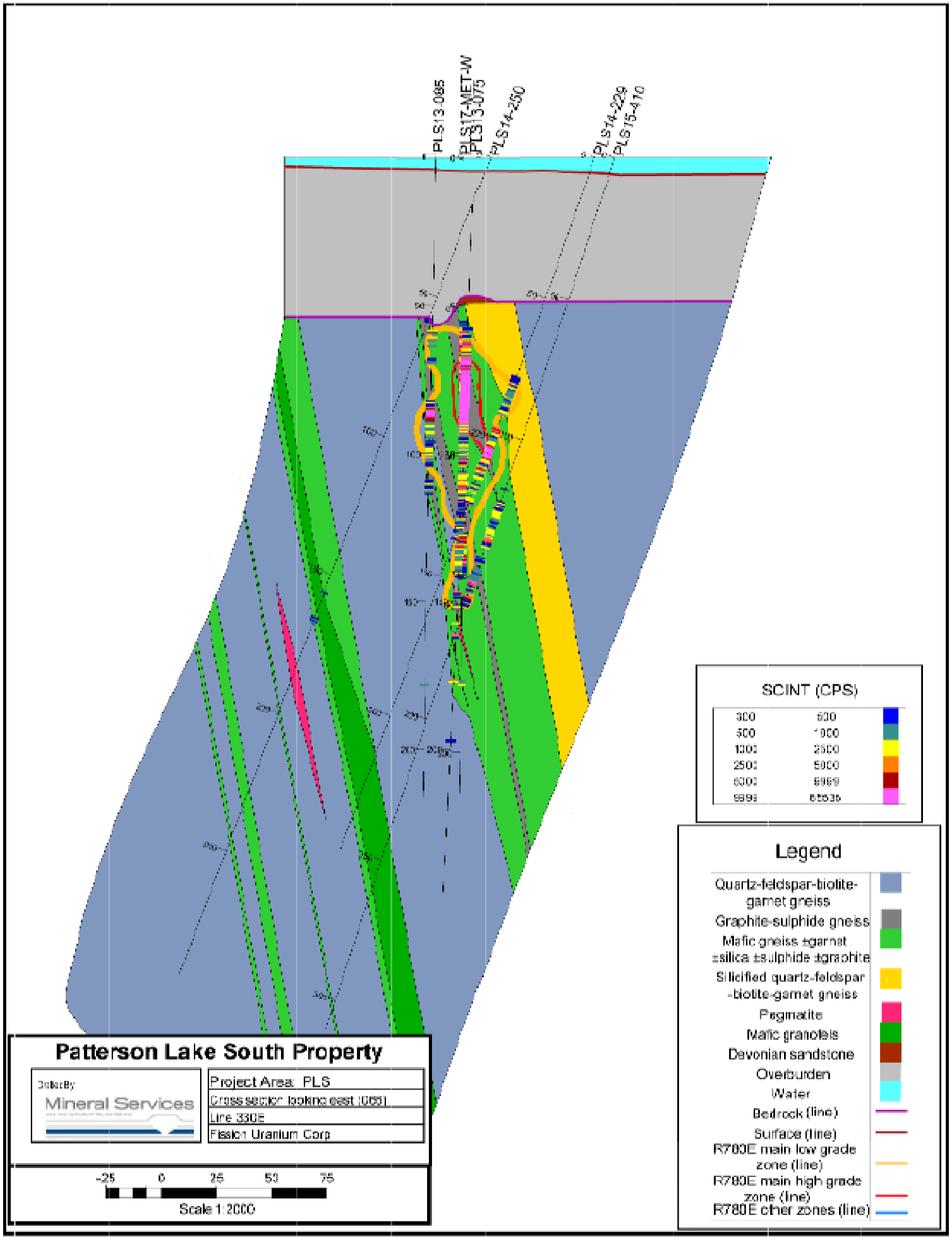


Figure 13-2: Location of Metallurgical Drill Holes – Section S330E

13.2.2 Results from the Test Program

The key test work results are summarized in the following subsections.

Comminution results

Three blended lithology composites and one gangue composite were subjected to grinding tests, including:

- CEET Crusher Index (C_i)
- SAG Power Index (SPI)
- Ball Mill Bond Work Index (BWI_{BM})
- Abrasion Index (A_i)

The samples tested were categorized as very soft to soft with respect to SAG milling and soft to moderately soft for ball milling, as listed in Table 13-2**.**

Table 13-2: Comminution Index Test Results for Lithology and Gangue Composites

Composite	Lithology	CEET C_i	SPI (minutes)	BWI_{BM} (kWh/t)	A_i (g)
CHG	Carbonaceous High Grade	5.9	18.1	11.2	0.004
QFG	Quartz-Feldspar-Gneiss	3.4	28.4	12.3	0.116
GG	Graphitic Gneiss	2.3	17.6	10.3	0.079
G	Unmineralized Gangue	1.3	36.3	12.7	0.279

Leaching Results

Leach tests were carried out on 22 composite samples. The head analysis of each sample is presented in Table 13-3**.** The following leach conditions were identified as optimum:

- A grind size P_{80} of 150 µm
- A leach pulp density of 45% to 50% solids (w/w)
- An eight-hour leach retention time, allowing some flexibility for head grade variations
- A leach temperature of 50°C
- A free acid of 15 g H_2SO_4/L in the first six hours of leaching, tapering to 10 g H_2SO_4/L by the end of the leach
- A target ORP of 450 to 500 mV using sodium chlorate or oxygen as an oxidant
- The average reagent consumptions were 64.8 kg H_2SO_4/t and 3.1 kg $NaClO_3$/t of leach feed
- Uranium extraction ranged from 94.6% to 98.8%
- Projected net recoveries were 97.1% for 2.0% U_3O_8 head grade and 94.9% for 0.5% U_3O_8 head grade.

Table 13-3: Head Analysis of 22 Composite Samples Used for Leach Test

Composite	U_3O_8 (%)	Au (g/t)	As (%)	Co (%)	Mo (%)	Ni (%)	Pb (%)	Y (%)
Lithology Sub-Composites								
ECHG	25.90	0.37	<0.008	0.007	0.264	0.016	0.661	0.037
EUQFG	2.16	0.29	<0.008	0.001	0.077	0.008	0.061	0.009
EMQFG	1.11	0.84	<0.008	0.002	0.092	0.009	0.053	0.009
ELQFG	0.51	0.23	<0.008	0.004	0.035	0.015	0.026	0.008
CCHG	17.5	0.04	<0.008	0.008	0.168	0.009	0.332	0.019
CUGG	2.57	0.09	<0.008	0.004	0.005	0.018	0.070	0.029
CMGG	0.80	0.50	0.024	0.018	0.085	0.010	0.058	0.002
CLGG	0.53	1.53	<0.008	0.003	0.105	0.026	0.038	0.008
WCHG	4.48	4.95	<0.008	0.005	0.706	0.016	0.144	0.021
WUGG	3.88	0.29	<0.008	0.010	0.098	0.025	0.136	0.029
WMGG	1.16	0.02	<0.008	0.002	0.177	0.007	0.033	0.007
WLGG	0.91	0.84	<0.008	0.002	0.072	0.005	0.041	0.015
Blended Composites								
O/P[(1)]	2.04/2.33	0.41	<0.008	0.008	0.095	0.012	0.073	0.011
U/G	0.65	0.77	<0.008	0.004	0.067	0.014	0.030	0.010
YR1	2.84	0.40	<0.008	0.006	0.100	0.011	0.082	0.012
YR3	2.08	0.24	<0.008	0.007	0.077	0.011	0.060	0.010
YR6	1.60	0.10	<0.008	0.006	0.078	0.012	0.054	0.011
LG	0.33	0.50	<0.008	0.008	0.029	0.010	0.016	0.006
Gangue Composites								
EG	0.02	0.02	<0.008	<0.0004	0.004	0.004	0.002	0.004
CG	0.14	0.08	<0.008	0.005	0.040	0.011	0.010	0.007
WG	0.06	<0.02	<0.008	0.017	0.015	0.015	0.011	0.007
G	0.09	0.06	<0.008	0.008	0.021	0.010	0.009	0.006

Note: [1] The calculated head grade from 11 leach tests on the open pit composite averages 2.75% U_3O_8

The analysis of the pregnant leach solution generated from the variability leach tests on lithology composites is presented in Table 13-4. The results indicate that the pregnant leach solution has a very low tenor of deleterious metals and is suitable for standard uranium concentration and purification by amine-based SX.

Table 13-4: Pregnant Leach Solution Analysis of Variability Leach Test on Lithology Composites

Analyte	VAL-1	VAL-2	VAL-3	VAL-4	VAL-5	VAL-6	VAL-7	VAL-8	VAL-9	VAL-10	VAL-11	VAL-12
Ag	<0.08	<2	<0.4	<0.2	<0.6	<2	<2	<0.2	<0.9	<2	<0.4	<0.6
Al	16,900	7,860	1,440	2,700	3,070	11,100	592	1,880	951	17,700	1,420	944
As	<20	<20	<20	35	<70	<20	91	13	<50	69	<20	<70
Ba	<0.3	<0.3	<0.08	<0.07	<0.1	<0.3	<0.3	<0.07	<0.07	<0.3	<0.08	<0.1
Be	1.96	0.03	<0.4	0.07	<1	0.6	0.1	<0.05	0.36	2.06	<0.4	<1
Bi	<40	<40	<4	2	29	<40	9	40	<70	15	13	12
Ca	600	699	672	822	1,530	658	296	907	1,110	665	754	593
Cd	<0.5	<0.4	<0.09	<0.09	<0.09	<0.4	<0.2	<0.09	<0.09	<0.2	<0.09	<0.09
Co	<20	<20	10.6	15	34.8	<20	62	7.4	14.2	67	6	6.4
Cr	59.7	35.1	27.3	16.5	36.8	53.4	20.1	17.2	33.9	39.5	18.1	22.9
Cu	<100	<100	40	15.7	543	<100	6,520	27.1	<30	152	<20	17
Fe	18,400	12,900	1,240	3,240	2,600	20,000	1,460	1,490	1,080	20,500	2,020	1,060
Fe^{2+}	14,140	8,610	909	2,043	2,202	14,260	<5	445	843	10,380	830	544
Fe^{3+}	4,260	4,290	331	1,197	398	5,740	1,460	1,045	237	10,120	1,190	516
K	1,400	471	312	298	658	159	137	170	363	601	348	340
Li	98	34	10	8	33	58	<2	9	7	77	8	4
Mg	6,420	4,300	800	1,430	766	7,470	89	1,250	323	9,250	766	427
Mn	488	42.1	17.3	40.6	62	72.1	10.5	10.8	22.2	65.5	12.5	7.8
Mo	744	247	269	73	155	5.9	464	343	1,430	395	500	310
Na	3,350	401	322	383	2,390	705	1,120	341	779	1,220	296	301
Ni	37	20	18	42	66	43	53	42	58	132	34	15
P	43	35	<30	21	192	134	9	84	49	1,530	<30	17

table continues…

Analyte	VAL-1	VAL-2	VAL-3	VAL-4	VAL-5	VAL-6	VAL-7	VAL-8	VAL-9	VAL-10	VAL-11	VAL-12
Pb	24	27	<8	18	25	33	24	17	8	70	<8	7
Sb	<3	<3	<3	<2	<3	<3	<3	<2	<3	<3	<3	<3
Se	<3	<3	<3	<3	<3	<3	<3	4	<3	<3	<3	<3
Sn	<30	<20	<2	<2	31	<20	<7	<2	<6	<7	<2	<20
Sr	<7	13	6.22	8.35	<10	16	2.24	11.3	6.68	7.46	12.3	<10
Ti	173	<8	<5	6	<50	<8	9.43	<3	<20	32.7	<5	<50
Tl	<3	<3	<3	<3	<3	<3	<3	<3	<3	<3	<3	<3
U	162,000	13,200	9,300	4,560	120,000	17,300	7,620	5,120	27,600	38,500	9,110	7,400
V	585	161	120	72.3	487	203	62.1	97	299	257	139	94
W	<4	<5	<5	<2	<9	<5	<3	<2	<5	4	<5	<9
Y	305	49.7	59.6	46.7	186	96.8	17.3	42.5	148	339	63	116
Zn	<3	<6	<2	2	<2	<6	3	<2	<2	37	<2	<3

Counter Current Decantation

Liquid/solid separation thickening and rheology tests were completed on leach residue from the bulk leach test on the composites. CCD simulations using the test data showed that a pregnant leach solution to ore ratio (P/O) of 3.0:1 and a total of six thickeners would provide a target soluble loss of ≤ 0.5% in the CCD circuit.

Solvent Extraction Results

SX batch shake-out tests and continuous mini SX extraction tests were completed on two pregnant leach solution composites diluted to 3.5:1.0 to simulate pregnant leach solution produced by counter-current decantation. The isotherms and mini continuous SX extraction tests were conducted using two extraction stages at a 1:1 organic to aqueous ratio with organic composed of alamine 336 and tridecanol in a kerosene carrier.

Loaded organic stripping tests were completed with both strong acid (425g H_2SO_4/L) and ammonium sulphate (180g $(NH_4)_2SO_4$/L). A five-stage strip was used for both strip chemistries. The results of the stripping tests indicated that strong acid stripping is more efficient (99.6%) than ammonium sulphate stripping (95%), with stripping essentially being complete after four stages.

The SX test work indicated that the flowsheet would include four-stage extraction, two-stage scrub, five-stage strong acid strip, one-stage acid recovery, and one-stage regeneration.

Gypsum and Yellowcake Precipitation

The metallurgical tests included gypsum precipitation from the loaded strong acid strip solution to avoid final product contamination with excess sulphate, followed by uranium peroxide precipitation from the purified solution. The final product quality from the strong acid strip circuit was highest at 81.2% U_3O_8 compared to 69% U_3O_8 from the ammonium sulphate circuit. An overall uranium recovery of 97.1% was projected for a feed grade of 2% U_3O_8 and 94.9% for 0.5% U_3O_8 head grade.

Gold Recovery

Flotation and carbon-in-leach (CIL) tests were completed to quantify potential gold recovery from the PLS mineralization on four composite samples. The average gold head grade was 0.52 g/t, and the average gold recovery using flotation/CIL and direct CIL was 78.4%. A high-level economic analysis indicated that it was not feasible to recover gold at the parameters and costs estimated at that time.

13.3 Feasibility Study: 2021 to 2022

In January 2021, FCU appointed Melis to oversee and lead a new metallurgical test work program on samples representative of the mill feed proposed for the potential mine schedule.

Historical test work was conducted on core samples from the 780 East Zone. However, the proposed mine plan is based on a mill feed mass ratio of 75% R780E material and 25% R840W material. The objective of the new test program was to confirm metallurgical response data for R780E and R840W mineralized samples and provide process design data for the FS.

The test work program encompassed:

- Preparation and analyses of test composites, including mineralogy

- Comminution testing
- Leaching and liquid-solid separation
- SX of uranium
- Gypsum removal and precipitation of uranium yellowcake
- Testing of gold and silver recovery
- Tailings and effluent treatment, tailings physical and chemical characterization, and environmental testing on prepared tailings

SGS Lakefield was appointed to conduct the metallurgical test work. The SRC in Saskatoon assisted with the preparation of samples and sub-composites.

13.3.1 Sample Selection, Composite Preparation, and Analysis

In 2021 four purpose-drilled holes were completed in the R840W zone, as depicted in Figure 13-3. A total of 189 HQ core samples were generated to prepare metallurgical test composites. The drill holes selected spatially represented the mineralization in the west (W) and east (E) locations of the R840W zone and were designated as upper (U), middle (M) and lower (L) according to drill hole intersection with two holes in the west area of the zone and two holes in the east area of the zone. Two bulk gangue composites from the R780E zone and R840W zone were also collected to dilute test composites to achieve target grades. An existing mineralized core from the R780E zone in storage at SGS was used to prepare R780E grade variation composites and an R780/R840 75%/25% composite blend with a target ROM grade of 1.54% U_3O_8.

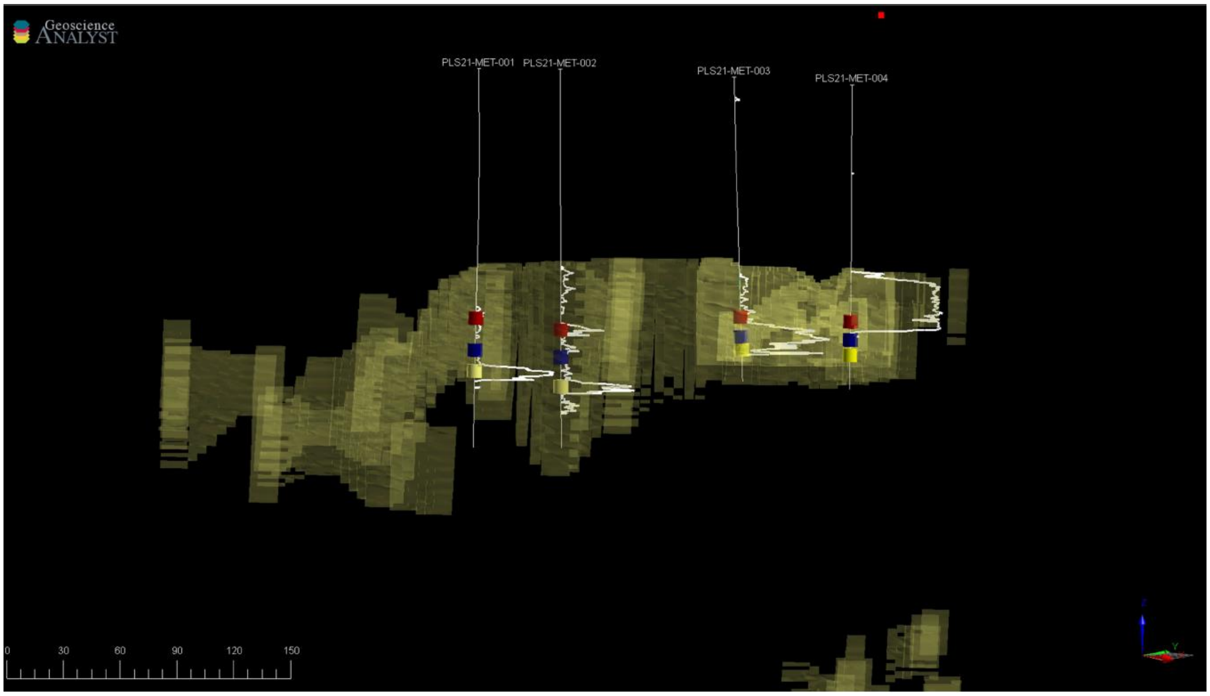


Figure 13-3: Location of Four Purpose Drill Holes – Zone R840W

Table 13-5 presents the analytical results of the metallurgical composites used in the metallurgical test program.

Table 13-5: Key Element Analysis of FS Overall Metallurgical Composite Samples

Analyte	Unit	R780 Overall	R840 Overall	R780/R840 Overall	R780/R840 Low Grade	R780/R840 High Grade
U_3O_8	%	1.74	1.15	1.56	1.37	1.92
Au	g/t	0.27	0.26	0.34	0.37	0.36
Ag	g/t	1.8	1.1	1.6	1.5	2.0
As	ppm	<30	<30	< 30	<30	<30
Co	ppm	34	63	41	35	44
Cu	ppm	368	313	326	268	368
Li	ppm	195	127	184	159	183
Mo	ppm	582	83	449	391	418
Ni	ppm	122	136	121	106	140
Pb	ppm	490	458	454	385	558
Sb	ppm	<10	<10	<10	<10	<10
Se	ppm	<30	<30	<30	<30	<30
Sr	ppm	85.6	433.0	176.0	129.0	228.0
Th	ppm	15.5	19.0	18.1	17.4	19.6
Zn	ppm	51	146	53	48	64

13.3.2 Mineralogy

Mineralogical examination of the two dominant rock type composites from R840W Zone, carbonaceous high grade (CHG) and sheared fine-grained granitoid (SFGG), revealed that the uranium is mostly uraninite with lesser coffinite with a secondary phase of uranium minerals inter-grown with silicates. Observations are as follows:

- The CHG composite contained 13.5% mass uraninite (UO_2) compared to 0.1% mass uraninite in the SFGG composite.
- Uranium minerals were 89% liberated in the CHG composite with a P_{80} of 118 µm and 52% liberated in the SFGG composite with a finer P_{80} of 100 µm.
- Most of the uranium minerals are well-formed uraninite crystals in the CHG sample. In contrast, the uranium minerals in the SFGG sample contain a few good-formed uraninite crystals. Still, the uranium seems to be more of a secondary origin due to re-mobilization and alteration of the primary uraninite by a coffinitization process.
- The uranium-bearing grains observed in the CHG contained an average of 88.9% UO_2, whereas the uranium-bearing grains observed in the SFGG contained an average of 76.5% UO_2.
- The chemistry has shown that the uraninite in the SFGG sample contains more impurities (Al, Si, P, Ti) than the CHG sample, and the uranium mineral might be more hydrated.
- The SFGG composite contained elevated levels of pyrite (FeS_2) at 8.3% by mass, and the CHG composite contained trace levels of pyrite at 0.04% by mass.
- The samples contained carbonaceous material, including well-crystallized graphite.
- Some electrum (Au-Ag) grains were identified.

13.3.3 Comminution Tests

SGS completed comminution tests on the two rock-type composites from the R840W zone. The comminution samples were produced by selecting appropriately sized material from subsampling composites of preliminary coarse crushed (-31.5 mm) core intervals. The comminution characterization results for both zones are presented in Table 13-6. The R840W Zone material is softer and less abrasive than the R780E zone that was tested during the PFS.

Table 13-6: Comminution Test Results for FS Composite Samples

Composite	Lithology	CEET Ci	SPI (minutes)	BWIBM (kWh/t)	Ai (g)
Current FS Test Work (2022)					
CHG	Carbonaceous High Grade	n/a	8.8	8.7	0.004
SFGG	Sheared Fine-Grained Granitoid	n/a	12.9	9.6	0.015
Previous PFS Study Test Work (2018)					
CHG	Carbonaceous High Grade	5.9	18.1	11.2	0.004
QFG	Quartz-Feldspar-Gneiss	3.4	28.4	12.3	0.116
GG	Graphitic Gneiss	2.3	17.6	10.3	0.079
G	Unmineralized Gangue	1.3	36.3	12.7	0.279

13.3.4 Leaching

A total of 18 leaching tests were conducted on composite samples, as listed in Table 13-7. Leach conditions, as previously determined in the pre-feasibility tests, were used as follows:

- Target grind size P_{80} of 150 µm
- An initial leach pulp density of 50% solids (w/w), dropping to 47% to 48% solids (w/w) over the leaching period
- A 24-hour leach with sampling at 4, 8, 12, and 24 hours
- A leach temperature of 50°C
- A target free acid of 15 g H_2SO_4/L in the first six hours of leaching, tapering to 10 g H_2SO_4/L by the end of the leach, reporting acid consumptions after 4, 8, 12, and 24 hours
- A target ORP of 450 to 500 mV using sodium chlorate, reporting the sodium chlorate consumptions after 4, 8, 12, and 24 hours. A comparative test using hydrogen peroxide (Test AL-6) was completed, followed by a test using oxygen as an oxidant (Test AL-11). Further leach variability tests were done using hydrogen peroxide as an oxidant.

The tests are named as follows:

- AL1: R780 Overall Composite
- AL2: R840 Overall Composite
- AL3: R780/R840 Overall Composite at 75%/25% ratio.

- AL4: R780/R840 Low Grade at 75%/25% ratio.
- AL5: R780/R840 High Grade at 75%/25% ratio.
- AL2R: Repeat of AL2
- AL5R: Repeat of AL5
- AL6: Repeat of AL3 using H_2O_2
- AL7 to AL15: Nine variability leach tests at different U_3O_8 head grades
- Bulk: Bulk Leach R780/R840 Overall Composite to produce pregnant leach solution for tests

The leach test results are presented in Table 13-7.

Observations

- The head grades for all 18 tests, including the bulk leach, ranged from 0.56% U_3O_8 to 2.40% U_3O_8.
- The extractions for 8-hour leaching ranged from 92.6% to 99.1% and 95.8% to 99.2% for 12-hour leaching.
- The average extraction for an 8-hour leach was 97.9%, and for 12-hour leach was 98.4%.
- Uranium extractions are virtually unaffected by the leach feed grade within the range tested (Figure 13-4).
- The average weight loss of solid material after leaching was 6.3%, ranging from 0.4% to 17.6%.
- The average acid consumption for a 12-hour leach was 66.4 kg H_2SO_4/t of leach feed
- The hydrogen peroxide consumption in the bulk leach test was 7.48 kg H_2O_2/t of leach feed for a 12-hour leach and 4.84 kg H_2O_2/t in test AL-6, averaging at 6.16 kg H_2O_2/t.
- Overall, high uranium extractions were achieved, averaging 98.4% in 12 hours for all the tests regardless of composite type, leach solid density, feed grind size, head grade variation, type of oxidant, oxidation potential, and free acid levels.

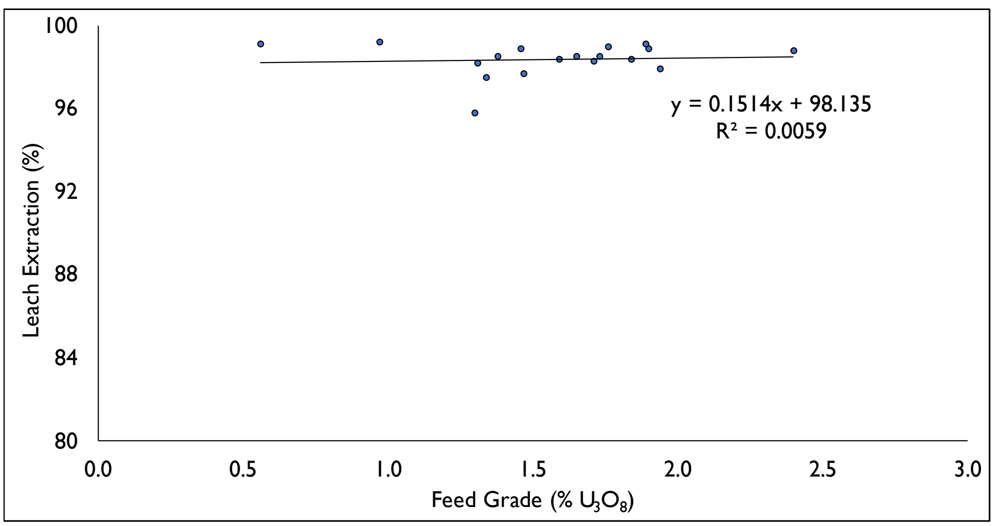


Figure 13-4: Leaching Extraction at 12 hours vs. Feed U_3O_8 Grades for all Tests

Table 13-7: Leach Test Results for FS Composite Samples

Test No.	Comp.	Calc. Head	Residue 12 Hours	% Weight Loss	Final Pregnant Solution (24 Hours)				% U_3O_8 Extraction[1]			
		% U_3O_8	% U_3O_8		g U_3O_8/L	g Fe/L	g Fe^{2+}/L	SG	4 h	8 h	12 h	24 h
AL-1	R780	1.73	0.028	4.4	17.34	5.94	2.78	1.086	97.1	98.2	98.5	98.8
AL-2	R840	1.30	0.057	3.5	12.62	9.29	3.33	1.082	93.8	92.6	95.8	97.7
AL-3	R780/R840	1.59	0.027	5.4	15.57	8.59	4.13	1.096	96.8	98.1	98.4	98.3
AL-4	R780/R840 LG	1.31	0.023	0.4	13.56	7.34	3.56	1.088	96.9	97.7	98.2	n/a
AL-5	R780/R840 HG	1.94	0.041	1.9	19.81	10.20	4.57	1.104	96.4	97.3	97.9	98.4
AL-2R	R840	1.34	0.035	5.8	13.21	10.60	2.80	1.088	96.5	97.2	97.5	97.2
AL-5R	R780/R840 HG	1.90	0.027	0.9	19.34	8.61	n/a	1.098	97.9	98.4	98.9	98.6
AL-6	R780/R840	1.65	0.027	2.3	16.63	9.83	3.26	1.096	96.7	97.2	98.5	98.0
AL-7	R780-0.5%	0.56	0.005	15.7	5.06	7.18	1.13	1.069	98.4	99.1	99.1	99.4
AL-8	R780-1.0%	0.97	0.009	17.6	10.60	10.20	2.07	1.127	98.8	98.6	99.2	99.6
AL-9	R780-1.5%	1.46	0.022	n/a	15.92	6.37	0.92	1.092	98.0	98.7	98.9	98.9
AL-10	R780-2.0%	2.40	0.031	3.5	23.70	7.24	2.09	1.105	98.3	98.6	98.8	99.0
AL-11	R780/R840	1.38	0.022	7.0	14.27	8.68	4.65	1.091	98.0	97.9	98.5	98.4
AL-12	R780/R840	1.89	0.021	2.1	15.21	7.20	1.80	1.081	98.9	99.0	99.1	99.0
AL-13	R780/R840	1.76	0.018	9.6	18.04	8.33	0.77	1.091	98.4	98.9	99.0	99.1
AL-14	R780/R840	1.84	0.028	n/a	17.92	8.42	1.10	1.099	97.5	97.9	98.4	98.2
AL-15	R780/R840	1.71	0.033	7.0	16.75	8.43	1.55	1.097	98.1	97.9	98.3	98.6
BULK	R780/R840	1.47	0.038	13.5	15.80	7.11	3.06	1.079	97.8	98.5	97.7	-

Notes:

[1] Uranium extractions calculated from residue assays and calculated head grade.

[2] The 8-hour residue assay for the bulk leach was 0.025% U_3O_8. The reported assay of the 12-hour residue was 0.038% U_3O_8. The leach residue stayed in the leach tank for an extended period after the leach completion, likely causing some reprecipitation of uranium, resulting in an increase in the final residue grade.

A summary of the 18 leach tests showing head grades and residue solid grades (after 24-hour leaching) and the concentration of the pregnant leach solution is presented in Table 13-8.

Table 13-8: Summary of 18 Leach Test Results after 24 Hours

Sample ID	Head Grade (% U_3O_8)	Residue Grade (% U_3O_8)	Extraction %	Pregnant Solution Grade g/L U_3O_8
AL-1	1.73	0.023	98.8	17.3
AL-2	1.30	0.031	97.7	12.6
AL-3	1.59	0.028	98.3	15.6
AL-4	1.31	0.019	98.5	13.6
AL-5	1.94	0.032	98.4	19.8
AL-2R	1.34	0.040	97.2	13.2
AL-5R	1.90	0.026	98.6	19.3
AL-6	1.65	0.034	98.0	16.6
AL-7	0.56	0.004	99.4	5.1
AL-8	0.97	0.009	99.6	10.6
AL-9	1.46	0.014	98.9	15.9
AL-10	2.40	0.025	99.0	23.7
AL-11	1.38	0.024	98.4	14.3
AL-12	1.89	0.019	99.0	15.2
AL-13	1.76	0.017	99.1	18.0
AL-14	1.84	0.032	98.2	17.9
AL-15	1.71	0.025	98.6	16.8
BULK	1.47	0.038	97.4	15.8
Average	1.57	0.024	98.4	15.6

Note: Numbers may not add due to rounding.

13.3.5 Pregnant Leach Solution Silica Removal Test

A bench scale test was completed on pregnant leach liquor generated from the bulk leach test to evaluate the use of Polyox 301 (a Dupont product) on the removal of very fine suspension of silica and colloidal silica from the solution. Three dosages were tested, 10, 20, and 50 mg/L, on the Millipore (0.45 µm) filtrate. The results of the test are presented in Table 13-9.

Table 13-9: Silica Removal from Pregnant Leach Solution using Polyox

Item	Filtrate				% Si Removal (based on assays)	
	U_3O_8, g/L		Si, g/L			
	1 h	24 h	1 h	24 h	1 h	24 h
Feed	16.75		1.20		--	--
Millipore (0.45 µm) Filtrate	16.27		1.20		--	--
Polyox Dosage, mg/L						
10	17.69	15.21	1.13	1.08	6	10
20	16.63	17.10	0.88	1.06	27	12
50	16.16	14.74	0.76	0.81	37	33

Based on assays, a 50 mg/L dosage would remove 37% of the silica with a one-hour reaction time. Extending the reaction time would lead to some silica re-dissolution. Results indicate that Polyox would be an effective coagulant for clarifying the pregnant leach solution before SX.

13.3.6 Solvent Extraction

A five-day continuous SX mini pilot test was completed on the pregnant leach solution generated from the 12-hour bulk leach on 200 kg of composite R780/R840 material. A 3:1 pregnant leach solution to ore wash ratio was used in the liquid-solid separation generating 600 L of pregnant leach solution. Two 2.5-day SX runs were completed based on a pregnant leach solution flow rate of 5 L/h. The organic make-up was 5% Alamine 336, 5% Isodecanol and 90% kerosene (Exxsol D80). The circuit was set up informed by successful parameters established at an SX operation using strong acid strip chemistry. The mini-pilot mixer settlers (MS) were set up as indicated in Figure 13-5, with details of the set-up presented in Table 13-10.

Figure 13-5: Continuous SX Mini Pilot Test Set-Up

Table 13-10: Mini-Pilot SX Set-up Details

Description	No. of Stages	Advance Flowrate Organic/Aqueous	Flow to mixer Organic/Aqueous	Reagent
Extraction	4	1/1 and 0.8/1*	1.5/1	Amine extractant
Scrub	1	1/0.1	3/1	Water
Strip	5	1/0.1 or 1/0.05*	3/1	Strong acid strip 425 g/L H_2SO_4
Acid Recovery	1	1/0.1	3/1	Water
Regeneration	1	1/0.1	3/1	Aqueous solution at Na_2CO_3/L

Note: * For SX run 2

The mini-pilot SX achieved the desired results by rejecting deleterious elements and concentrating uranium by a factor of 15 into a strong acid strip solution. Table 13-11 presents a snapshot of selected elements grades in various SX streams.

Table 13-11: Concentration of Selected Elements in the Pilot SX Streams

Stream	Aqueous (mg/L): Organics (g/t)								
	U	Mo	V	As	Cu	Y	Pb	Ca	Mg
Pregnant Leach Solution	5,110	54.6	28	4	61	15.2	10	761	790
Extraction 1 Aqueous	<0.02	3.4	26	3	51	13.3	4	713	683
Scrub Aqueous	143	1.5	<3	<3	<40	0.078	<2	33.5	6
Strip 1 Aqueous	52,700	136	<3	<3	<40	0.026	68	15	<4
Extraction 4 Organic	4,560	172	<7	<80	<10	<1	<200	<90	10
Scrub Organic	4,560	164	<7	<80	<10	<1	<200	<90	9
Strip 5 Organic	5	129	<7	<80	<10	<1	<200	<90	8
Regenerated Organic	3	124	<7	<80	<10	<1	<200	<90	9

Key observations from the SX Mini Pilot testing:

- With a measured specific gravity of 1.03, the pregnant leach solution contained low levels of most potential impurity elements assayed, including arsenic and lead. Other than uranium and molybdenum, other metals, namely all of vanadium, trace level of arsenic, most of the copper and yttrium, and approximately 50% of the lead, stayed in the raffinate.
- The four extraction stages achieved a 99.9% uranium extraction in both SX Run No.1 and SX Run No. 2. The corresponding molybdenum extraction was similar for both runs at 90% to 95%.
- Uranium in loaded organic feeding the stripping circuit ranged from 5,870 mg/L U to a high of 9,180 mg/L U.
- Uranium concentration on loaded strip solution achieved a high of approximately 80,000 mg/L U at a maximum stripping efficiency of 99.9%. Uranium in the pregnant leach solution was concentrated by a factor of 15 into the loaded strip solution.
- Of the deleterious elements in the pregnant leach solution entering the SX, only approximately 30% of the molybdenum was transferred to the loaded strip solution at a low grade of approximately 200 mg/L Mo.
- The organic regeneration aqueous was tested at Na_2CO_3 strengths from 50 g/L to 150 g/L and organic-to-aqueous ratios of 5:1 and 10:1. It was found that molybdenum could be stripped from barren recycled organic by regenerating at 150 g/L Na_2CO_3 aqueous at an organic-to-aqueous ratio of 5:1.
- Acid recovery aqueous achieved a maximum acid strength of 68 g/L H_2SO_4 for recycle after recovery of acid from barren organic. Acid recovery aqueous assayed an average of 50 g/L H_2SO_4 in SX Run No. 1 and 35 g/L H_2SO_4 in SX Run No. 2.

13.3.7 Acid Recovery Nanofiltration Tests

Two nanofiltration tests were completed on the pregnant strip liquor from the two SX runs, Test NF1 on the SX Run No.2 liquor and Test NF2 on the SX Run No.1 liquor. The membrane used was a KH1812 membrane supplied by MDSAmericas with an equivalent area of 0.511 m^2. The tests were run at 800 psi, and the concentrate was recycled to the feed tank. The permeate was collected, and the concentrate and permeate were submitted for uranium and free acid analysis. Test results are presented in Table 13-12.

Table 13-12: Acid Recovery Tests using Nanofiltration

Stream	SG	Volume (mL)	Flowrate		Conductivity (mS)	Temp (°C)	H_2SO_4 (g/L)	U_3O_8 (g/L)	% Distribution	
			mL/min	L/m²/day					U_3O_8	H_2SO_4
Test NF1 (SX No. 2 Pregnant Strip Liquor as Feed)										
Feed	1.291	5,011	--	--	--	--	374	76.8	--	--
Permeate	1.191	2,432	67-116	189-327	>200	41	304	17.0	12.7	48.6
Concentrate	1.297	2,489	--	--	173	64	314	114.0	87.3	51.4
Test NF2 (SX No. 1 Pregnant Strip Liquor as Feed)										
Feed	1.275	5,017	--	--	--	--	396	52.1	--	--
Permeate	1.197	2,590	70-136	197-383	>200	n/a	323	11.9	15.1	52.6
Concentrate	1.268	2,277	--	--	176	29-64	332	76.2	84.9	47.4

Results indicate that nanofiltration can provide the potential for recovering 50% of the sulphuric acid in the pregnant strip liquor for recycling to the process, but this would be accompanied by a 15% recycling of uranium. The volume split between the permeate and concentrate was roughly 50/50. The uranium grade of the pregnant strip liquor feeding the yellowcake precipitation circuit would increase by a factor of 1.5.

13.3.8 Gypsum Precipitation

Bench-scale gypsum precipitation tests were completed on the loaded strong acid solution to determine the optimum precipitation pH from a test range of pH 3 to 4.

The key results from bench-scale tests are:

- pH 3.7 was optimum for minimizing uranium associated with the gypsum precipitate.
- 1.2% of the uranium in the loaded strip solution was reported to the washed gypsum cake, which had a uranium grade of 0.17% U_3O_8.
- Pre-precipitation H_2SO_4 was approximately 355 g/L, and post-precipitation H_2SO_4 was approximately 13 g/L.
- Lime Addition was 0.81 kg $Ca(OH)_2$/kg H_2SO_4.
- The precipitate was 2.0 kg dry gypsum/kg $Ca(OH)_2$ or 1.64 kg dry gypsum/kg H_2SO_4.

Following the bench scale tests, two bulk gypsum precipitation tests were undertaken at pH 3.6 and 3.7. The key results were as follows:

- The average lime consumption in the bulk tests was 0.74 kg $Ca(OH)_2$/kg H_2SO_4.
- The average bulk gypsum precipitate weight (dry basis) was 2.16 kg/kg $Ca(OH)_2$ added and 1.58 kg/kg H_2SO_4 in the pregnant strip liquor feed.
- The gypsum cake uranium grade was variable and higher than the bench scale tests at 0.56% U_3O_8 and 2.32%, respectively, for the two tests.
- The gypsum cake was re-slurried to a density of 15% to 16% solids (w/w) and leached for four hours with sulphuric acid at a free acid level of 10 g H_2SO_4/L to recover the co-precipitated uranium. Recovery of the co-

precipitated uranium was 80%, and the grade of the final washed gypsum cake was 0.05% U_3O_8. Further second-stage washing of the gypsum cake could decrease the final washed gypsum cake grade to roughly 0.025% U_3O_8.

13.3.9 Yellowcake Precipitation

Yellow cake precipitation tests were undertaken at the bench and bulk scale to confirm the process and define the parameters required for uranyl peroxide precipitation using hydrogen peroxide as the reactant and magnesia for pH control.

Two bench-scale yellowcake precipitation tests were completed on the uranium-bearing filtrate produced after gypsum precipitation. The yellowcake was precipitated with the addition of 30% H_2O_2 and 5% MgO slurry to control pH at 3.0 to 3.5. The total reaction time for the tests was eight hours. The uranyl peroxide precipitate was filtered, washed, dried, and submitted for assay. Details of the tests are presented in Table 13-13.

Table 13-13: Bench Scale Yellowcake Precipitation Test Results

Item	Unit	Test 1	Test 2
Calculate Feed U_3O_8	mg/L	27,110	20,511
Barren solution U_3O_8	mg/L	75	121
Uranium recovery to precipitate	%	99.7	99.3
Yellowcake U_3O_8 grade	%	81.6	80.0
Initial pH	#	4.2	3.3
Final pH	#	3.4	3.4
H_2O_2 (100%) addition	kg/kg U_3O_8	0.23	0.26
MgO (100%) addition	kg/kg U_3O_8	0.125	0.244

Regarding deleterious elements, the yellowcake product, assaying 0.095% Mo and 0.159% Mo, is within the molybdenum refinery limits of 0.3% Mo and according to the uranium refinery specifications.

The bulk pregnant strip liquor, including the acid wash from the bulk gypsum precipitation tests, was submitted to bulk yellowcake precipitation tests using the test conditions established in the bench scale yellowcake precipitation tests. The test produced a uranyl peroxide precipitate at an average 99.6% recovery. The uranium analysis of the dry precipitate was approximately 80% U_3O_8, and all elements were within the refinery specifications. The test results are summarized in Table 13-14.

Table 13-14: Bulk Yellowcake Precipitation Test Results

Item	Unit	Test 1	Test 2
Feed U_3O_8	mg/L	12,847	26,245
Barren solution U_3O_8	mg/L	97	2
Uranium recovery to precipitate	%	99.2	100.0

table continues…

Item	Unit	Test 1	Test 2
Yellowcake U_3O_8 grade	%	79.3	80.1
Initial pH	#	3.5	3.6
Final pH	#	3.3	2.7
H_2O_2 (100%) addition	kg/kg U_3O_8	0.17	0.23
MgO (100%) addition	kg/kg U_3O_8	0.15	0.28

13.3.10 Yellowcake Calcination Tests

Yellowcake from the bulk precipitation test was submitted for calcination tests at temperatures of 150°C, 450°C, and 850°C and was held at temperature for one hour. The results of the tests are presented in Table 13-15.

Table 13-15: Bulk Yellowcake Calcination Test Results

Item	150°C		450°C		850°C	
	Feed	Calcined	Feed	Calcined	Feed	Calcined
Assay (% U_3O_8)	79.4	80.8	79.4	94.9	79.4	99.8
Weight loss (wet cake)	1.50%		14.80%		21.00%	

The results indicate a marked improvement in uranium grade and packing density with increasing temperature. The product quality was well within the refinery specifications.

13.3.11 Effluent Treatment and Tailings Preparation Tests

Bench-scale effluent treatment tests followed by bulk effluent tests were completed to evaluate effluent treatment processes and parameters required to achieve the environmental release specifications prescribed by the MDMER guidelines.

The following liquid effluents were combined in a ratio estimated from the PFS process mass balance to prepare the feed for the effluent treatment tests, namely:

- Raffinate from the SX mini pilot plant
- Spent Regeneration Solution from the SX mini pilot plant
- Barren Strip Liquor
- Simulated Tailings Water (simulated tailings water was prepared by contacting water with the bulk leach residue).

The effluent treatment tests simulated three consecutive stages of precipitation at pH 4.0, 7.0, and 10.0, respectively, with doses of lime (CaO), iron as ferric sulphate ($Fe_2(SO_4)_3$), and barium chloride ($BaCl_2$) at each stage. Flocculant screening tests were conducted on precipitate thickening at the three pH regimes.

All effluent treatment tests yielded similar treated effluent quality meeting MDMER guidelines. A dosage of 75 ppm Fe^{3+} in the first stage of each test appeared adequate. Based on these results, a three-stage treatment was used in the bulk effluent treatment and tailings preparation.

Two tailings scenarios were evaluated, one where 50% of the solids would be used for underground mine backfill and the other where 100% of the tailings would report to the surface TMF. The tailings neutralization tests were conducted at three consecutive stages of neutralization at pH 4.5, 7.0, and 10.0, respectively, with doses of lime, iron as ferric sulphate ($Fe_2(SO_4)_3$), and barium chloride ($BaCl_2$) in Stage 1, and lime alone in Stages 2 and 3.

The final treated effluent in both tests was low in deleterious elements, meeting MDMER guidelines. The results of the bulk effluent treatment were in line with the bench-scale effluent treatment tests. The final treated effluent was produced low in deleterious elements, meeting MDMER effluent regulations. A target ferric iron dosage of 75/50/25 ppm in the first, second and third stages of each test appeared adequate, as did the barium chloride target addition of 50/50/25 ppm.

Liquid analyses for the three stages and the final pH 7 filtrate of the bulk effluent treatment tests for the 100% surface tails test and the 50% surface tails test produced results within the MDMER effluent release regulations. The liquid assays on 0.45 µm Millipore filtered liquid include the filtrate from treatment effluent stages 1, 2, and 3 and the final filtrate after the pH adjustment to 7 using sulphuric acid.

The analysis of the solution of the final neutralized tailings shows elevated levels of molybdenum (15.5 mg/L Mo and 21.0 mg/L Mo for 100% and 50% surface tails tests, respectively). The molybdenum would be removed in the effluent treatment circuit because the supernatant from the tailings storage pond will be recycled back to the processing plant.

Two additional bench scale effluent treatment tests were carried out to confirm test conditions and results with final tailings filtrate from 100% Surface Tails and 50% Surface Tails tests instead of simulated tailings water and increasing the ferric iron dosage in Stage 1 of the treatment to 150 ppm. The effluent was treated in three stages at incremental pH's (pH 4, 7 and 10) with ferric iron increased (150, 75 and 50 ppm, respectively, relative to the stage feed volume) to account for the higher molybdenum content in the tailings filtrate compared to the simulated tailings water.

Tests indicated that the stage 3 (pH 10) filtrate and the final pH 7 adjusted filtrate are well within the MDMER guidelines. It was suggested that the ferric iron dosage for effluent treatment should be 275 mg Fe^{3+}/L of the solution, split into roughly 50%, 35% and 15% for Stages 1, 2 and 3, respectively. Based on these tests, the target selenium level in the end-of-pipe treated effluent would be <0.015 mg/L Se.

13.3.12 Gold and Silver Recovery

CIL tests were carried out on the washed acid leach residue (CCD Tails) from the bulk leach with varying leach retention times. The total gold extraction increased to 91.7% with a 48-hour leach, with 86.6% of the gold recovered on carbon. The total silver extraction achieved for a 48-hour leach was 76.5%, with 74.3% recovered on carbon for a 2.12 g/t Ag head grade.

The gold recovery on carbon, 86.6% for a 48-hour leach based on a calculated head grade of 0.30 g Au/t, was higher than the previously achieved 79.3% recovery on Composite O/P in the PFS test work, with a similar calculated head grade of 0.34 g Au/t.

13.3.13 Liquid/Solid Separation Tests

Leach Feed

A dynamic settling test on the ground leach feed using Magnafloc 333 showed that the highest underflow density, 53.6% solids (w/w), and the lowest overflow total suspended solids (TSS) level, 16 mg/L, was achieved with 6% solids (w/w) diluted feed density operating at a thickener unit area of 0.25 m^2/t/day (no safety factor applied).

Leach Residue

Flocculant tests on the overall composite leach discharge showed that Magnafloc 333 and Magnafloc 351 were acceptable flocculants yielding a clear supernatant.

Dynamic testing of the leach discharge revealed that based on a target thickener unit area of 0.33 m^2/t/day (no safety factor applied), the underflow density would be 49.5% solids (w/w) with an overflow TSS of 52 ppm based on a diluted feed density of 4% solids (w/w).

Washed Leach Residue

In dynamic settling testing of the washed leach residue, the highest underflow density of 44% solids (w/w) was achieved with 3% solids (w/w) diluted feed density operating at a thickener unit area of 0.22 m^2/t/day (no safety factor applied). Staying with a thickener unit area of 0.33 m^2/t/day (no safety factor applied) would yield a 43% solids (w/w) underflow density and an overflow TSS level of 46 ppm using a 3% solids (w/w) feed density.

Gypsum Precipitate

Liquid-solid separation tests were performed on the bulk gypsum precipitate prepared from the SX Run No. 1 pregnant strip liquor. Thickening tests were attempted, but the gypsum slurry provided was already at a high density of 15 to 20% solids (w/w); hence the slurry was only submitted for filtration and centrifuge tests.

High filtration rates were achieved in both vacuum and pressure filtration without filter aid addition. Vacuum filtration yielded a 43% solids (w/w) filter cake, and pressure filtration yielded a 66% solids (w/w) filter cake. Centrifuge tests at a 3200 g-force yielded a centrifuged slurry with 40% solids (w/w) (more than five minutes spin time).

Yellowcake Precipitate

The yellowcake from the bulk yellowcake precipitation test was submitted to liquid-solid separation tests, including a static settling test, a pressure filter test, a vacuum filtration test, and a centrifuge test. Achieved densities were 18% solids (w/w) for thickening, 48% solids (w/w) for pressure filtration, 43% solids (w/w) for vacuum filtration, and 48% solids (w/w) for the centrifuge (25 minutes spin time).

It is noted that the yellowcake produced under laboratory batch test conditions was very fine, and consequently, liquid/solid separation results were generally poor. These results are likely not indicative of yellowcake produced under plant operating conditions.

Tailings

Based on dynamic settling tests, the 100% Surface Tails yielded the highest underflow density of 41% solids (w/w) with 3% solids (w/w) diluted feed density operating at a thickener unit area of 0.21 m^2/t/day (no safety factor applied) with a 141 mg/L TSS overflow. For the 50% Surface Tails, the highest underflow density of 32.5% solids (w/w) was achieved with 4% solids (w/w) diluted feed density operating at a thickener unit area of 0.38 m^2/t/day (no safety factor applied) with a 113 mg/L TSS overflow.

The 100% Surface Tails contained 24.9% gypsum, while the 50% Surface Tails contained 36.7% gypsum, impacting the settling characteristics of the 50% Surface Tails.

The pressure filtration tests with the 100% Surface Tails provided filter cake densities of 69.6 to 73.0% solids (w/w) and unit filtration rates of 99 to 131 kg/h/m^2, whereas the 50% surface tails provided filter cake densities of 66.6 to 71.0% solids (w/w) and unit filtration rates of 83 to 118 kg/h/m^2.

13.4 Overall Uranium Recovery

The leaching test work indicated that the uranium extraction averaged 98.4% in 12 hours for all the tests regardless of composite type, leach density, feed grind size, head grade variation, type of oxidant, oxidation potential and free acid levels.

The leaching extraction at varying feed grades can be estimated using the equation below.

$$\text{Leach Extraction (\%)} = 0.1514 \times \text{Feed } U_3O_8 \text{ grade (\%)} + 98.1350$$

The average uranium recovery and losses during the subsequent circuits are listed in Table 13-16.

Table 13-16: Uranium Recovery and Loss in Different Circuits

Stage	Unit	Value
CCD Circuit Recovery	%	99.5
SX Circuit Recovery	%	99.6
Uranium Loss during Gypsum Precipitation	%	0.1
Yellowcake Precipitation Circuit	%	99.9
Unaccountable Losses	%	0.3

Accounting for all subsequent circuits, the overall uranium recovery can be estimated using the equation below.

$$\text{Uranium Recovery (\%)} = 0.1493 \times \text{Feed } U_3O_8 \text{ grade (\%)} + 96.7682$$

14.0 MINERAL RESOURCE ESTIMATE

Mineral Resources have been classified in accordance with the definitions for Mineral Resources in the CIM Definition Standards for Mineral Resources and Mineral Reserves dated May 10, 2014 (CIM 2014).

Table 1-2 summarizes Mineral Resources based on a US$50/lb uranium price at a COG of 0.25% U_3O_8 and a potential underground scenario. Indicated Mineral Resources total 2.69 Mt at an average grade of 1.94% U_3O_8 for a total of 114.9 Mlb U_3O_8. Inferred Mineral Resources total 0.64 Mt at an average grade of 1.10% U_3O_8 for a total of 15.4 Mlb U_3O_8. Estimated grades are based on chemical assays only. Gold grades were also estimated and average 0.61 g/t for the Indicated Mineral Resources and 0.44 g/t for the Inferred Mineral Resources. Mineral Resources are inclusive of Mineral Reserves.

The cut-off date of the Mineral Resource database is December 22, 2021, which represents the date in which all assays were received from FCU's Summer 2021 drill program. The effective date of the Mineral Resource estimate is May 17, 2022.

The SLR QP is not aware of any environmental, permitting, legal, title, taxation, socioeconomic, marketing, political, or other relevant factors that could materially affect the current Mineral Resource estimate.

Table 14-1: Mineral Resource Statement – May 17, 2022

Category	Tonnage	Metal Grade		Contained Metal	
	(000 t)	(% U_3O_8)	(g/t Au)	(Mlb U_3O_8)	(000 oz Au)
Indicated	2,688	1.94	0.61	114.9	52.7
Inferred	635	1.10	0.44	15.4	9.0

Notes:

1. CIM (2014) definitions were followed for Mineral Resources.
2. Mineral Resources are reported at a COG of 0.25% U_3O_8, based on a long term price of US$50/lb U_3O_8, an exchange rate of C$1.00/US$0.75, and cost estimates derived during the PFS with a metallurgical recovery of 95%.
3. Minimum mining width of 1 m was applied to the resource domain wireframe.
4. Mineral Resources are inclusive of Mineral Reserves.
5. Numbers may not add due to rounding.

14.1 Resource Database

FCU maintains a Property-wide drill hole database in Microsoft Access. Of the total 844 drill holes drilled on the PLS Property, 696 drill holes totaling 213,969 m of drilling were used in the Mineral Resource estimate (Table 14-2). The wireframe models representing the mineralized zones are intersected in 434 of 696 drill holes (Table 14-3).

Table 14-2: Drill Hole Resource Estimation Database Record Count

Table Name / Field	Number of Records
Collar / DHID	696
Collar/ td (m)	213,969
Survey / Depth	26,540
Assay / U_3O_8%	114,561
Lithology / Litho	12,676
Density / dens_wax	17,509
Composites	61,235

Table 14-3: Number of Drill Hole Intersecting Wireframes

Deposit Wireframes	Number of Drill Holes	Total Depth Drilled (m)
R780E	281	98,735
R00E	23	4,779
R1620E	23	5,989
R840W	91	27,606
R1515W	18	6,147
Grand Total	**436**	**143,256**

Section 12, Data Verification, describes the verification steps made by SLR. In summary, no discrepancies were identified, and the SLR QP is of the opinion that the Triple R drill hole database is valid and suitable to estimate Mineral Resources for the Triple R deposit.

14.2 Geological Interpretation and 3D Solids

Basement hosted mineralization at the PLS Property occurs in a variety of styles, the most common of which appears to be fine grained disseminated and fracture filling uranium minerals strongly associated with hydrocarbon/carbonaceous matter within the graphitic pelitic gneiss. Uranium minerals, where visible, appear to be concordant with the regional foliation and dominant structural trends identified through oriented core and fence drilling (i.e., steeply dipping to the southeast).

The initial Resource Estimate prepared by RPA (RPA 2015a) was based only on mineralization contained within and around the R00E and R780E areas. Subsequent drilling programs conducted identified three additional zones of mineralization. Mineralization is now shown to occur at five locations on the PLS Property: 1) R780E, 2) R00E, 3) R1515W, 4) R840W, and 5) R1620E. The R780E zone hosts higher grade, thicker, and more continuous mineralization compared to other areas as defined by current drilling.

Geological interpretations supporting the estimate were generated by SLR and reviewed by Triple R personnel. Wireframe models of mineralized zones were used to constrain the block model grade interpolation process. SLR interpreted and constructed low grade wireframe models using a nominal COG of 0.05% U_3O_8 and a minimum core length of 1 m. SLR considers the selection of 0.05% U_3O_8 to be appropriate for construction of mineralized wireframe outlines, as this value reflects the lowest COG that is expected to be applied for reporting of the Mineral Resources in an underground operating scenario and is consistent with other known deposits in the Athabasca Basin. Sample intervals with assay results less than the nominated COG were included within the mineralized wireframes if the core length was less than 2 m or allowed for modelling of grade continuity. Wireframes of the HG domain were created using a grade intercept limit equal to or greater than 1 m with a minimum grade of 5% U_3O_8, although lower grades were incorporated in places to maintain continuity and to meet a minimum thickness of 1 m.

SLR built the wireframe models using three-dimensional (3D) polylines on east looking vertical sections spaced 15 m apart. Infill polylines were added to accommodate for irregular geometries. Polylines were "snapped" to assay intervals along the drill hole traces such that the sectional interpretations "wobbled" in 3D space. Polylines were joined together in 3D using tie lines and the continuity was checked using a longitudinal section and level plans. Extension distance for the mineralized wireframes was half-way to the next hole, or approximately 25 m vertically and horizontally past the last drill intercept.

The Triple R deposit as defined in the Mineral Resource estimate is comprised of several nearly vertical, stacked lenses across five mineralized zones that are generally oriented with an azimuth 66.2° northeast. The zones range from 60 m to 100 m wide with an overall strike length of 3.2 km starting at approximately 50 m from surface and extending to 300 m at depth. The deposit remains open in most directions.

The R00E zone is located at the western end and the much larger R780E zone. The R00E and R780E zones have an overall strike length of approximately 1.2 km, with the R00E measuring approximately 125 m in strike length and the R780E zones measuring approximately 900 m in strike length. A 225 m gap separates the R00E zone to the west and the R780E zone to the east.

The R780E zone is located beneath Patterson Lake, which is approximately 6 m deep in the area of the deposit. The R00E and R780E zones are covered by approximately 50 m of overburden. The deposit extends from immediately beneath the overburden to a maximum depth of 330 m below the topographic surface.

A total of 80 wireframe models (domains) of the mineralization were constructed by SLR and used in the Resource Estimate (Table 14-4, Figure 14-1, and Figure 14-2). Of the 82 wireframes, 16 are high grade wireframes located within the low grade R780E_MZ wireframe, and two high grade wireframes are contained within the low grade R840W_001 wireframe. Wireframe names were assigned to zones as identified by FCU disclosures.

Table 14-4: Summary of Wireframe Models

Block Codes	2021 Wireframes	Zone	Target	Volume (m^3)
101	101_2021_cut.00t	R780E_MZ	R780E	1,180,598
102	102_2018.00t	R780E_MZ	R780E	17,340
201	201_2015.00t	R780E_OTHER	R780E	32,716
202	202_2015.00t	R780E_OTHER	R780E	4,435
203	203_2015.00t	R780E_OTHER	R780E	52,118
204	204_2021.00t	R780E_OTHER	R780E	46,120
205	205_2018.00t	R780E_OTHER	R780E	39,887
206	206_2018.00t	R780E_OTHER	R780E	7,680

table continues…

Block Codes	2021 Wireframes	Zone	Target	Volume (m^3)
301	301_2021.00t	R780E_OTHER	R780E	18,038
302	302_2021.00t	R780E_OTHER	R780E	21,874
303	303_2021.00t	R780E_OTHER	R780E	30,923
304	304_2015.00t	R780E_OTHER	R780E	6,260
305	305_2021.00t	R780E_OTHER	R780E	65,707
306	306_2018.00t	R780E_OTHER	R780E	20,459
307	307_2015.00t	R780E_OTHER	R780E	2,426
308	308_2018.00t	R780E_OTHER	R780E	7,471
401	401_2021.00t	R780E_OTHER	R780E	102,382
501	501_2018.00t	R780E_OTHER	R780E	65,788
601	601_2015.00t	R00E	R00E	57,228
602	602_2015.00t	R00E	R00E	4,003
10111	10111_2018.00t	R780E_HG_PARTS	R780E	2,187
10112	10112_2018.00t	R780E_HG_PARTS	R780E	363
10113	10113_2018.00t	R780E_HG_PARTS	R780E	5,527
10114	10114_2018.00t	R780E_HG_PARTS	R780E	167
10115	10115_2018.00t	R780E_HG_PARTS	R780E	307
10211	10211_2018.00t	R780E_HG_PARTS	R780E	2,701
10212	10212_2018.00t	R780E_HG_PARTS	R780E	12,397
10213	10213_2018.00t	R780E_HG_PARTS	R780E	3,006
10214	10214_2018.00t	R780E_HG_PARTS	R780E	1,987
10215	10215_2018.00t	R780E_HG_PARTS	R780E	1,407
10311	10311_2021.00t	R780E_HG_PARTS	R780E	1,336
10312	10312_2021.00t	R780E_HG_PARTS	R780E	528
10411	10411_2021.00t	R780E_HG_PARTS	R780E	12,403
10611	10611_2018.00t	R780E_HG_PARTS	R780E	2,209
10711	10711_2018.00t	R780E_HG_PARTS	R780E	18,961
10712	10712_2018.00t	R780E_HG_PARTS	R780E	1,893
84001	84001_2022_CUT.00t	R840W	R840W	183,323
84002	84002_2018.00t	R840W	R840W	1,433
84003	84003_2022.00t	R840W	R840W	6,307
84005	84005_2018.00t	R840W	R840W	2,108
84006	84006_2022.00t	R840W	R840W	10,448

table continues…

Block Codes	2021 Wireframes	Zone	Target	Volume (m^3)
84007	84007_2018.00t	R840W	R840W	3,505
84008	84008_2022.00t	R840W	R840W	14,183
84009	84009_2022.00t	R840W	R840W	7,748
84010	84010_2022.00t	R840W	R840W	746
84011	84011_2022.00t	R840W	R840W	1,626
84012	84012_2022.00t	R840W	R840W	3,320
84013	84013_2018.00t	R840W	R840W	1,065
84014	84014_2018.00t	R840W	R840W	1,687
84015	84015_2018.00t	R840W	R840W	265
84016	84016_2018.00t	R840W	R840W	792
84017	84017_2018.00t	R840W	R840W	227
84018	84018_2018.00t	R840W	R840W	553
84019	84019_2018.00t	R840W	R840W	745
84020	84020_2018.00t	R840W	R840W	429
84021	84021_2018.00t	R840W	R840W	1,223
84022	84022_2018.00t	R840W	R840W	2,161
84023	84023_2018.00t	R840W	R840W	48
84024	84024_2018.00t	R840W	R840W	672
84025	84025_2018.00t	R840W	R840W	406
84026	84026_2018.00t	R840W	R840W	376
84027	84027_2018.00t	R840W	R840W	222
84028	84028_2018.00t	R840W	R840W	90
840101	840101_2022.00t	R840W_HG_PART	R840W	2,968
840102	840102_2022.00t	R840W_HG_PART	R840W	1,279
15151	15151_2018.00t	R1515W	R1515W	49,595
15152	15152_2018.00t	R1515W	R1515W	34,502
15153	15153_2018.00t	R1515W	R1515W	26,516
15154	15154_2018.00t	R1515W	R1515W	22,163
15155	15155_2018.00t	R1515W	R1515W	13,848
15156	15156_2018.00t	R1515W	R1515W	10,477
15157	15157_2018.00t	R1515W	R1515W	3,082
15158	15158_2018.00t	R1515W	R1515W	4,712
15159	15159_2018.00t	R1515W	R1515W	1,170

table continues…

Block Codes	2021 Wireframes	Zone	Target	Volume (m^3)
16201	16201_2018.00t	R1620E	R1620E	21,364
16202	16202_2018.00t	R1620E	R1620E	32,334
16203	16203_2018.00t	R1620E	R1620E	7,893
16204	16204_2018.00t	R1620E	R1620E	3,271
16205	16205_2018.00t	R1620E	R1620E	1,308

Infill and delineation drilling completed since September 25, 2018, resulted in the following updates to the wireframe domains:

- A total of 20 of 80 mineralized domains were intersected with the 2019-2021 drilling.
  - Table 14-5 presents the total volume change of the wireframes intersected, while Table 14-6 presents the total block volume in each wireframe at a 0.25% U_3O_8 COG.
  - Figure 14-3 and Figure 14-4 present the volume of the mineralized domains as of September 19, 2019, for R780E and R840W, respectively.
  - Figure 14-5 and Figure 14-6 present the volume of the mineralized domains as of May 17, 2022, for R780E and R840W, respectively.

The volume of the two main zones (R780E-101 and R840W-84001), controlling approximately 73% of the R780E and R480W Mineral Resources, remain relatively unchanged with less than 1% change in both total volume and block volume.

- LG domains 301, 302, 303, 84003, 84006, 84008, 84009, and 84011 and HG domains 10311 and 10312 exhibit the largest change in volume between 2019 and 2022, ranging from a decrease of 40% in total volume (10312) to an increase in total volume exceeding 200% (84006) as presented in Table 14-5.
  - The increases and decreases in volume of these domains are less than 7% of the overall total volume.
  - The overall total volume change is nearly double, at 15%, of the total block volume using a 0.25% U_3O_8 COG.
- The overall volume change in the aforementioned LG and HG domains has a minimal impact on the overall Mineral Resource estimate as the largest volume changes are coupled with decreasing grade values (Table 14-6 and Table 14-7).
  - Infill drilling increased the volume of domain 301 by 27% but resulted in a decrease in grade of 43% (2.02% U_3O_8 to 1.15% U_3O_8).
  - Infill drilling in domain 302 resulted in a volume increase of 12.6% but resulted in a decrease in grade of 17% (1.55% U_3O_8 to 1.27% U_3O_8).
  - Infill drilling decreased the volume in 303 by 12.6% along with a slight increase in grade of 2.4% (1.14% U_3O_8 to 1.16% U_3O_8).
  - Infill drilling increased the volume in domains 84003, 84006, 84009, and 84011 between a low of 15% to a high of 154%, however, overall grade values decreased from a low of 10% to greater than 50%.

  - Infill drilling in domain 84008 increased volumes by 154% along with increasing grade by 74%.

- Lower overall grade values along with a change in volume resulted in an overall decrease of 3.9 Mlb U_3O_8, or approximately 6.3% (Table 14-8).

- The May 17, 2022, Mineral Resource estimate has excluded mineralization previously associated with the Halo zone, as this zone does not meet CIM (2014) definition criteria for reasonable prospects for eventual economic extraction (RPEEE) based on an underground mining only scenario.

- Parameters for classification of Mineral Resources remain unchanged from previously reported Mineral Resource estimates.

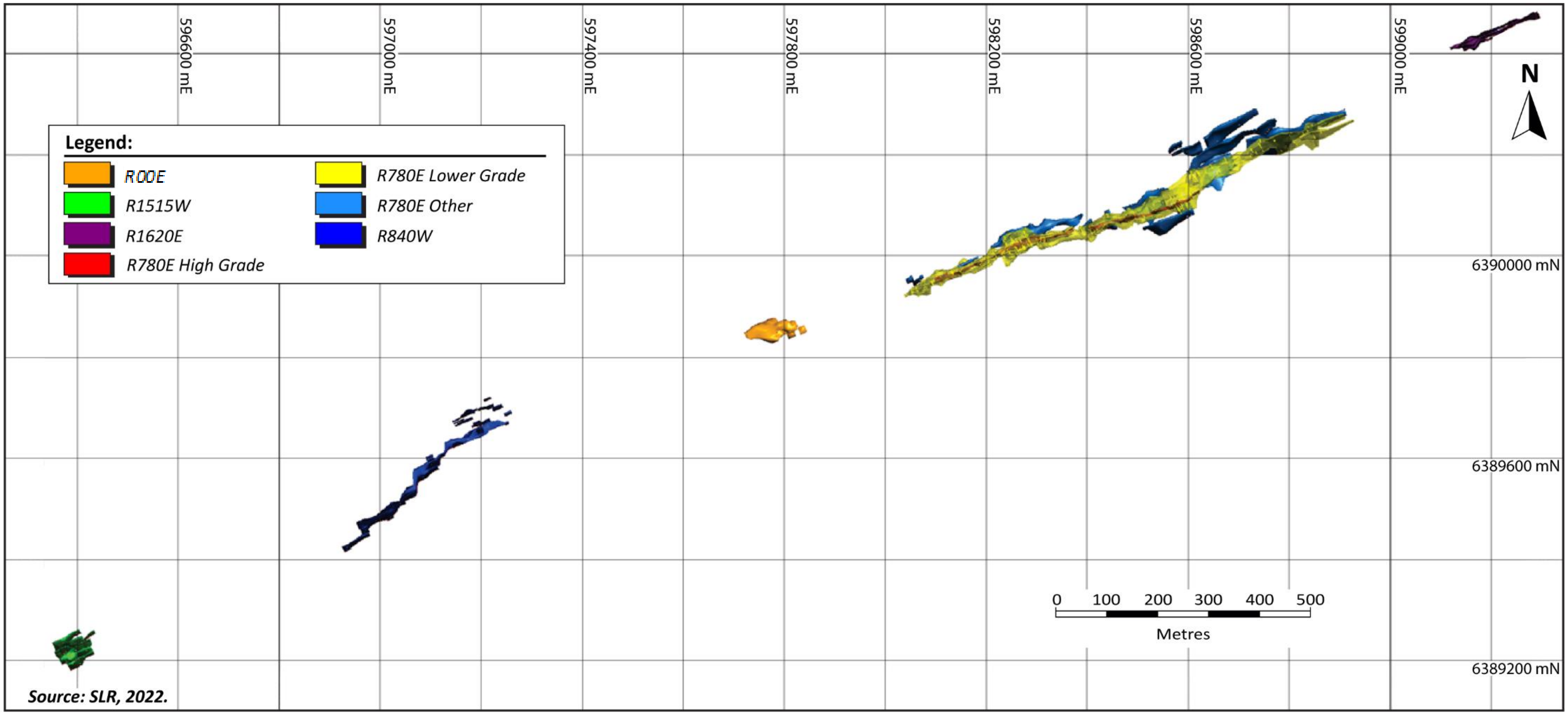


Figure 14-1: Wireframe Solids Plan View

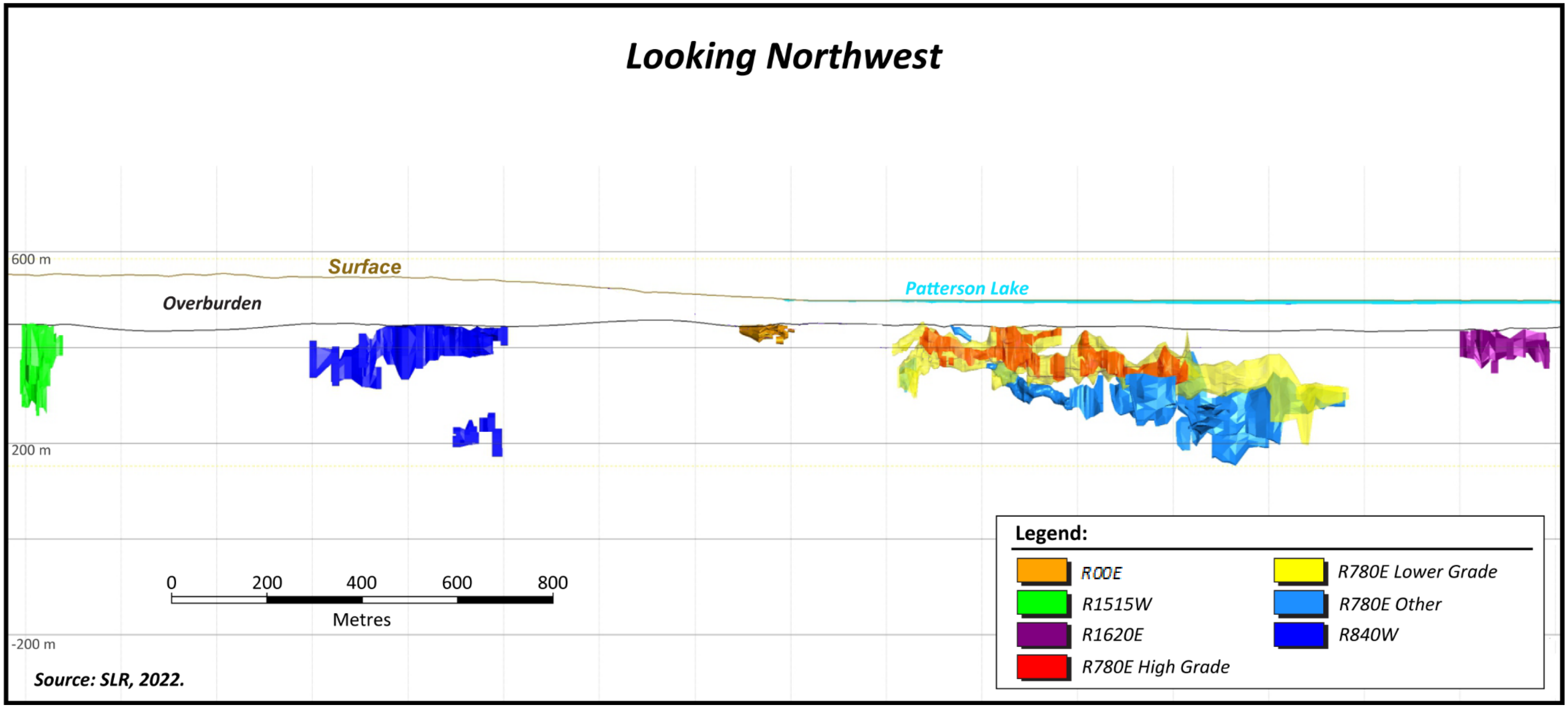
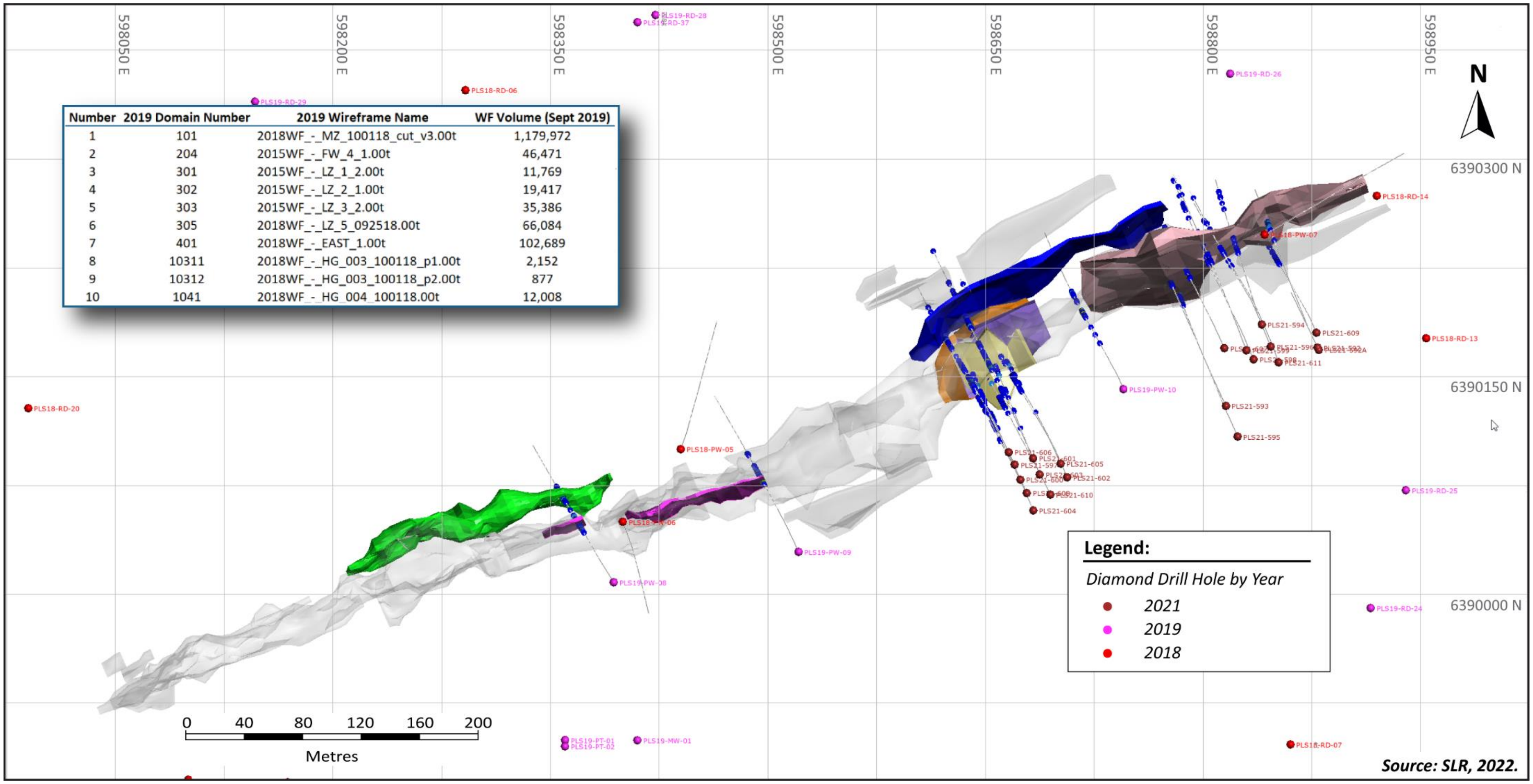
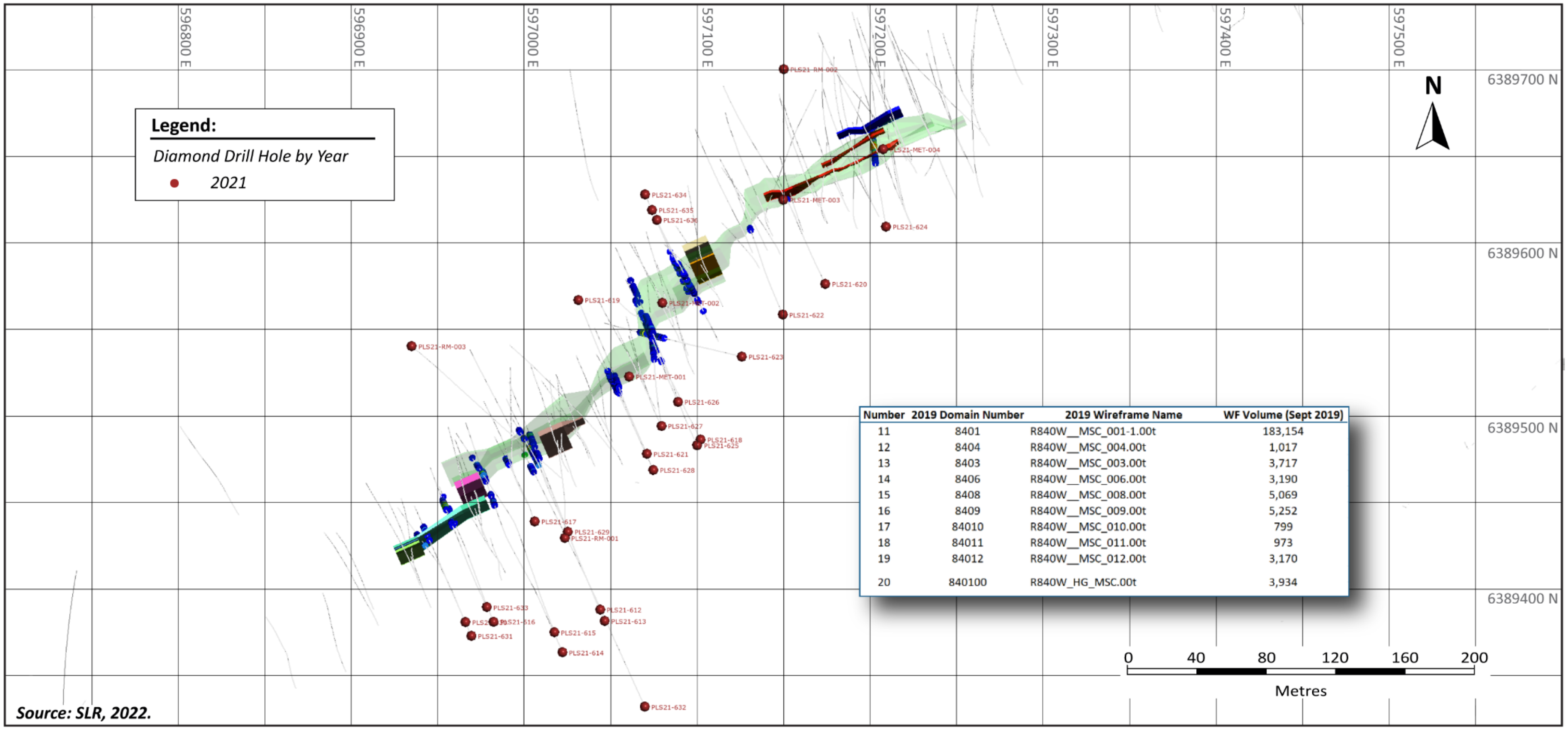


Figure 14-2: Wireframe Solids Longitudinal Section



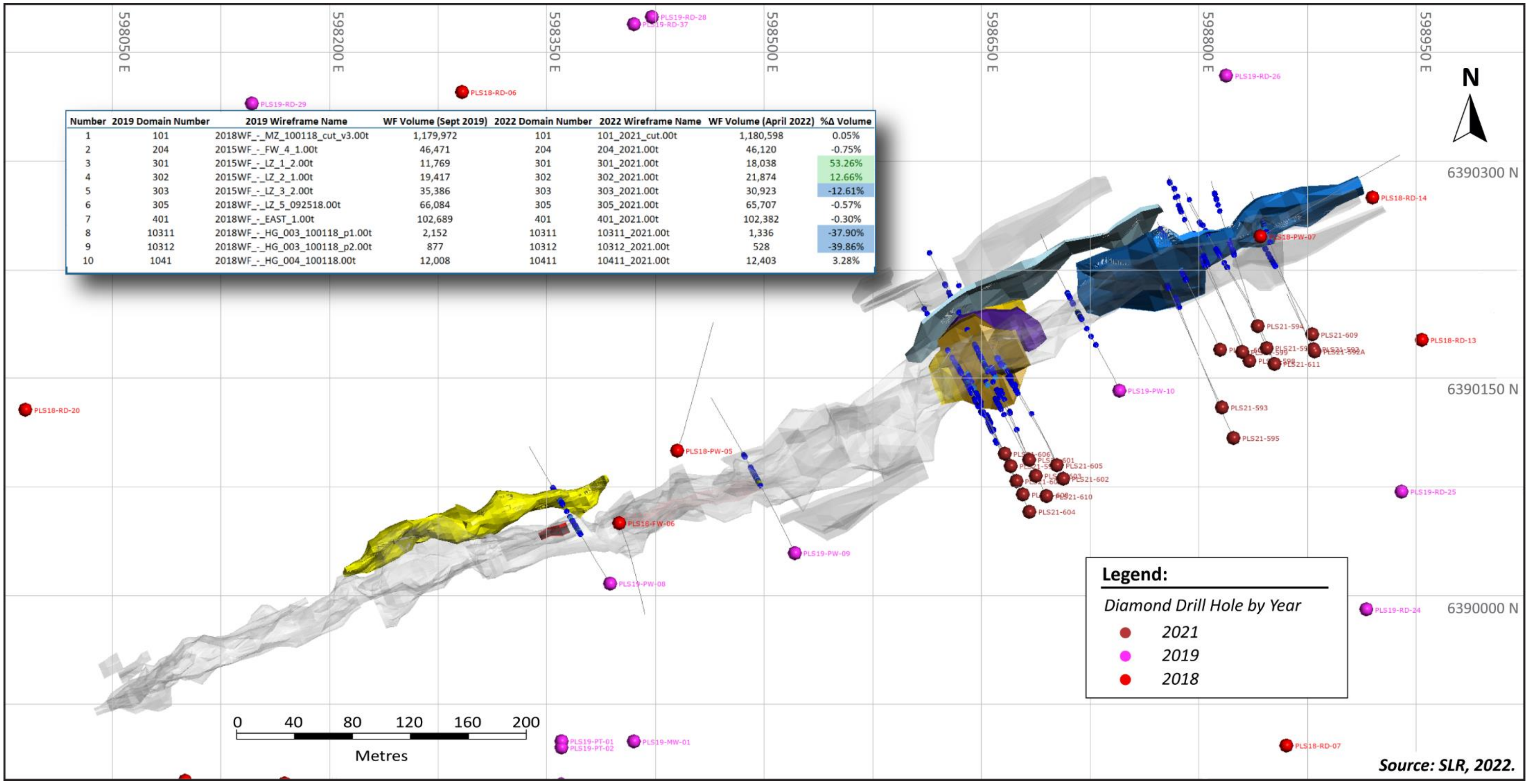
Number	2019 Domain Number	2019 Wireframe Name	WF Volume (Sept 2019)
1	101	2018WF_-_MZ_100118_cut_v3.00t	1,179,972
2	204	2015WF_-_FW_4_1.00t	46,471
3	301	2015WF_-_LZ_1_2.00t	11,769
4	302	2015WF_-_LZ_2_1.00t	19,417
5	303	2015WF_-_LZ_3_2.00t	35,386
6	305	2018WF_-_LZ_5_092518.00t	66,084
7	401	2018WF_-_EAST_1.00t	102,689
8	10311	2018WF_-_HG_003_100118_p1.00t	2,152
9	10312	2018WF_-_HG_003_100118_p2.00t	877
10	1041	2018WF_-_HG_004_100118.00t	12,008

Figure 14-3: 2022 R780E LG and HG Wireframe Volume



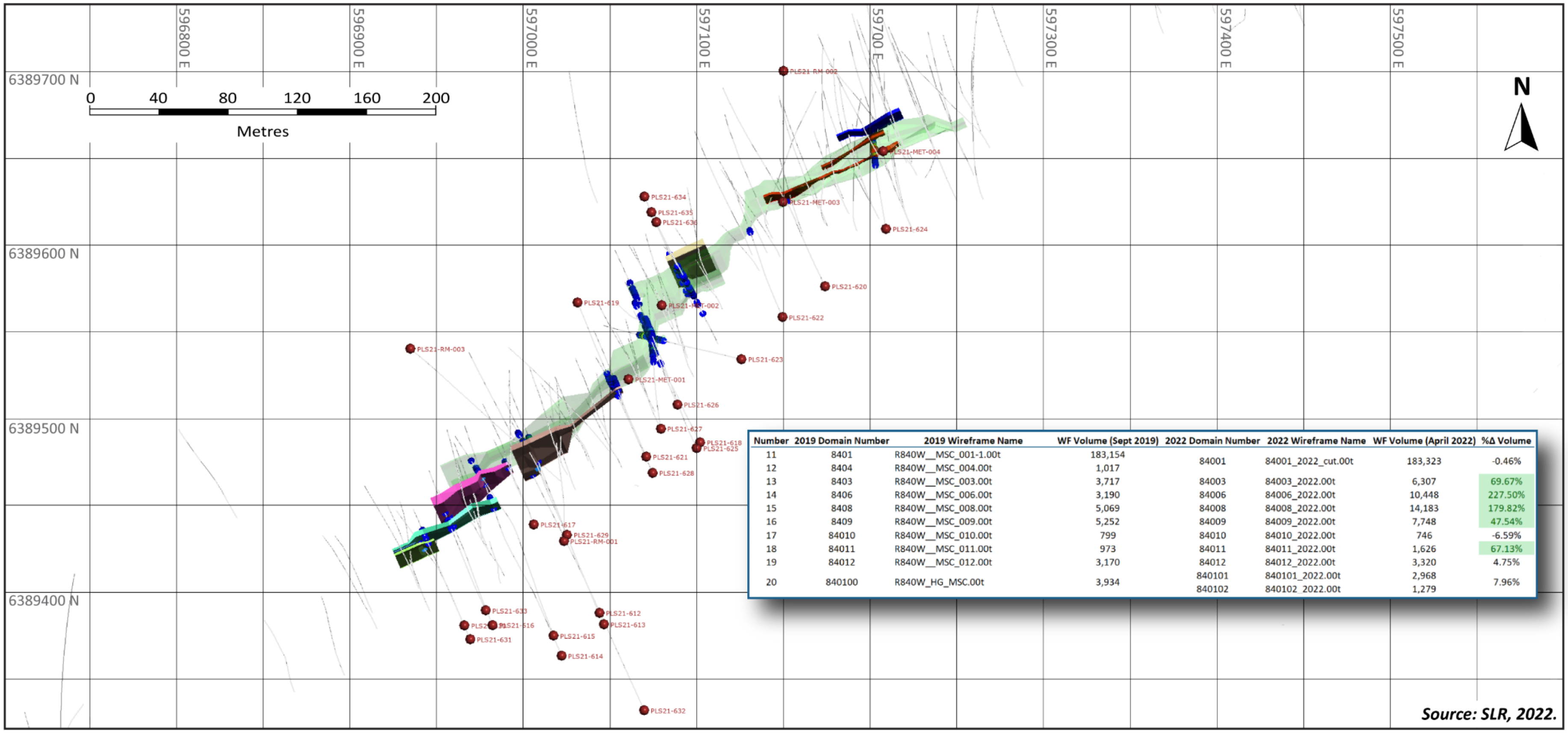
Number	2019 Domain Number	2019 Wireframe Name	WF Volume (Sept 2019)
11	8401	R840W__MSC_001-1.00t	183,154
12	8404	R840W__MSC_004.00t	1,017
13	8403	R840W__MSC_003.00t	3,717
14	8406	R840W__MSC_006.00t	3,190
15	8408	R840W__MSC_008.00t	5,069
16	8409	R840W__MSC_009.00t	5,252
17	84010	R840W__MSC_010.00t	799
18	84011	R840W__MSC_011.00t	973
19	84012	R840W__MSC_012.00t	3,170
20	840100	R840W_HG_MSC.00t	3,934

Figure 14-4: 2022 R840W LG and HG Wireframe Volume



Number	2019 Domain Number	2019 Wireframe Name	WF Volume (Sept 2019)	2022 Domain Number	2022 Wireframe Name	WF Volume (April 2022)	%Δ Volume
1	101	2018WF_-_MZ_100118_cut_v3.00t	1,179,972	101	101_2021_cut.00t	1,180,598	0.05%
2	204	2015WF_-_FW_4_1.00t	46,471	204	204_2021.00t	46,120	-0.75%
3	301	2015WF_-_LZ_1_2.00t	11,769	301	301_2021.00t	18,038	53.26%
4	302	2015WF_-_LZ_2_1.00t	19,417	302	302_2021.00t	21,874	12.66%
5	303	2015WF_-_LZ_3_2.00t	35,386	303	303_2021.00t	30,923	-12.61%
6	305	2018WF_-_LZ_5_092518.00t	66,084	305	305_2021.00t	65,707	-0.57%
7	401	2018WF_-_EAST_1.00t	102,689	401	401_2021.00t	102,382	-0.30%
8	10311	2018WF_-_HG_003_100118_p1.00t	2,152	10311	10311_2021.00t	1,336	-37.90%
9	10312	2018WF_-_HG_003_100118_p2.00t	877	10312	10312_2021.00t	528	-39.86%
10	1041	2018WF_-_HG_004_100118.00t	12,008	10411	10411_2021.00t	12,403	3.28%

Figure 14-5: 2022 R780E LG and HG Wireframe Volume



Number	2019 Domain Number	2019 Wireframe Name	WF Volume (Sept 2019)	2022 Domain Number	2022 Wireframe Name	WF Volume (April 2022)	%Δ Volume
11	8401	R840W__MSC_001-1.00t	183,154	84001	84001_2022_cut.00t	183,323	-0.46%
12	8404	R840W__MSC_004.00t	1,017				
13	8403	R840W__MSC_003.00t	3,717	84003	84003_2022.00t	6,307	69.67%
14	8406	R840W__MSC_006.00t	3,190	84006	84006_2022.00t	10,448	227.50%
15	8408	R840W__MSC_008.00t	5,069	84008	84008_2022.00t	14,183	179.82%
16	8409	R840W__MSC_009.00t	5,252	84009	84009_2022.00t	7,748	47.54%
17	84010	R840W__MSC_010.00t	799	84010	84010_2022.00t	746	-6.59%
18	84011	R840W__MSC_011.00t	973	84011	84011_2022.00t	1,626	67.13%
19	84012	R840W__MSC_012.00t	3,170	84012	84012_2022.00t	3,320	4.75%
20	840100	R840W_HG_MSC.00t	3,934	840101	840101_2022.00t	2,968	7.96%
				840102	840102_2022.00t	1,279	

Figure 14-6: 2022 R840W LG and HG Wireframe Volume

Table 14-5: September 2019 versus May 2022 Domain Wireframe Volume Change

Number	2019 Domain Number	2019 Wireframe Name	WF Volume (Sept 2019)	2022 Domain Number	2022 Wireframe Name	WF Volume (May 2022)	%Δ Volume
1	101	2018WF_-_MZ_100118_cut_v3.00t	1,179,972	101	101_2021_cut.00t	1,180,598	0.05%
2	204	2015WF_-_FW_4_1.00t	46,471	204	204_2021.00t	46,120	-0.75%
3	301	2015WF_-_LZ_1_2.00t	11,769	301	301_2021.00t	18,038	53.26%
4	302	2015WF_-_LZ_2_1.00t	19,417	302	302_2021.00t	21,874	12.66%
5	303	2015WF_-_LZ_3_2.00t	35,386	303	303_2021.00t	30,923	-12.61%
6	305	2018WF_-_LZ_5_092518.00t	66,084	305	305_2021.00t	65,707	-0.57%
7	401	2018WF_-_EAST_1.00t	102,689	401	401_2021.00t	102,382	-0.30%
8	10311	2018WF_-_HG_003_100118_p1.00t	2,152	10311	10311_2021.00t	1,336	-37.90%
9	10312	2018WF_-_HG_003_100118_p2.00t	877	10312	10312_2021.00t	528	-39.86%
10	1041	2018WF_-_HG_004_100118.00t	12,008	10411	10411_2021.00t	12,403	3.28%
11	8401	R840W__MSC_001-1.00t	183,154	84001	84001_2022_cut.00t	183,323	-0.46%
12	8404	R840W__MSC_004.00t	1,017				
13	8403	R840W__MSC_003.00t	3,717	84003	84003_2022.00t	6,307	69.67%
14	8406	R840W__MSC_006.00t	3,190	84006	84006_2022.00t	10,448	227.50%
15	8408	R840W__MSC_008.00t	5,069	84008	84008_2022.00t	14,183	179.82%
16	8409	R840W__MSC_009.00t	5,252	84009	84009_2022.00t	7,748	47.54%
17	84010	R840W__MSC_010.00t	799	84010	84010_2022.00t	746	-6.59%
18	84011	R840W__MSC_011.00t	973	84011	84011_2022.00t	1,626	67.13%
19	84012	R840W__MSC_012.00t	3,170	84012	84012_2022.00t	3,320	4.75%
20	840100	R840W_HG_MSC.00t	3,934	840101	840101_2022.00t	2,968	7.96%
				840102	840102_2022.00t	1,279	
Total	--	--	**1,687,099**	--	--	**1,711,859**	**1.47%**

Table 14-6: September 2019 versus May 2022 Domain Volume Change (0.25% U_3O_8 COG)

Number	2019 Domain Number	2019 Wireframe Name	Block Volume (Sept 2019)	2022 Domain Number	2022 Wireframe Name	Block Volume (May 2022)	Percent Change (Volume)
1	101	2018WF_-_MZ_100118_cut_v3.00t	680,100	101	101_2021_cut.00t	677,490	-0.38%
2	204	2015WF_-_FW_4_1.00t	20,700	204	204_2021.00t	19,310	-6.71%
3	301	2015WF_-_LZ_1_2.00t	10,650	301	301_2021.00t	13,550	27.23%
4	302	2015WF_-_LZ_2_1.00t	17,720	302	302_2021.00t	17,650	-0.40%
5	303	2015WF_-_LZ_3_2.00t	29,550	303	303_2021.00t	23,580	-20.20%
6	305	2018WF_-_LZ_5_092518.00t	26,830	305	305_2021.00t	29,150	8.65%
7	401	2018WF_-_EAST_1.00t	77,930	401	401_2021.00t	69,610	-10.68%
8	10311	2018WF_-_HG_003_100118_p1.00t	2,040	10311	10311_2021.00t	1,620	-20.59%
9	10312	2018WF_-_HG_003_100118_p2.00t	730	10312	10312_2021.00t	550	-24.66%
10	1041	2018WF_-_HG_004_100118.00t	12,260	10411	10411_2021.00t	12,520	2.12%
11	8401	R840W__MSC_001-1.00t	12,760	84001	84001_2022_cut.00t	127,700	-6.08%
12	8404	R840W__MSC_004.00t	123,210				
13	8403	R840W__MSC_003.00t	1,530	84003	84003_2022.00t	2,000	30.72%
14	8406	R840W__MSC_006.00t	3,420	84006	84006_2022.00t	7,050	106.14%
15	8408	R840W__MSC_008.00t	4,960	84008	84008_2022.00t	12,640	154.84%
16	8409	R840W__MSC_009.00t	5,330	84009	84009_2022.00t	6,150	15.38%
17	84010	R840W__MSC_010.00t	430	84010	84010_2022.00t	430	0.00%
18	84011	R840W__MSC_011.00t	840	84011	84011_2022.00t	1,580	88.10%
19	84012	R840W__MSC_012.00t	470	84012	84012_2022.00t	490	4.26%
20	840100	R840W_HG_MSC.00t	3,980	840101	840101_2022.00t	2,980	8.04%
				840102	840102_2022.00t	1,320	
Total	**--**	**--**	**1,035,440**	**--**	**--**	**1,027,370**	**-0.78%**

Table 14-7: September 2019 versus May 2022 Domain Grade Change (0.25% U_3O_8 COG)

Number	2019 Domain Number	2019 Wireframe Name	2019 Grade (% U_3O_8)	2022 Domain Number	2022 Wireframe Name	2022 Grade (% U_3O_8)	Percent Change (Grade)
1	101	2018WF_-_MZ_100118_cut_v3.00t	0.81	101	101_2021_cut.00t	0.79	-2.83%
2	204	2015WF_-_FW_4_1.00t	0.37	204	204_2021.00t	0.38	4.93%
3	301	2015WF_-_LZ_1_2.00t	2.02	301	301_2021.00t	1.15	-43.15%
4	302	2015WF_-_LZ_2_1.00t	1.55	302	302_2021.00t	1.27	-17.87%
5	303	2015WF_-_LZ_3_2.00t	1.14	303	303_2021.00t	1.16	2.42%
6	305	2018WF_-_LZ_5_092518.00t	0.48	305	305_2021.00t	0.53	9.79%
7	401	2018WF_-_EAST_1.00t	0.86	401	401_2021.00t	0.95	11.09%
8	10311	2018WF_-_HG_003_100118_p1.00t	22.96	10311	10311_2021.00t	23.33	1.63%
9	10312	2018WF_-_HG_003_100118_p2.00t	16.21	10312	10312_2021.00t	16.46	1.50%
10	1041	2018WF_-_HG_004_100118.00t	11.01	10411	10411_2021.00t	11.47	4.10%
11	8401	R840W__MSC_001-1.00t	1.67	84001	84001_2022_cut.00t	1.21	-20.03%
12	8404	R840W__MSC_004.00t	1.50				
13	8403	R840W__MSC_003.00t	0.58	84003	84003_2022.00t	0.54	-6.26%
14	8406	R840W__MSC_006.00t	2.14	84006	84006_2022.00t	1.22	-43.12%
15	8408	R840W__MSC_008.00t	1.22	84008	84008_2022.00t	2.12	74.56%
16	8409	R840W__MSC_009.00t	4.62	84009	84009_2022.00t	2.22	-51.99%
17	84010	R840W__MSC_010.00t	0.83	84010	84010_2022.00t	0.82	-1.30%
18	84011	R840W__MSC_011.00t	2.06	84011	84011_2022.00t	1.84	-10.65%
19	84012	R840W__MSC_012.00t	0.36	84012	84012_2022.00t	0.35	-2.72%
20	840100	R840W_HG_MSC.00t	10.89	840101	840101_2022.00t	11.51	2.22%
				840102	840102_2022.00t	10.86	
Total	**--**	**--**	**1.17**	**--**	**--**	**1.11**	**-5.35%**

Table 14-8: September 2019 versus May 2022 Domain Contained Metal Change (0.25% U_3O_8 COG)

Number	2019 Domain Number	2019 Wireframe Name	Contained Metal (lb U_3O_8)	2022 Domain Number	2022 Wireframe Name	2022 Contained Metal (lb U_3O_8)	Percent Change (Contained Metal)
1	101	2018WF_-_MZ_100118_cut_v3.00t	28,452,000	101	101_2021_cut.00t	27,551,000	-3.2%
2	204	2015WF_-_FW_4_1.00t	411,000	204	204_2021.00t	401,000	-2.4%
3	301	2015WF_-_LZ_1_2.00t	1,120,000	301	301_2021.00t	823,000	-26.5%
4	302	2015WF_-_LZ_2_1.00t	1,411,000	302	302_2021.00t	1,144,000	-18.9%
5	303	2015WF_-_LZ_3_2.00t	1,704,000	303	303_2021.00t	1,377,000	-19.2%
6	305	2018WF_-_LZ_5_092518.00t	670,000	305	305_2021.00t	804,000	20.0%
7	401	2018WF_-_EAST_1.00t	3,491,000	401	401_2021.00t	3,449,000	-1.2%
8	10311	2018WF_-_HG_003_100118_p1.00t	2,746,000	10311	10311_2021.00t	2,214,000	-19.4%
9	10312	2018WF_-_HG_003_100118_p2.00t	575,000	10312	10312_2021.00t	437,000	-24.0%
10	1041	2018WF_-_HG_004_100118.00t	7,030,000	10411	10411_2021.00t	7,469,000	6.2%
11	8401	R840W__MSC_001-1.00t	1,080,000	84001	84001_2022_cut.00t	7,702,000	-25.6%
12	8404	R840W__MSC_004.00t	9,279,000				
13	8403	R840W__MSC_003.00t	45,000	84003	84003_2022.00t	56,000	24.4%
14	8406	R840W__MSC_006.00t	411,000	84006	84006_2022.00t	462,000	12.4%
15	8408	R840W__MSC_008.00t	299,000	84008	84008_2022.00t	1,312,000	338.8%
16	8409	R840W__MSC_009.00t	1,422,000	84009	84009_2022.00t	734,000	-48.4%
17	84010	R840W__MSC_010.00t	19,000	84010	84010_2022.00t	19,000	0.0%
18	84011	R840W__MSC_011.00t	74,000	84011	84011_2022.00t	137,000	85.1%
19	84012	R840W__MSC_012.00t	9,000	84012	84012_2022.00t	9,000	0.0%
20	840100	R840W_HG_MSC.00t	1,983,000	840101	840101_2022.00t	1,574,000	11.4%
				840102	840102_2022.00t	635,000	
Total	**--**	**--**	**62,232,000**	**--**	**--**	**58,308,000**	**-6.31%**

The R780E_MZ domain (which contains both a low grade and high grade domain) is the largest continuous domain within the R780E area. The MZ is elongated in the grid east-west direction, dips steeply to the south, and measures approximately 740 m along strike. Both the down dip and true thickness of the MZ vary due to the irregular shape of the mineralization, however, in general, the down dip measurement ranges between 50 m and 80 m, and the true thickness is in most places between 20 m and 30 m but can be as little as 2 m to a maximum of 45 m.

The R780E_HG domain consists of 16 lenses within the R780E_MZ low grade zone. The R780E_HG domain alone contains more than half the contained pounds of U_3O_8 classified as Indicated Mineral Resources. It was modelled as seven steeply dipping wireframe solids located within the R780E_MZ. The high grade zones span over 500 m of strike length, measure from 10 m to 40 m down dip, and range from 3 m to 10 m thick. Combined, the R780_MZ and R780E_HG domains account for approximately 68% of total contained pounds of U_3O_8 in the Mineral Resource.

A number of other wireframe solids make up a smaller portion of the Mineral Resources. Most of the secondary domains are oriented similarly to the MZ. Some, including R00E, were modelled with a horizontal orientation. Additional drilling is recommended to better define the geometry of mineralization.

14.3 Statistical Analysis

Assay values located inside the wireframe models were tagged with domain identifiers and exported for statistical analysis. Results were used to help verify the modelling process. Basic statistics by domain are summarized in Table 14-9 and histograms of resource assays for each domain are illustrated in Figure 14-7 to Figure 14-14.

Table 14-9: Summary Statistics of Uncapped % U_3O_8 Assays

Zone	Count	Min	Max	Mean	Variance	SD	CV
R780E_MZ	17,891	0.00	43.50	0.56	3.26	1.81	3.22
R780E_HG	1,464	0.00	65.70	15.16	186.40	13.65	0.90
R780E_Other	5,261	0.00	59.20	0.77	8.03	2.83	3.68
R00E	878	0.00	48.80	1.85	25.28	5.03	2.71
R1620E	853	0.00	36.80	1.78	19.11	4.37	2.46
R840W	3,681	0.00	41.20	0.94	7.70	2.77	2.96
R840W_HG	179	0.01	52.30	14.64	148.60	12.19	0.83
R1515W	1,769	0.00	30.90	0.94	5.58	2.36	2.51

CV = coefficient of variation

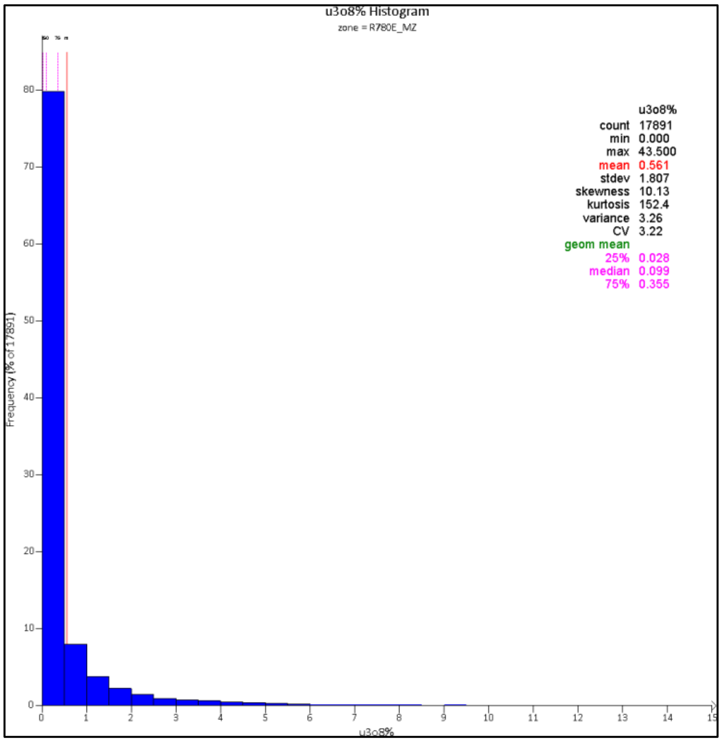


Figure 14-7: Histogram of Resource Assays in R780E_MZ Domain

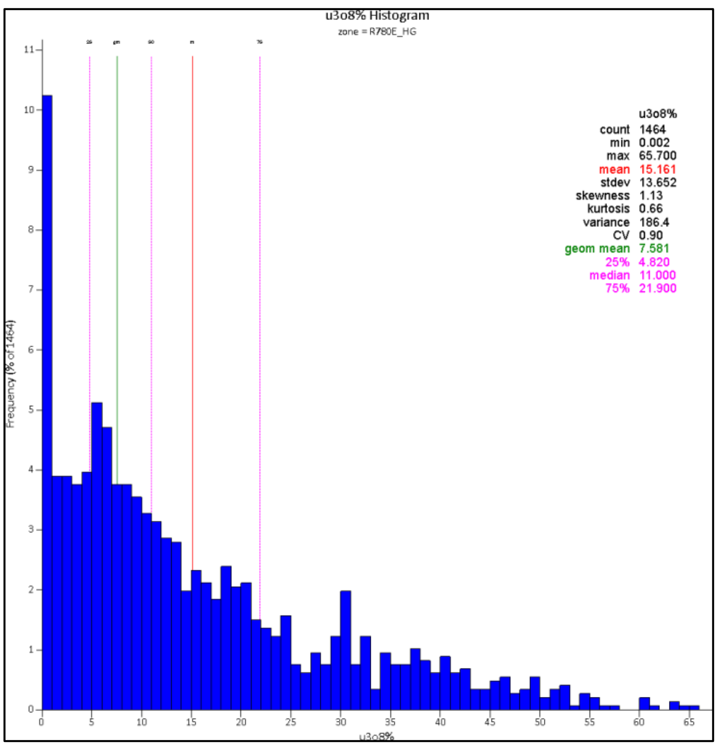


Figure 14-8: Histogram of Resource Assays in R780E_HG Domain

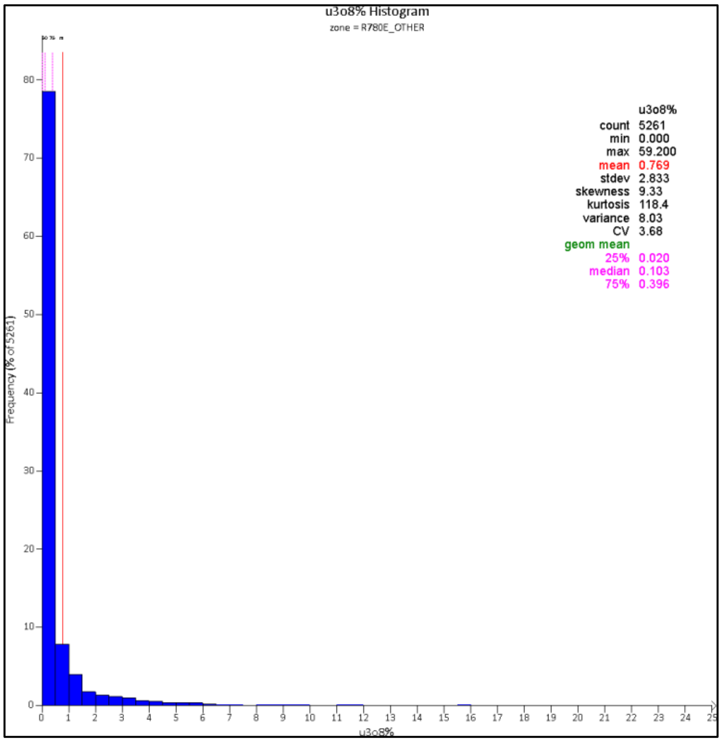


Figure 14-9: Histogram of Resource Assays in R780E_Other Zone Domain

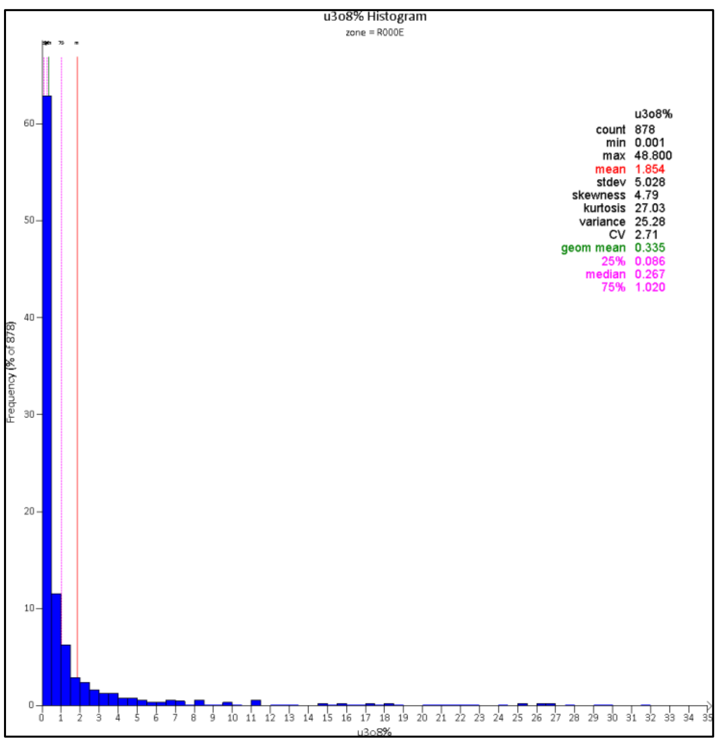


Figure 14-10: Histogram of Resource Assays in R00E Zone Domain

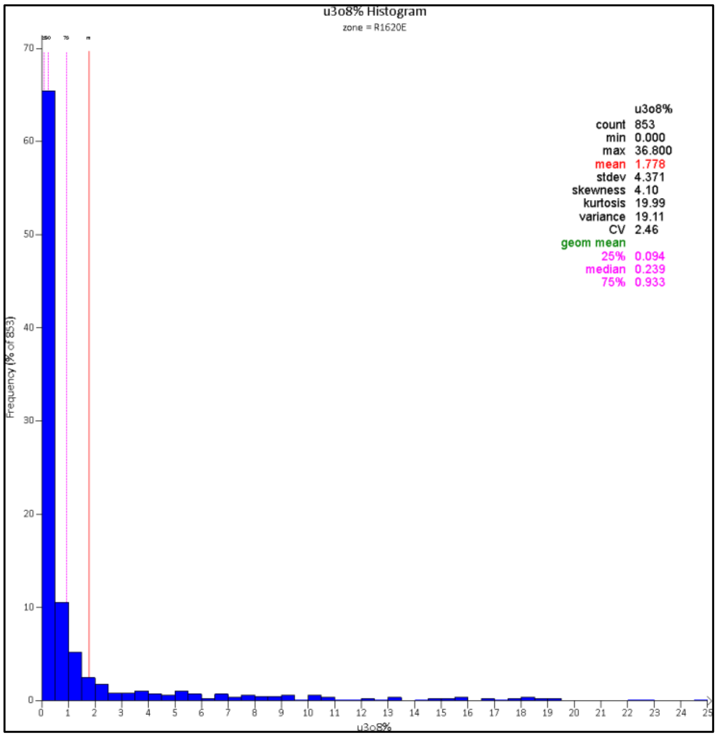


Figure 14-11: Histogram of Resource Assays in R1620E Zone Domain

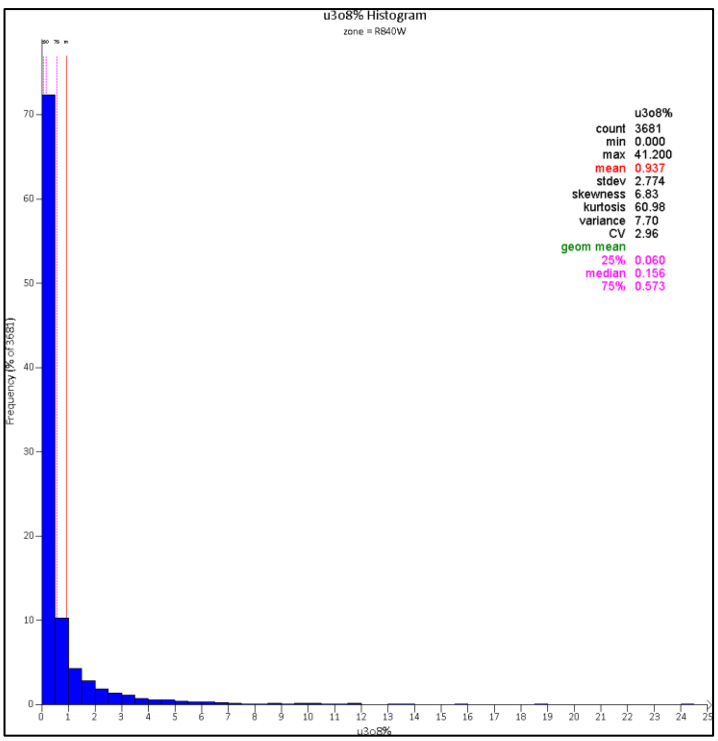


Figure 14-12: Histogram of Resource Assays in R840W Zone Domain

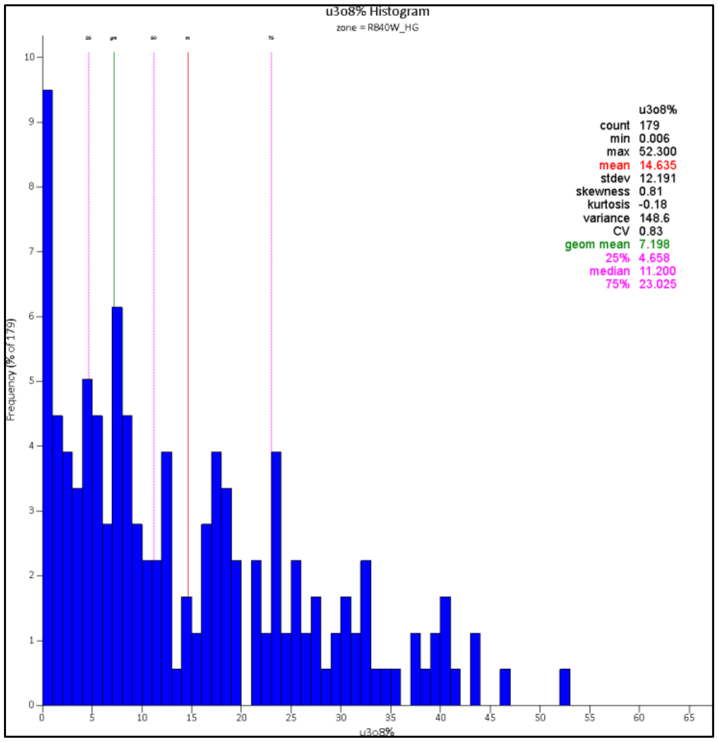


Figure 14-13: Histogram of Resource Assays in R840W-HG Zone Domain

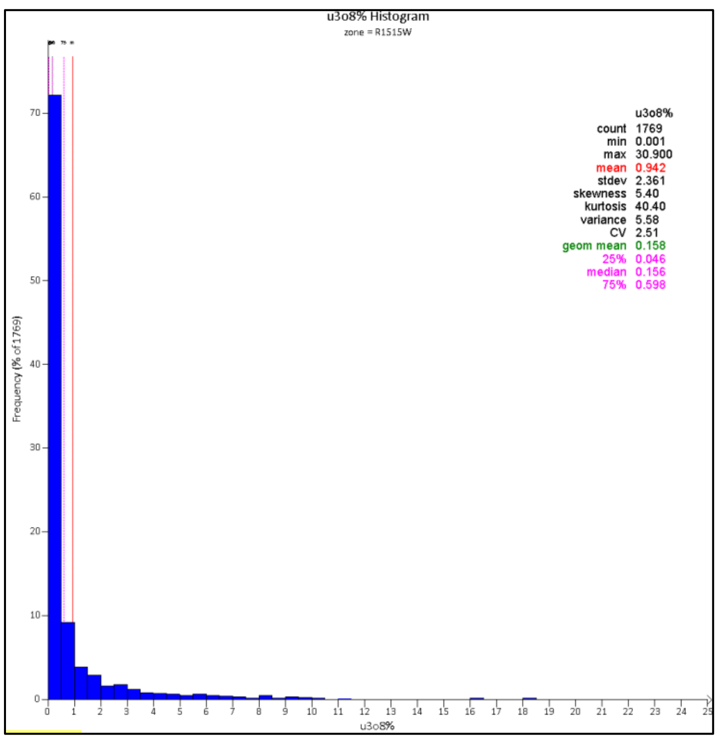


Figure 14-14: Histogram of Resource Assays in R1515W Zone Domain

14.3.1 Cutting High Grade Values

Where the assay distribution is skewed positively or approaches log-normal, erratic high grade assay values can have a disproportionate effect on the average grade of a deposit. One method of treating these outliers in order to reduce their influence on the average grade is to cut or cap them at a specific grade level.

The SLR QP is of the opinion that the influence of high grade uranium assays must be reduced or controlled and uses a number of industry best practice methods to achieve this goal, including capping of high grade values. Assessing the influence of outliers involves several statistical analytical methods to determine an appropriate capping value including preparation of frequency histograms, probability plots, decile analyses, and capping curves. Using these methodologies, SLR revisited the selected capping values for each of the 80 mineralized domains and 9 zones in the Triple R deposit.

Review of the resource assay within the wireframe domains and a visual inspection of high grade values on vertical sections suggest cutting high grade values to 7%, 10%, 15%, and 20% U_3O_8 in the low grade domains and 55% and 35% U_3O_8 in the R780E_HG and R840W_HG domains, respectively, resulting in a total of 312 (0.98%) capped U_3O_8 assay values (Table 14-10). Gold assays were capped at 15 g/t Au within the R780E_HG domain, and 10 g/t Au in all remaining wireframes resulting in a total of 236 (0.74%) capped gold assay values (Table 14-10). Examples of the capping analysis are shown in Figure 14-15 and Figure 14-16.

Table 14-10: Capping of Resource Assay Values by Zone

Zone	Cap Levels (% U_3O_8)	Number of Assays	Number U_3O_8 Assays Capped	Number Au Assays Capped	% U_3O_8 Capped	% Au Capped
R780E_MZ	10	17,891	105	89	0.59%	0.50%
R780E_HG	55	1,464	13	41	0.89%	2.80%
R780E_OTHER	7	561	17	13	3.03%	2.32%
	10	4,227	45	41	1.06%	0.97%
	20	473	4	8	0.85%	1.69%
R00E	10	878	43	3	4.90%	0.34%
R1620E	20	853	10	4	1.17%	0.47%
R840W	7	113	4	0	3.54%	0.00%
	15	2,871	30	12	1.04%	0.42%
	20	697	2	2	0.29%	0.29%
R840W_HG	35	179	14	14	7.82%	7.82%
R1515W	7	196	12	1	6.12%	0.51%
	10	1,573	13	8	0.83%	0.51%
Total	**--**	**31,976**	**312**	**236**	**0.98%**	**0.74%**

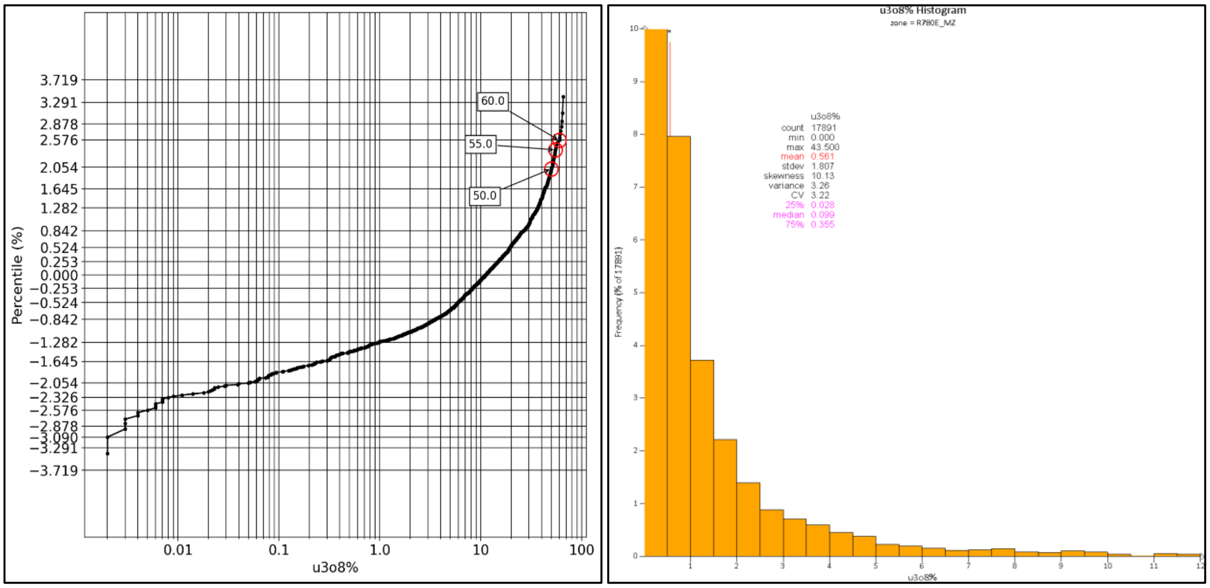


Figure 14-15: Log Probability and Histogram of Resource Assays in R780E_HG Domain

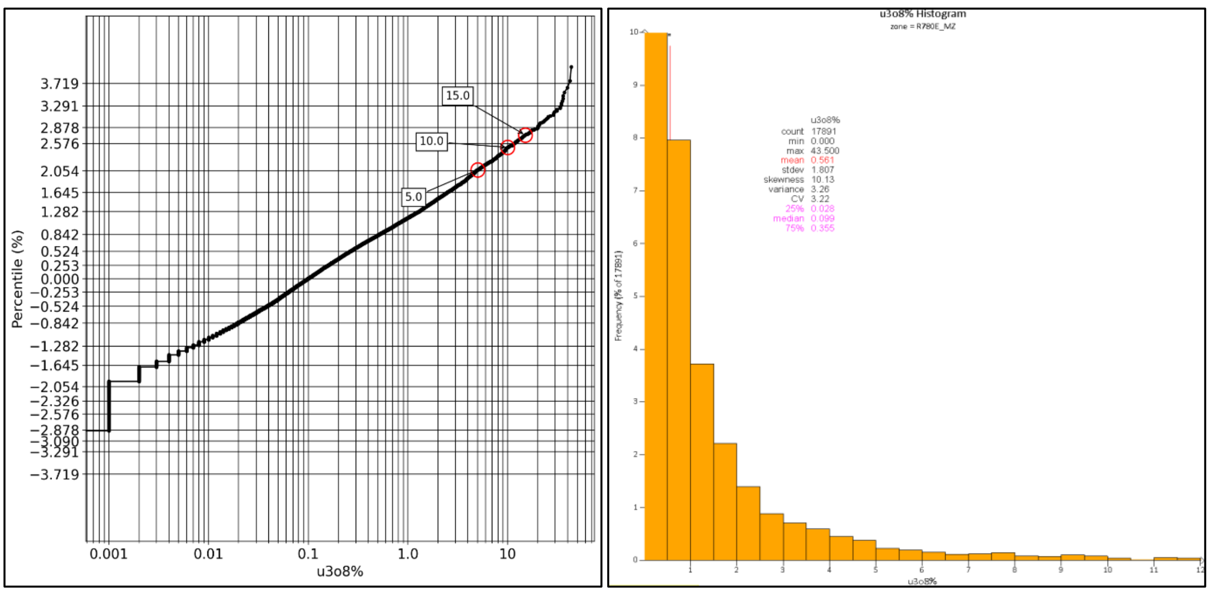


Figure 14-16: Log Probability and Histogram of Resource Assays in R780E_MZ Domain

Capped assay statistics by zones are summarized in Table 14-11 and compared with uncapped assay statistics. For the R780E_MZ domain, by cutting 132 high values to 10% U_3O_8, the average grade was reduced from 0.59% U_3O_8 to 0.54% U_3O_8 and the CV was reduced from 3.27 to 2.50. For the R780E_HG domain, by cutting 14 high values to 55% U_3O_8, the average grade was reduced from 14.74% U_3O_8 to 14.70% U_3O_8 and CV was reduced from 0.92 to 0.91

Table 14-11: Summary Statistics of Uncapped vs. Capped Assays

Zone	R780E_MZ		R780E_HG		R780E_Other		R00E	
Descriptive Statistics	Raw	Cap	Raw	Cap	Raw	Cap	Raw	Cap
Number of Samples	17,891	17,891	1,464	1,464	5,261	5,261	878	878
Min	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Max	43.50	10.00	65.70	55.00	59.20	20.00	48.80	10.00
Mean	0.56	0.51	15.16	15.12	0.77	0.63	1.85	1.31
Variance	3.26	1.64	186.40	182.50	8.03	2.73	25.28	6.23
SD	1.81	1.28	13.65	13.51	2.83	1.65	5.03	2.50
CV	3.22	2.49	0.90	0.89	3.68	2.62	2.71	1.91
Number of Caps	0	105	0	13	0	66	0	43
Zone	R1620E		R840W		R840W_HG		R1515W	
Descriptive Statistics	Raw	Cap	Raw	Cap	Raw	Cap	Raw	Cap
Number of Samples	853	853	3,681	3,681	179	179	1,769	1,769
Min	0.00	0.00	0.00	0.00	0.01	0.01	0.00	0.00
Max	36.80	20.00	41.20	20.00	52.30	35.00	30.90	10.00
Mean	1.78	1.69	0.94	0.87	14.64	14.16	0.94	0.85
Variance	19.11	14.68	7.70	4.74	148.60	124.50	5.58	3.17
SD	4.37	3.83	2.77	2.18	12.19	11.16	2.36	1.78
CV	2.46	2.27	2.96	2.52	0.83	0.79	2.51	2.09
Number of Caps	0	10	0	36	0	14	0	25

14.3.2 Compositing

Composites were created from the capped assay values using hard boundaries from the wireframe models using the downhole compositing function of the Vulcan modelling software package. Sample lengths range from a few cm to 3.0 m within the wireframe models, with 99% of the samples taken at 0.5 m intervals (Figure 14-17). The composite lengths used during interpolation were chosen considering the predominant sampling length, the minimum mining width, style of mineralization, and continuity of grade. Given this distribution, and considering the width of the mineralization, SLR chose to composite to 2 m lengths.

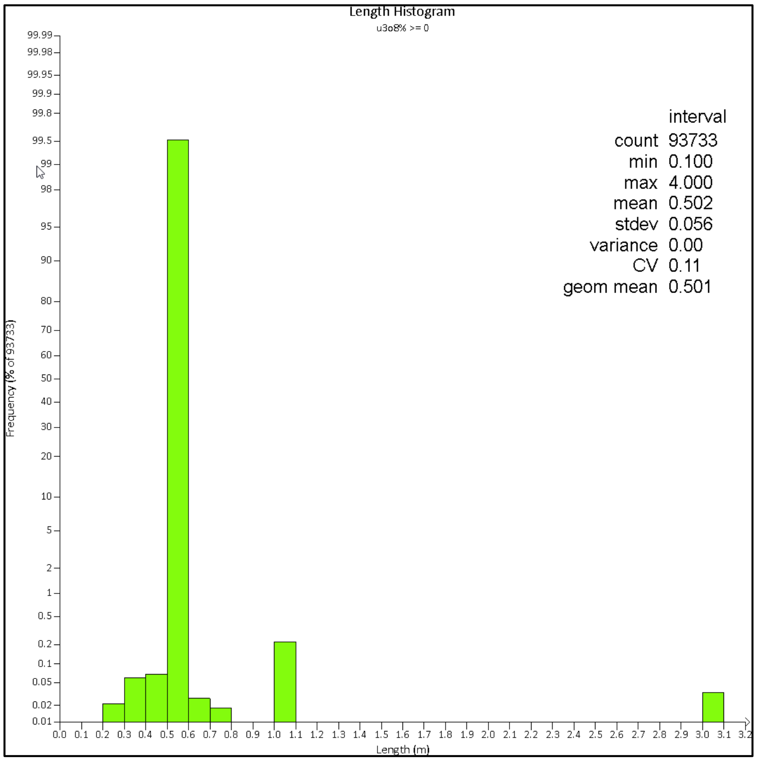


Figure 14-17: Histogram of Sampling Length

Assays within the wireframe domains were composited starting at the first mineralized wireframe boundary from the collar and resetting at each new wireframe boundary. Composites less than 0.53 m, located at the bottom of the mineralized intercept, were added to the previous interval. Table 14-12 shows the composite statistics by domain.

Table 14-12: Descriptive Statistics of Composite Values by Domain

Zone	Count	Min	Max	Mean	Variance	SD	CV
R780E_MZ	4,717	0.00	10.00	0.51	0.91	0.95	1.87
R780E_HG	386	0.06	53.18	15.17	116.60	10.80	0.71
R780E_Other	1,424	0.00	16.68	0.60	1.57	1.25	2.09
R00E	223	0.03	10.00	1.28	4.71	2.17	1.69
R1620E	223	0.00	18.73	1.64	10.12	3.18	1.94
R840W	993	0.00	15.00	0.83	3.30	1.82	2.20
R840W_HG	49	0.01	34.81	13.53	89.45	9.46	0.70
R1515W	453	0.003	7.989	0.842	1.79	1.339	1.59

14.3.3 Continuity Analysis

SLR generated downhole and directional variograms using the 2-m composite U_3O_8 values located within the R780E_MZ mineralized wireframe including high grade mineralization. The downhole variogram suggests a relative nugget effect of less than 12% (Figure 14-18). Variograms were of poor to fair quality considering the number of composite data and not adequate to generate meaningful variograms to derive kriging parameters. Long range directional variograms were focused in the plane of mineralization, which most commonly strikes northeast and dips steeply to the southeast. To improve the variogram for the MZ, only composite values ranging from 0.10% U_3O_8 to 20% U_3O_8 were used (Figure 14-19). Most ranges were interpreted to be approximately 20 m to 30 m. These ranges were used to derive search ellipse dimensions for block interpolations.

SLR also visually reviewed and contoured the drill hole results to identify trends of high grade mineralization. Several shallow to moderately eastward plunging higher grade zones were identified and these were mostly modelled as part of the HG domain within the MZ.

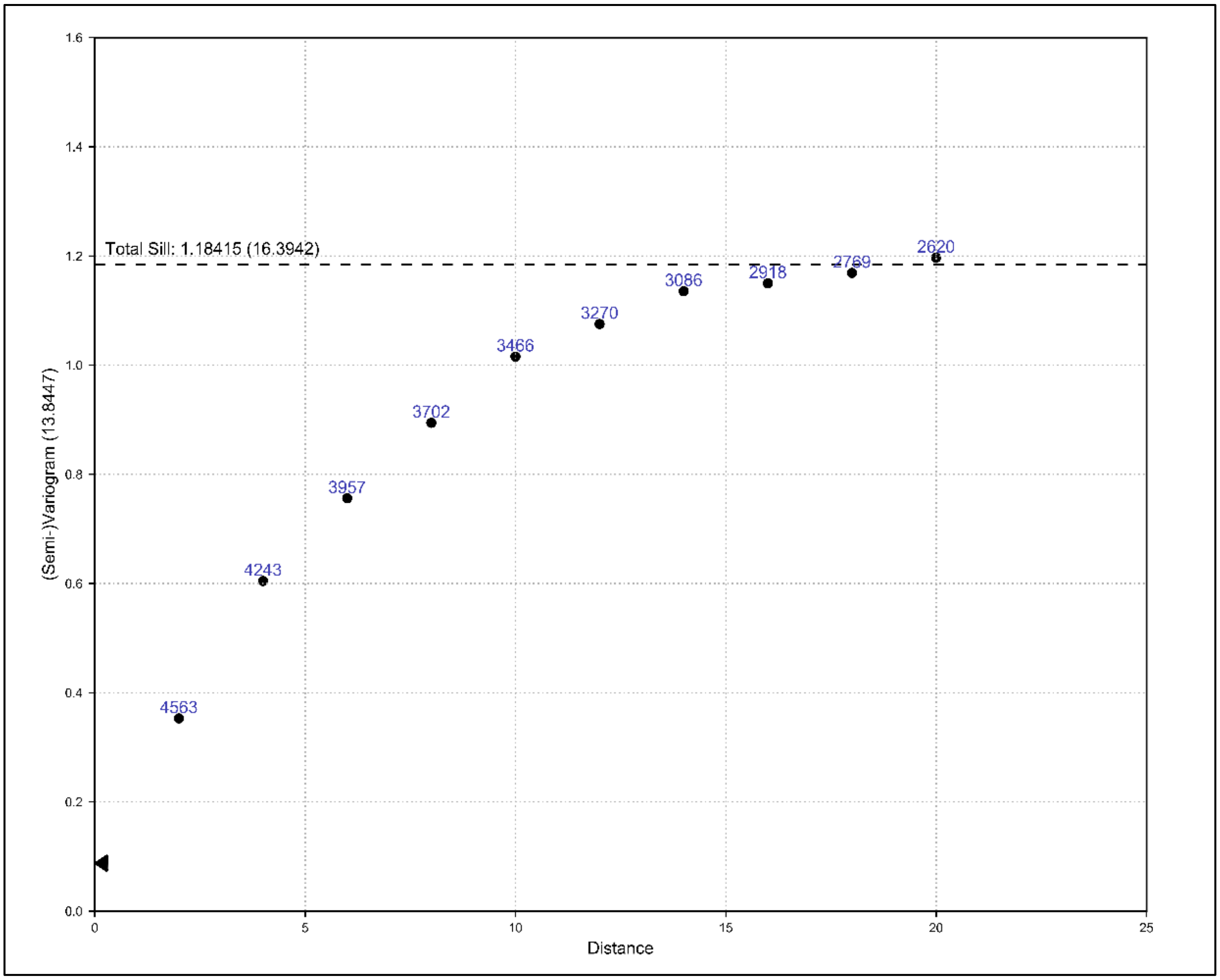


Figure 14-18: Downhole Variogram

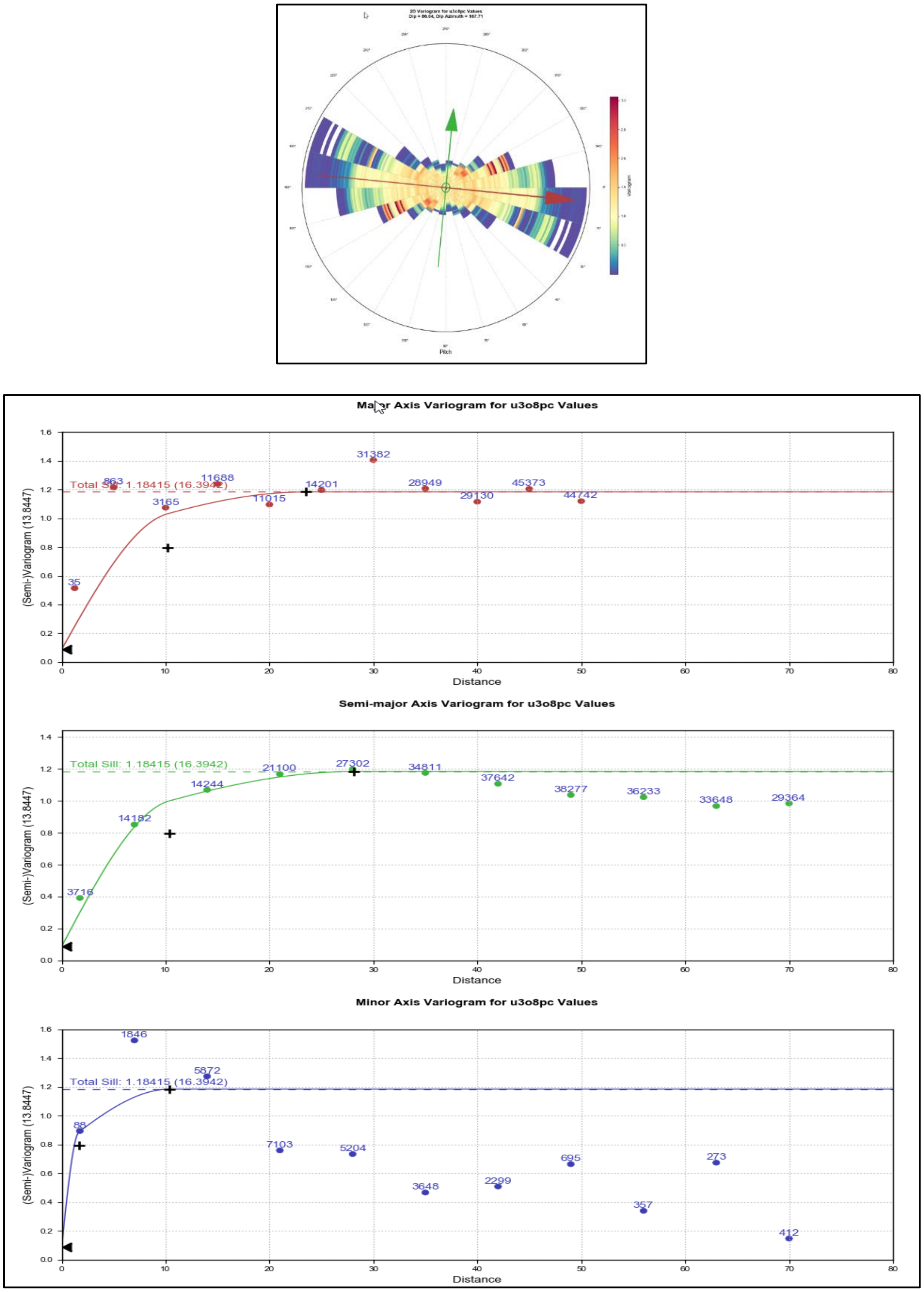


Figure 14-19: Directional Variograms for R780E_MZ Deposit

14.4 Density

Bulk density estimates are used to convert volume to tonnage and, in some cases, can be used to weight block grade estimates. For example, high grade uranium deposits of the Athabasca Basin have bulk densities that commonly vary with grade due to the very high density of pitchblende/uraninite compared to host lithologies. Bulk density also varies with clay alteration and in situ rock porosity. When modelling high grade uranium deposits, it is common to estimate bulk density values throughout the deposit and to weight uranium grades by density, since small volumes of high grade material contain large quantities of uranium oxide.

SLR carried out correlation analyses of the bulk density measurements against uranium grades. Unlike most deposits in the Athabasca Basin, the high grade uranium mineralization at the Triple R deposit has relatively low density values. Uranium grade ranges of 20% U_3O_8 to 70% U_3O_8, within the Athabasca Basin, more commonly exhibit density values ranging from 3.0 t/m^3 to 6.0 t/m^3 correlated with grade. Triple R high grade mineralization is often associated with carbon which may account for the lower than expected density values. In general, the average density of mineralization commonly ranges from 2.25 t/m^3 to 2.41 t/m^3.

Since bulk density does not have a clear correlation with grade, SLR did not weight grades by density in the block interpolation. Block grade values and density values were estimated independently.

Block densities were estimated from the density measurements using inverse distance cubed (ID^3) and a similar search strategy as used for uranium grade. Hard boundaries were used between domains. The Triple R resource database includes 17,509 density measurements of which 15,920 were used in the resource estimation. Table 14-13 compares the average densities of the blocks within the mineralized zones to the average densities of measurements associated with grades greater than 0.1% U_3O_8.

Table 14-13: Comparison of Estimated Block Densities and Measured Core Densities by Zone

Zone	Blocks (t/m^3)	Measurements (t/m^3)
R780E_MZ	2.40	2.35
R780E_HG	2.35	2.35
R780E_OTHER	2.38	2.37
R00E	2.27	2.28
R1620E	2.33	2.35
R840W	2.28	2.27
R840W_HG	2.06	2.04
R1515W	2.30	2.28

14.5 Block Model

The Vulcan block model origin (lower-left corner at lowest elevation) is at UTM coordinates 596,304.821 mE, 6,388,726.026 mN, and 0.0 m elevation and is made up of 737 columns, 380 rows, and 270 levels. Each block is 1 m wide, 2 m high, and 5 m along strike. A regularized whole block approach was used whereby the block was assigned to the domain where its centroid was located. The models fully enclose the modelled resource wireframes and are oriented with an azimuth of 66.2°, dip of 0.0°, and a plunge of 0.0° so as to align with the overall strike of the mineralization within the given model area. A summary of the block model extents is provided in Table 14-14.

A number of attributes were created to store such information as bulk density, estimated uranium grades, wireframe code, Mineral Resource classification, etc., for each block model area as listed in Table 14-15.

Table 14-14: Block Model Set-Up

Item	Value	Schematic
Origin		
X minimum	596,304.821	
Y minimum	6,388,726.026	
Z minimum	0	
Offset:		
X	3,685	
Y	760	
Z	540	
Block Size		
X	5	
Y	1	
Z	2	
Number of Blocks		
X	737	
Y	760	
Z	270	
Model Rotation		
Bearing	66.2	
Plunge	0	
Dip	0	
Other		
Project Units	Metres	
Coordinate System	NAD83 UTM Zone12N	
Number of Blocks	151,232,400	

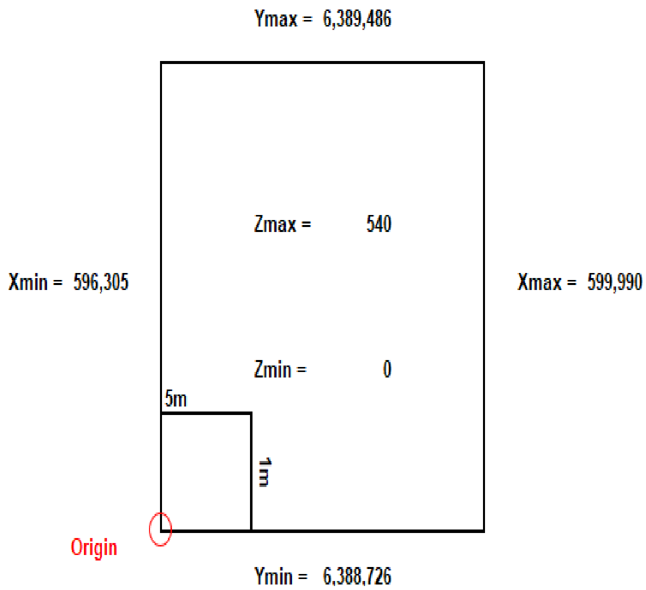


Table 14-15: Triple R Block Model Parameters and Variables

Variable	Default Value	Data Type	Description
au	-99	double	Au grade (g/t)
class	-99	integer	Classification
den	-99	double	Estimated measured density
den2	-99	double	Calculated polynomial density
est_flag_id	-99	integer	Estimation flag ID3
est_flag_ok	-99	integer	Estimation flag OK
grade_id3	-99	double	$\%U_3O_8$ ID3
grade_ok	-99	double	$\%U_3O_8$ grade OK
gxd	-99	double	grade_id3 x density
gxd_d	-99	double	Calculated (density (den2) weighted) $\%U_3O_8$
gxd2	-99	double	$\%U_3O_8$ * density (den2)
litho	unclass	name	Lithology
nholes	-99	short	Number of holes
nn	-99	double	Nearest neighbour
nn_distance	-99	double	Distance to nearest neighbour
nsamp	-99	short	Number of samples
open_pit	-99	double	Open pit flag
ore	-99	integer	Mineralized domain (wireframe)
ug	-99	double	Underground flag
topo_flag	-99	double	Topo flag
overburden	-99	double	Overburden flag

14.6 Interpolation Parameters

Grade interpolations for U_3O_8 and gold were carried out using ID^3 in a single pass with a minimum of two to a maximum of seven composites per block estimate. The search ellipse orientation varied slightly by domain. Hard boundaries were used to limit the use of composites between domains. Most search ellipse dimensions were 50 m by 50 m by 10 m for a 5:5:1 anisotropic ratio.

To reduce the influence of high grade composites, grades greater than a designated threshold level for some domains were restricted to a search ellipse dimension of 25 m by 25 m by 5 m (high yield restriction). The threshold grade levels were chosen from the basic statistics and from visual inspection of the apparent continuity of very high grades within each domain, which indicated the need to limit their influence to approximately half the distance of the main search. Interpolation parameters are listed in Table 14-16 for the Triple R Deposit Mineral Resource domains.

Table 14-16: Block Estimate Search Strategy by Domain

Block Code	Estimation Method	Cap (% U_3O_8)	Cap (g/t Au)	High Yield Threshold (% U_3O_8)	Bearing (°)	Plunge (°)	Dip (°)	Major (m)	Semi (m)	Minor (m)	Min Sample	Max Sample
101	ID3	10	10	5	66.2	0	-90	50	50	10	2	7
102	ID3	10	10	--	66.2	0	-80	50	50	10	2	7
201	ID3	10	10	--	66.2	0	-70	50	50	10	2	7
202	ID3	10	10	--	66.2	0	-70	50	50	10	2	7
203	ID3	10	10	--	66.2	0	-70	50	50	10	2	7
204	ID3	10	10	--	66.2	0	-70	50	50	10	2	7
205	ID3	20	10	--	66.2	0	-70	50	50	10	2	7
206	ID3	10	10	--	66.2	0	-70	50	50	10	2	7
301	ID3	10	10	--	66.2	0	-90	50	10	50	2	7
302	ID3	10	10	--	66.2	0	-70	50	10	50	2	7
303	ID3	7	10	--	66.2	0	-70	50	50	10	2	7
304	ID3	7	10	--	66.2	0	-70	50	50	10	2	7
305	ID3	10	10	--	66.2	0	-70	50	50	10	2	7
306	ID3	10	10	--	66.2	0	-70	50	50	10	2	7
307	ID3	10	10	--	66.2	0	-90	25	25	5	2	7
308	ID3	10	10	--	66.2	0	-70	50	50	10	2	7
401	ID3	10	10	--	66.2	0	-90	50	50	10	2	7
501	ID3	10	10	--	66.2	0	-70	50	50	10	2	7

table continues…

Block Code	Estimation Method	Cap (% U_3O_8)	Cap (g/t Au)	High Yield Threshold (% U_3O_8)	Bearing (°)	Plunge (°)	Dip (°)	Major (m)	Semi (m)	Minor (m)	Min Sample	Max Sample
601	ID3	10	10	--	66.2	0	-90	50	10	50	2	7
602	ID3	10	10	--	66.2	0	-90	50	10	50	2	7
10111	ID3	55	15	20	66.2	0	-90	50	50	10	2	7
10112	ID3	55	15	20	66.2	0	-90	50	50	10	2	7
10113	ID3	55	15	20	66.2	0	-90	50	50	10	2	7
10114	ID3	55	15	20	66.2	0	-90	50	50	10	2	7
10115	ID3	55	15	20	66.2	0	-90	50	50	10	2	7
10211	ID3	55	15	20	66.2	0	-90	50	50	10	2	7
10212	ID3	55	15	20	66.2	0	-90	50	50	10	2	7
10213	ID3	55	15	20	66.2	0	-90	50	50	10	2	7
10214	ID3	55	15	20	66.2	0	-90	50	50	10	2	7
10215	ID3	55	15	20	66.2	0	-90	50	50	10	2	7
10311	ID3	55	15	20	66.2	0	-90	50	50	10	2	7
10312	ID3	55	15	20	66.2	0	-90	50	50	10	2	7
1041	ID3	55	15	20	66.2	0	-90	50	50	10	2	7
1061	ID3	55	15	20	66.2	0	-90	50	50	10	2	7
10711	ID3	55	15	20	66.2	0	-90	50	50	10	2	7
10712	ID3	55	15	20	66.2	0	-90	50	50	10	2	7
84001	ID3	15	10	10	66.2	0	-90	50	50	10	2	7

table continues…

Block Code	Estimation Method	Cap (% U_3O_8)	Cap (g/t Au)	High Yield Threshold (% U_3O_8)	Bearing (°)	Plunge (°)	Dip (°)	Major (m)	Semi (m)	Minor (m)	Min Sample	Max Sample
84002	ID3	20	10	--	66.2	0	-90	50	50	10	2	7
84003	ID3	20	10	--	66.2	0	-90	50	50	10	2	7
84005	ID3	20	10	--	66.2	0	-90	50	50	10	2	7
84006	ID3	7	10	--	66.2	0	-90	50	50	10	2	7
84007	ID3	20	10	--	66.2	0	-90	50	50	10	2	7
84008	ID3	20	10	--	66.2	0	-90	50	50	10	2	7
84009	ID3	15	10	--	66.2	0	-90	50	50	10	2	7
84010	ID3	20	10	--	66.2	0	-90	50	50	10	2	7
84011	ID3	20	10	--	66.2	0	-90	50	50	10	2	7
84012	ID3	20	10	--	66.2	0	-90	50	50	10	2	7
84013	ID3	20	10	--	66.2	0	-90	50	50	10	2	7
84014	ID3	20	10	--	66.2	0	-90	50	50	10	2	7
84015	ID3	20	10	--	66.2	0	-90	50	50	10	2	7
84016	ID3	20	10	--	66.2	0	-90	50	50	10	2	7
84017	ID3	20	10	--	66.2	0	-90	50	50	10	2	7
84018	ID3	20	10	--	66.2	0	-90	50	50	10	2	7
84019	ID3	20	10	--	66.2	0	-90	50	50	10	2	7
84020	ID3	20	10	--	66.2	0	-90	50	50	10	2	7
84021	ID3	20	10	--	66.2	0	-90	50	50	10	2	7

table continues…

Block Code	Estimation Method	Cap (% U_3O_8)	Cap (g/t Au)	High Yield Threshold (% U_3O_8)	Bearing (°)	Plunge (°)	Dip (°)	Major (m)	Semi (m)	Minor (m)	Min Sample	Max Sample
84022	ID3	20	10	--	66.2	0	-90	50	50	10	2	7
84023	ID3	20	10	--	66.2	0	-90	50	50	10	2	7
84024	ID3	20	10	--	66.2	0	-90	50	50	10	2	7
84025	ID3	20	10	--	66.2	0	-90	50	50	10	2	7
84026	ID3	20	10	--	66.2	0	-90	50	50	10	2	7
84027	ID3	7	10	--	66.2	0	-90	50	50	10	1	7
84028	ID3	20	10	--	66.2	0	-90	50	50	10	2	7
840101	ID3	35	10	--	66.2	0	-90	50	50	10	2	7
840102	ID3	35	10	--	66.2	0	-90	50	50	10	2	7
15151	ID3	10	10	--	66.2	0	-90	50	50	10	2	7
15152	ID3	10	10	--	66.2	0	-90	50	50	10	2	7
15153	ID3	10	10	--	66.2	0	-90	50	50	10	2	7
15154	ID3	10	10	--	66.2	0	-90	50	50	10	2	7
15155	ID3	7	10	--	66.2	0	-90	50	50	10	2	7
15156	ID3	10	10	--	66.2	0	-90	50	50	10	2	7
15157	ID3	10	10	--	66.2	0	-90	50	50	10	2	7
15158	ID3	10	10	--	66.2	0	-90	50	50	10	2	7
15159	ID3	10	10	--	66.2	0	-90	50	50	10	2	7
16201	ID3	20	10	--	66.2	0	-90	50	50	10	2	7

table continues…

Block Code	Estimation Method	Cap (% U_3O_8)	Cap (g/t Au)	High Yield Threshold (% U_3O_8)	Bearing (°)	Plunge (°)	Dip (°)	Major (m)	Semi (m)	Minor (m)	Min Sample	Max Sample
16202	ID3	20	10	10	66.2	0	-90	50	50	10	2	7
16203	ID3	20	10	--	66.2	0	-90	50	50	10	2	7
16204	ID3	20	10	--	66.2	0	-90	50	50	10	2	7
16205	ID3	20	10	--	66.2	0	-90	50	50	10	2	7

14.7 Block Model Validation

SLR validated the block model using the following methods:

- Swath plots of composite grades versus ID^3 and nearest neighbour (NN) grades in the X, Y, and Z (Figure 14-20 to Figure 14-22)
- Volumetric comparison of blocks versus wireframes
- Visual Inspection of block versus composite grades on plan, vertical, and long section
- Parallel secondary estimation using ID^3
- Statistical comparison of block grades with assay and composite grades

SLR found grade continuity to be reasonable and confirmed that the block grades were reasonably consistent with local drill hole composite grades.

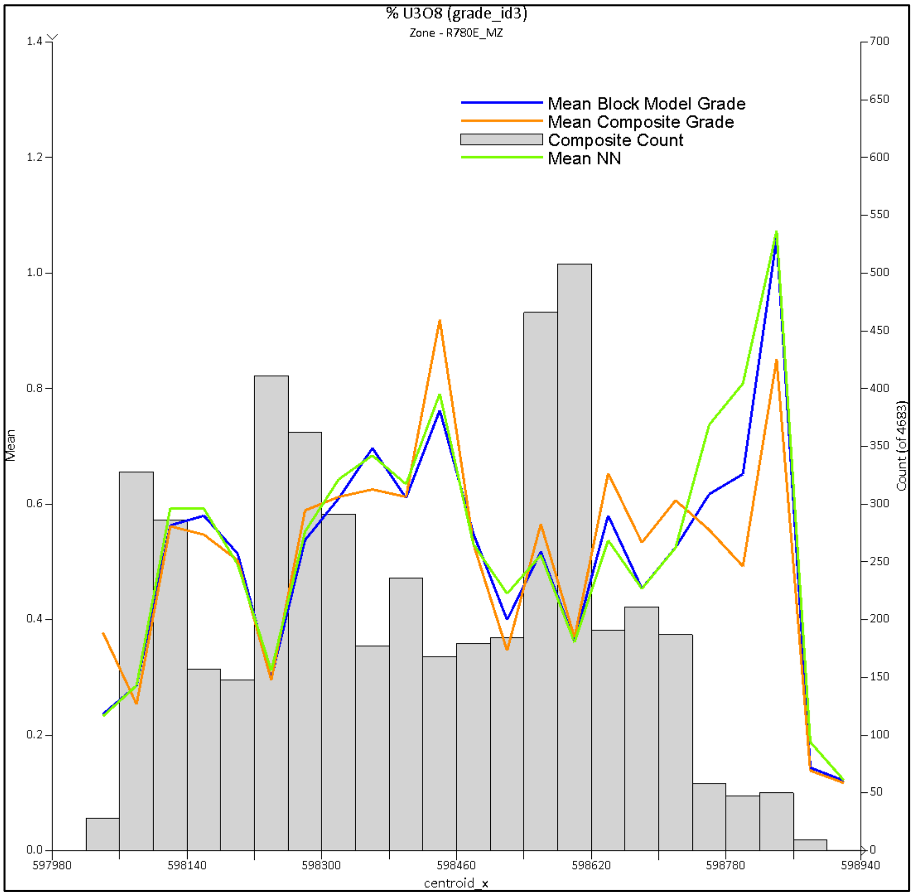


Figure 14-20: East-West (X) Swath Plot of R780E_MZ Deposit

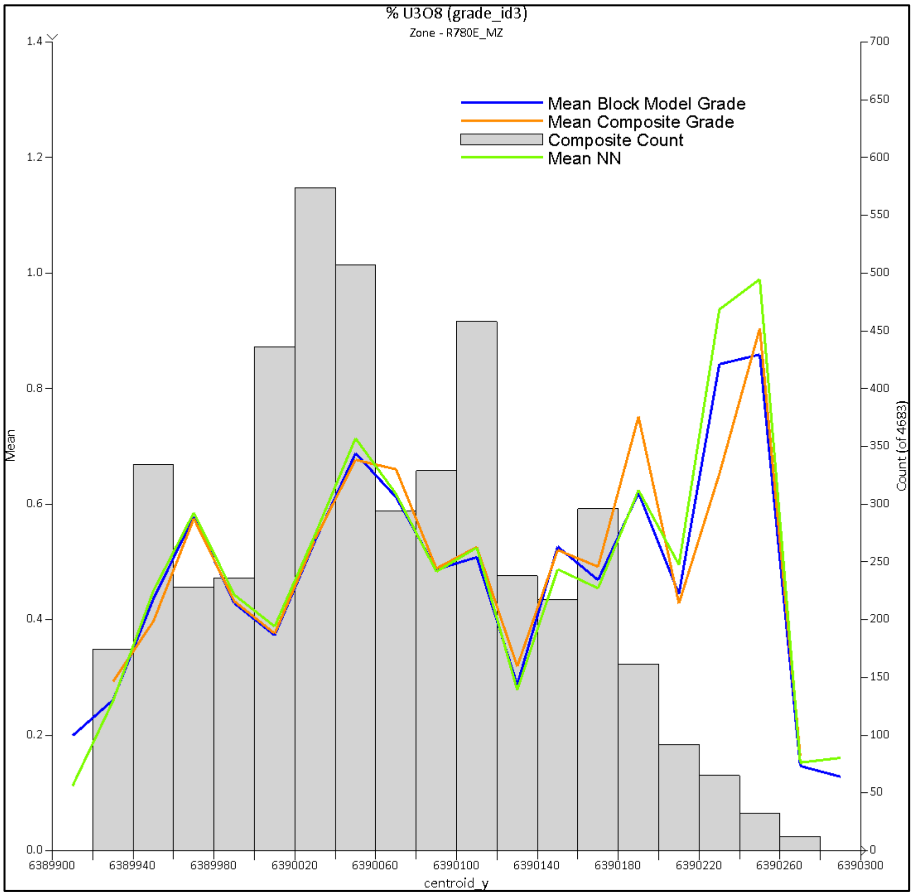


Figure 14-21: North-South (Y) Swath Plot of R780E_MZ Deposit

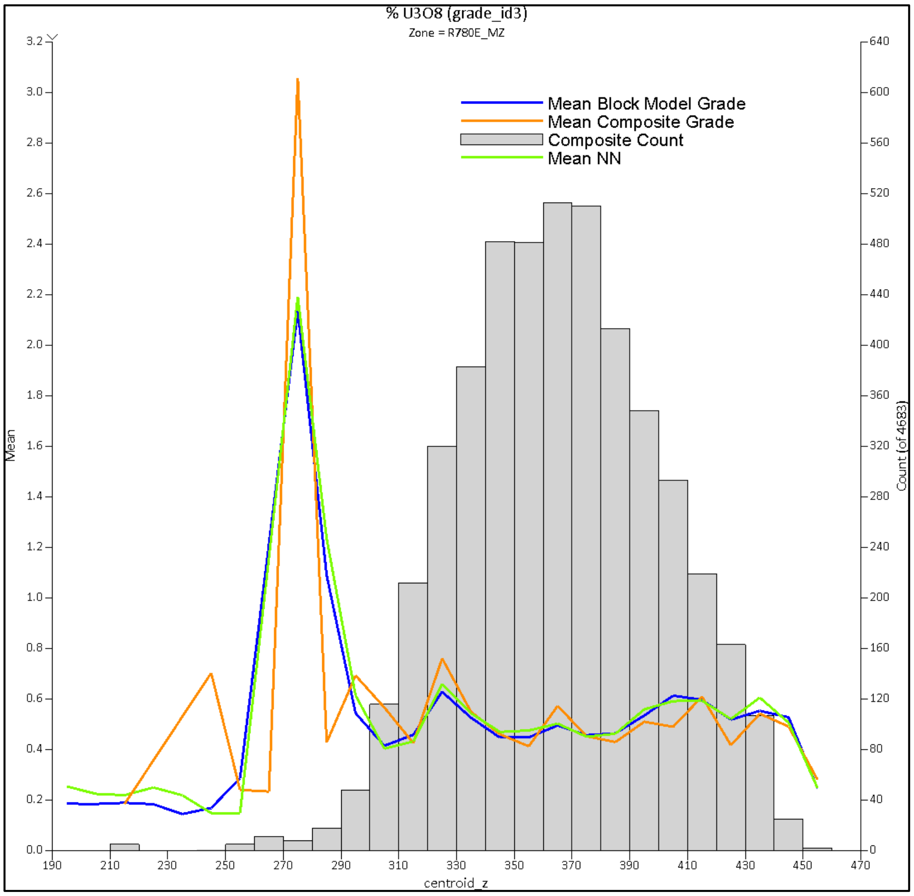


Figure 14-22: Vertical (Z) Swath Plot of R780E_MZ Deposit

14.7.1 Volume Comparison

The comparison of wireframe volumes to block volumes at a zero grade cut-off shows good agreement (Table 14-17).

Table 14-17: Volume Comparison

Zone	Wireframe Volume (m^3)	Block Model Volume (m^3)	% Difference
R780E_MZ	1,197,937	1,196,780	0.10%
R780E_HG	67,377	67,540	-0.24%
R780E_OTHER	61,231	61,020	0.35%
R00E	245,706	246,520	-0.33%
R1620E	4,248	4,310	-1.47%
R840W	166,064	169,660	-2.17%
R840W_HG	**2,333,018**	**2,337,120**	**-0.18%**

table continues…

Zone	Wireframe Volume (m^3)	Block Model Volume (m^3)	% Difference
1515W	1,197,937	1,196,780	0.10%

14.7.2 Visual Comparison

Block grades were visually compared with drill hole composites on cross-sections, longitudinal sections, and plan views. The block grades and composite grades correlate very well visually within the Triple R deposit (Figure 14-23 through Figure 14-26)

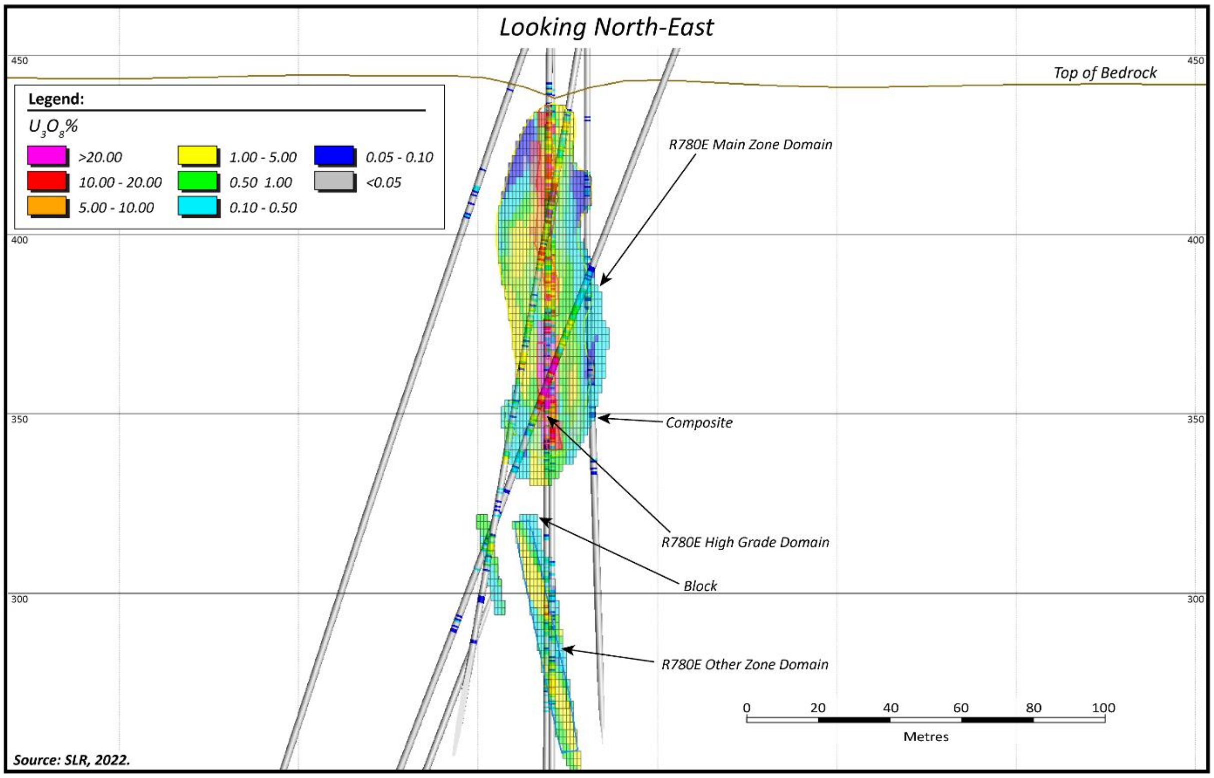


Figure 14-23: Vertical Section 655E

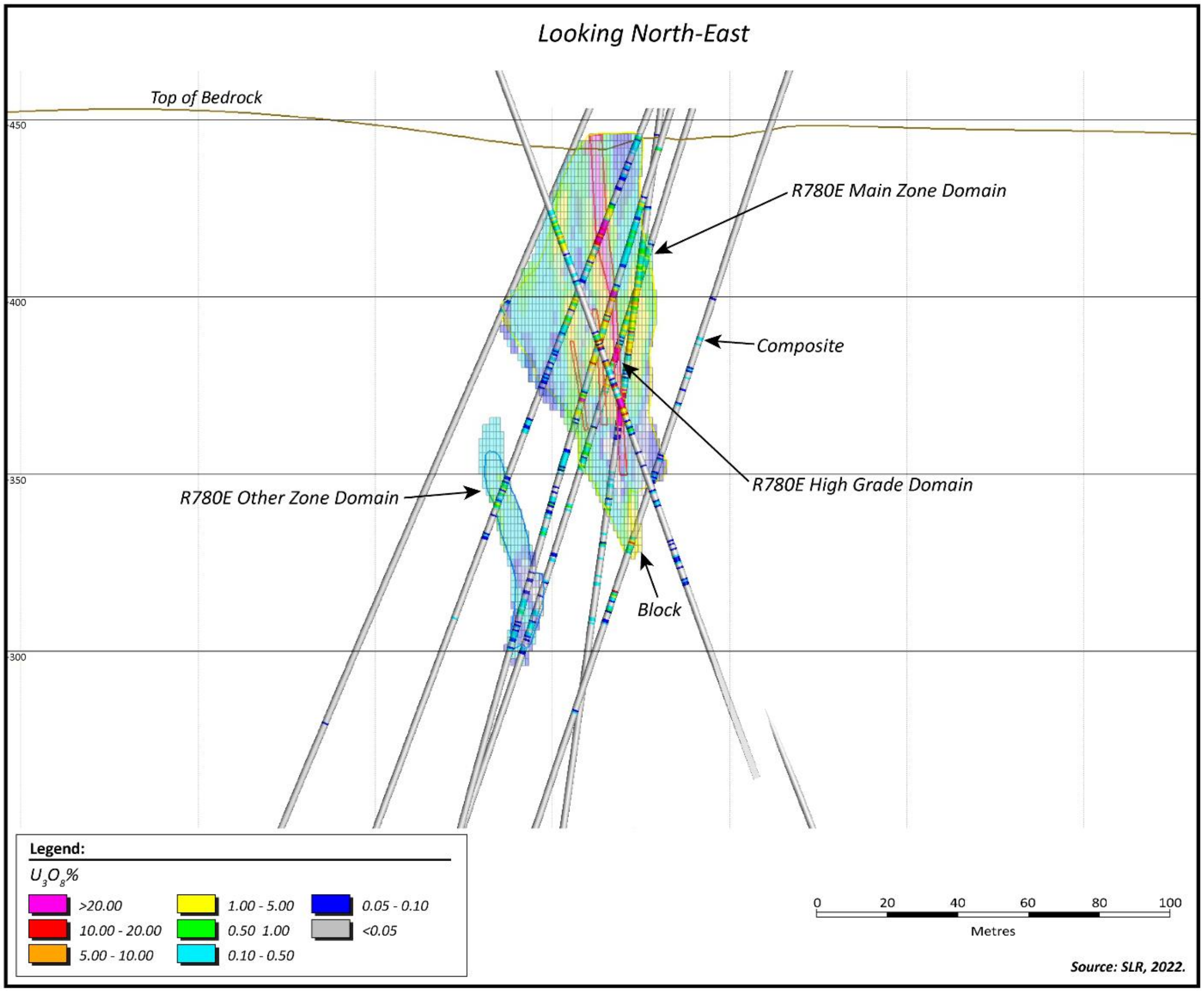


Figure 14-24: Vertical Section 510E

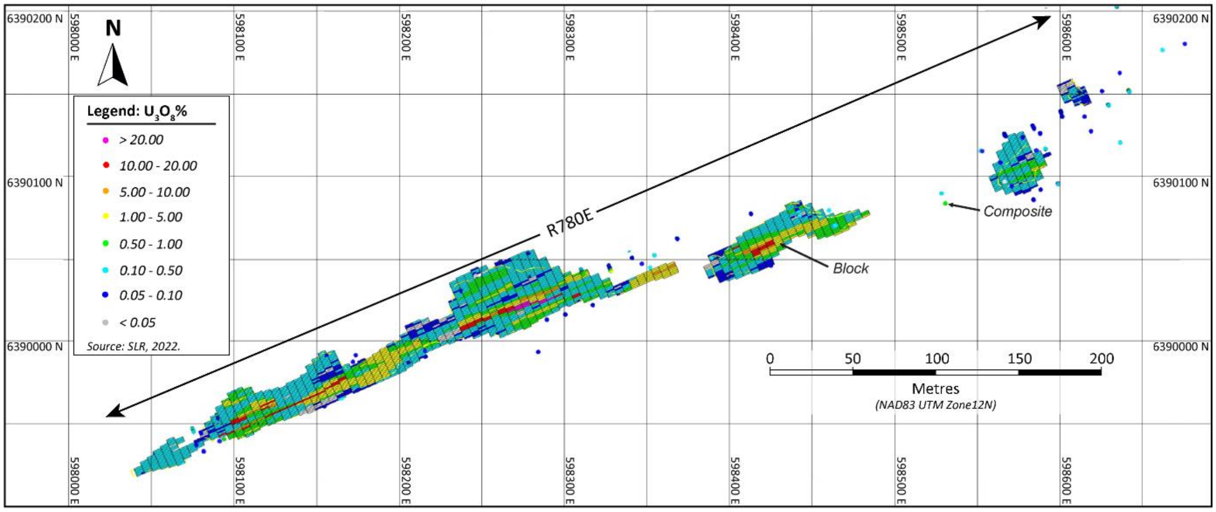


Figure 14-25: Level Plan 400Z

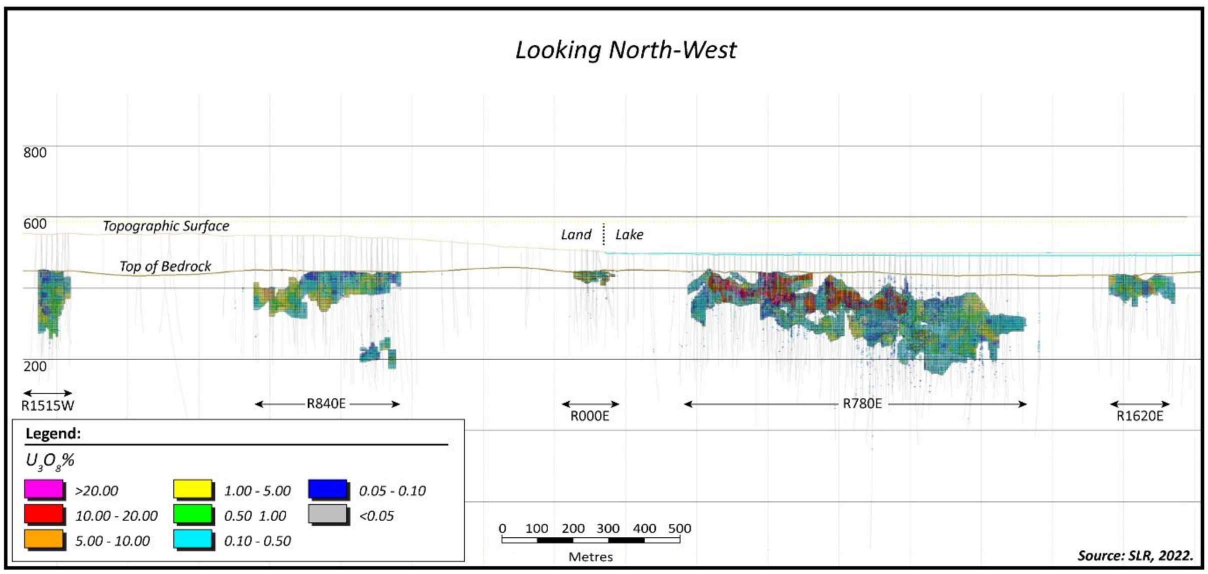


Figure 14-26: Longitudinal Section

14.7.3 Statistical Comparison

Statistics of the block grades are compared with statistics of composite grades in Table 14-18 for all blocks and composites within the Triple R deposit zones.

Table 14-18: Statistics of Block Grades vs. Composite Grades

Zone	R780E_MZ		R780E_HG		R780E_Other		R00E	
Descriptive Statistics	Comp	Block	Comp	Block	Comp	Block	Comp	Block
Number of Samples	4,717	119,704	386	6,764	1,424	52,528	223	6,102
Min	0.000	0.000	0.063	0.255	0.000	0.000	0.025	0.040
Max	10.00	6.78	53.18	46.61	16.68	9.68	10.00	8.97
Mean	0.51	0.51	15.17	16.52	0.60	0.52	1.28	1.35
Variance	0.91	0.32	116.60	62.15	1.57	0.47	4.71	3.22
SD	0.95	0.56	10.80	7.88	1.25	0.69	2.17	1.80
CV	1.87	1.11	0.71	0.48	2.09	1.32	1.69	1.33
Zone	**R1620E**		**R840W**		**R840W_HG**		**R1515W**	
Descriptive Statistics	**Comp**	**Block**	**Comp**	**Block**	**Comp**	**Block**	**Comp**	**Block**
Number of Samples	223	6,617	993	24,653	49	431	453	16,966
Min	0.000	0.002	0.000	0.000	0.01	0.08	0.003	0.020
Max	18.73	14.45	15.00	13.17	34.81	27.00	7.99	5.30
Mean	1.64	1.96	0.83	0.90	13.53	11.46	0.84	0.81
Variance	10.12	8.15	3.30	1.66	89.45	36.31	1.79	0.62
SD	3.18	2.86	1.82	1.29	9.46	6.03	1.34	0.79
CV	1.94	1.46	2.20	1.43	0.70	0.53	1.59	0.97

14.8 Mineral Resource Reporting Criteria

To fulfill the NI 43-101 requirement of “reasonable prospects for eventual economic extraction”, SLR estimated an underground mining COG using assumptions based on historical and known operating costs for mines operating in the Athabasca Basin, as well as previous studies completed.

Mineral Resources are reported at an underground COG of 0.25% U_3O_8. The COG is based on a long term price of US$50/lb U_3O_8 and cost estimates derived during the PFS with a metallurgical recovery of 95% and standard revenue and royalty assumptions for the province of Saskatchewan.

The assumptions on estimated operating costs used to calculate the COG are summarized in Section 21.

14.9 Classification

Classification of Mineral Resources as defined in CIM definition Standards for Mineral Resources and Mineral Reserves (CIM 2014) were followed for classification of Mineral Resources.

A Mineral Resource is defined as a concentration or occurrence of material of economic interest in or on the Earth's crust in such form, grade or quality, and quantity that there are reasonable prospects for economic extraction. A Mineral Resource is a reasonable estimate of mineralization, considering relevant factors such as COG, likely mining dimensions, location, or continuity, that with the assumed and justifiable technical and economic conditions, is likely to, in whole or in part, become economically extractable. It is not merely an inventory of all mineralization drilled or sampled. A Mineral Reserve is defined as the "economically mineable part of a Measured and/or Indicated Mineral Resource" demonstrated by studies at Pre-Feasibility or Feasibility level as appropriate. Mineral Reserves are classified into Proven and Probable categories.

Based on this definition of Mineral Resources, the Mineral Resources estimated in this Technical Report have been classified according to the definitions below based on geology, grade continuity, and drill hole spacing.

Measured Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of conclusive geological evidence and sampling. The level of geological certainty associated with a measured Mineral Resource is sufficient to allow a QP to apply modifying factors, as defined in this section, in sufficient detail to support detailed mine planning and final evaluation of the economic viability of the deposit. Because a measured Mineral Resource has a higher level of confidence than the level of confidence of either an indicated Mineral Resource or an inferred Mineral Resource, a measured Mineral Resource may be converted to a proven mineral reserve or to a probable mineral reserve.

Indicated Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of adequate geological evidence and sampling. The level of geological certainty associated with an indicated Mineral Resource is sufficient to allow a QP to apply modifying factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Because an indicated Mineral Resource has a lower level of confidence than the level of confidence of a measured Mineral Resource, an indicated Mineral Resource may only be converted to a probable mineral reserve.

Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. The level of geological uncertainty associated with an inferred Mineral Resource is too high to apply relevant technical and economic factors likely to influence the prospects of economic extraction in a manner useful for evaluation of economic viability. Because an inferred Mineral Resource has the lowest level of geological confidence of all Mineral Resources, which prevents the application of the modifying factors in a manner useful for evaluation of economic viability, an inferred Mineral Resource may not be considered when assessing the economic viability of a mining project and may not be converted to a mineral reserve.

SLR considered the following factors that can affect the uncertainty associated with the class of Mineral Resources:

- Reliability of sampling data:
  - Drilling, sampling, sample preparation, and assay procedures follow industry standards.
  - Data verification and validation work confirm drill hole sample databases are reliable.
  - No significant biases were observed in the QA/QC analysis results.

- Confidence in interpretation and modelling of geological and estimation domains:
  - Mineralization domains are interpreted manually in cross-sections and refined in longitudinal sections by an experienced resource geologist.
  - There is good agreement between the drill holes and mineralization wireframe shapes.
- Confidence in block grade estimates:
  - Indicated block grades correlate well with composite data, statistically and spatially, and locally and globally.
  - Mineral Resources were classified as Indicated or Inferred based on drill hole spacing and the apparent continuity of mineralization.

Indicated Resource

Indicated Mineral Resources are defined as blocks within the domains where drill sections are spaced 15 m apart along strike, vertical holes are spaced approximately 10 m to 15 m along each section, grade continuity indicated by two or more drill holes that meet the minimum cut-off criteria (0.05% U_3O_8), and a distance to NN less than 15 m. Angle holes are spaced from 15 m to 45 m apart, averaging 30 m, along the strike direction (Figure 14-27).

Inferred Resource

Inferred Mineral Resources are defined as all remaining blocks within the domains that are not classified as Indicated.

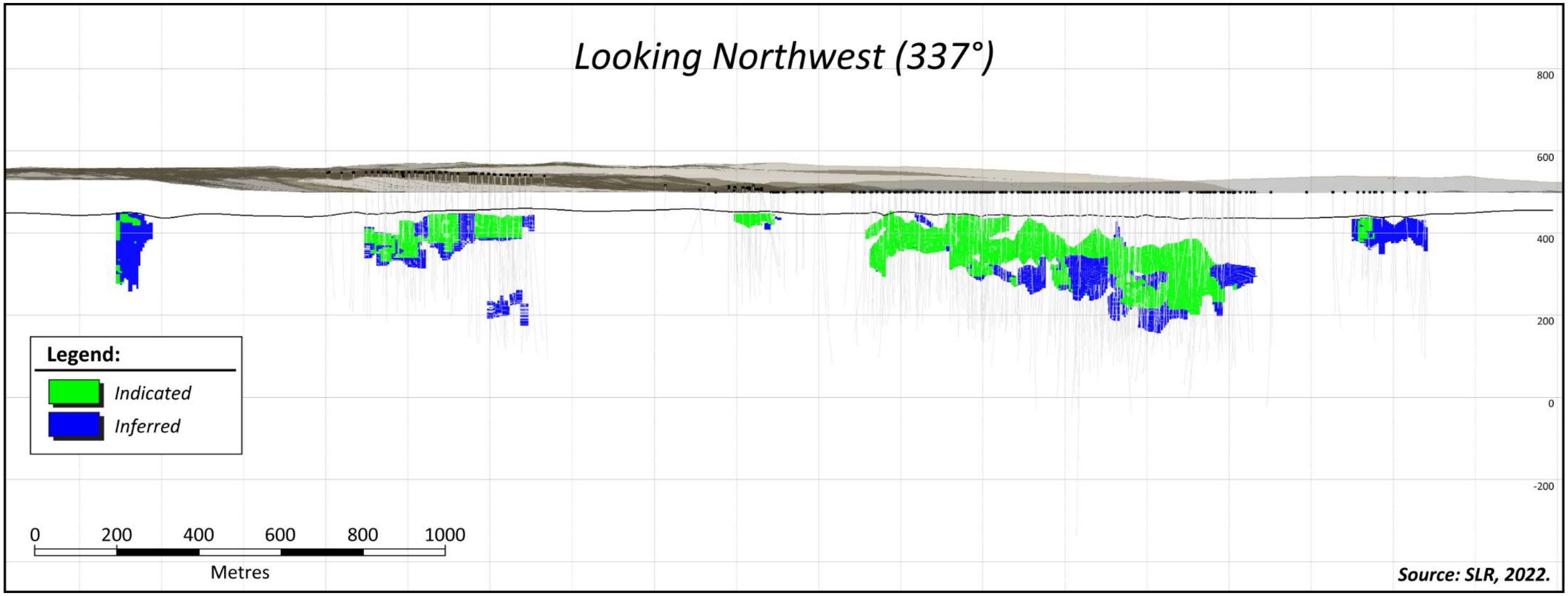


Figure 14-27: 3D View of the Mineral Resource Classification

14.10 Mineral Resource Reporting

At a COG of 0.25% U_3O_8 for Mineral Resources potentially mineable by underground methods, Indicated Mineral Resources total 2.69 Mt at an average grade of 1.94% U_3O_8 for a total of 114.9 Mlb U_3O_8. Inferred Mineral Resources total 0.64 Mt at an average grade of 1.10% U_3O_8 for a total of 15.4 Mlb U_3O_8. Estimated grades are based on chemical assays only. Gold grades were also estimated and average 0.61 g/t for the Indicated Mineral Resources and 0.44 g/t for the Inferred Mineral Resources. Mineral Resources are inclusive of Mineral Reserves.

The zones are those areas traditionally referred to by FCU in press releases and on its website and are generally defined by differences in location with respect to local grid easting. The R780E_HG domain consists of several lenses within the R780E_MZ and, when combined, the two zones account for approximately 68% of the total resources at Triple R. Table 14-19 reports Mineral Resources summarized by zone.

Table 14-19: Mineral Resource Statement by Zone – May 17, 2022

Zone	Tonnage	Metal Grade		Contained Metal	
	(000 t)	(% U_3O_8)	(g/t Au)	(Mlb U_3O_8)	(000 oz Au)
Indicated					
R780E_HG	162	16.91	2.73	60.4	14.2
R780E_MZ	1,578	0.79	0.48	27.5	24.1
R780E_OTHER	429	0.95	0.62	9.0	8.6
R00E	98	1.50	0.15	3.2	0.5
R1620E	42	1.98	0.19	1.9	0.3
R840W	303	1.35	0.36	9.0	3.6
R840W_HG	9	11.32	2.38	2.2	0.7
R1515W	67	1.15	0.38	1.7	0.8
Indicated Total	**2,688**	**1.94**	**0.61**	**114.9**	**52.7**
Inferred					
R780E_HG	0	11.80	5.73	0.1	0.1
R780E_MZ	16	0.33	0.29	0.1	0.2
R780E_OTHER	254	0.60	0.46	3.4	3.8
R00E	9	3.83	0.79	0.7	0.2
R1620E	59	3.55	0.48	4.6	0.9
R840W	63	1.10	0.37	1.5	0.7
R1515W	234	0.96	0.42	5.0	3.1
Inferred Total	**635**	**1.10**	**0.44**	**15.4**	**9.0**

Notes:

1. CIM (2014) definitions were followed for Mineral Resources.
2. Mineral Resources are reported at a COG of 0.25% U_3O_8, based on a long term price of US$50/lb U_3O_8, an exchange rate of C$1.00/US$0.75, and cost estimates derived during the PFS with a metallurgical recovery of 95%.
3. A minimum mining width of 1 m was applied to the resource domain wireframe.

4. Mineral Resources are inclusive of Mineral Reserves.
5. Numbers may not add due to rounding.

14.11 Grade Tonnage Sensitivity

Table 14-20 and Figure 14-28 show the sensitivity of the Triple R block model to various COG. SLR notes that, although there is some sensitivity of average grade and t to COG, the contained metal is less sensitive.

Table 14-20: Grade Tonnage Sensitivity Indicated Mineral Resource

COG	Tonnage	Metal Grade		Contained Metal	
	(000 t)	(% U_3O_8)	(g/t Au)	(Mlb U_3O_8)	(000 oz Au)
0.05	4,132	1.31	0.43	119.6	57.4
0.1	3,771	1.43	0.47	119.0	56.6
0.15	3,371	1.59	0.51	117.9	55.5
0.2	3,005	1.76	0.56	116.5	54.1
0.25	**2,688**	**1.94**	**0.61**	**114.9**	**52.7**
0.3	2,419	2.12	0.66	113.3	51.2
0.35	2,193	2.31	0.70	111.7	49.7
0.4	2,012	2.48	0.75	110.2	48.4
0.45	1,863	2.65	0.79	108.8	47.2
0.5	1,733	2.81	0.83	107.4	46.1
0.6	1,504	3.16	0.90	104.7	43.7
0.7	1,327	3.49	0.97	102.1	41.5
0.8	1,169	3.86	1.05	99.5	39.3
0.9	1,028	4.28	1.12	96.9	37.2
1	911	4.71	1.20	94.4	35.2

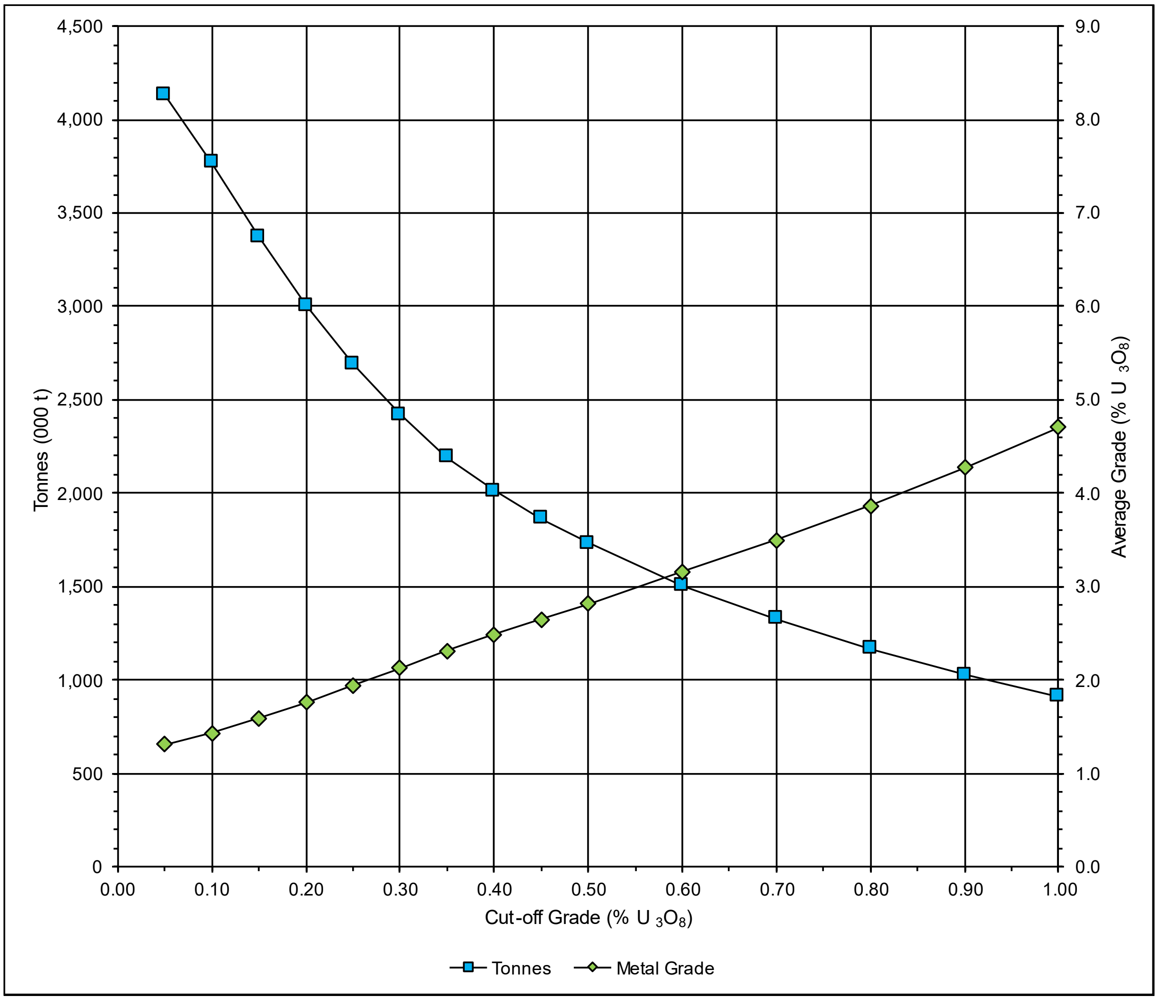


Figure 14-28: Indicated Mineral Resource Tonnes and Grade at Various COGs

14.12 Comparison to Previous Estimate

Table 14-21 compares the May 17, 2022, Mineral Resource estimate with the September 19, 2019, Mineral Resource estimate. Indicated Mineral Resources have increased by 21%, or approximately 472,000 t, with a minor decrease in grade from 2.10% U_3O_8 to 1.94% U_3O_8. Inferred Mineral Resources have decreased 48%, or approximately 586,000 t, with a decrease in grade from 1.22% U_3O_8 to 1.10% U_3O_8.

Highlights of the Mineral Resource update as of May 17, 2022, include:

- Total Indicated t have increased by approximately 21.3% with an associated increase of approximately 12.3% in contained U_3O_8 and 7.5% decrease in % U_3O_8 grade.
- The increase in Indicated Mineral Resources is due to conversion of all the 301, 302, and 303 domain Inferred resources to Indicated along with reclassification of Indicated material in domains 101, 204, 401, 84001, 84003, 84006, 84008, 84009, 84010, 84011, and 84012.
- Total Inferred t have decreased by approximately 48% with an associated decrease in grade of 9.9% U_3O_8 resulting in a 53.2% decrease in contained U_3O_8.

- Total contained gold remains relatively unchanged with a small decrease of 1,000 gold ounces.

Inferred mineralization contained within the Halo domain (901) has been excluded from the May 17, 2022, Mineral Resources estimate as this zone does not meet CIM (2014) definition criteria for RPEEE based on an underground mining only scenario. The reported resource for the Halo zone included in the September 19, 2019, PFS was a holdover from the initial work completed when the Resource estimate was both an open pit and underground scenario and should have been removed in the underground only scenario. In addition, the Halo domain did not extend southwest to cover the R840W target area. Removal of the Halo zone from the overall Mineral Resource is negligible as the estimated contained metal is less than 1.0% (1.24 Mlb U_3O_8) of the total Mineral Resource.

Table 14-21: Comparison to Previous Resource Estimate

Estimate	Tonnage	Metal Grade		Contained Metal	
	(000 t)	(% U_3O_8)	(g/t Au)	(Mlb U_3O_8)	(000 oz Au)
May 17, 2022, Estimate					
Indicated	2,688	1.94	0.61	114.9	52.7
Inferred	635	1.10	0.44	15.4	9.0
September 19, 2019, Estimate					
Indicated	2,216	2.10	0.61	102.4	43.1
Inferred	1,221	1.22	0.50	32.8	19.6
Difference					
Indicated	472	-0.16	0.00	12.6	9.6
Inferred	-586	-0.12	-0.06	-17.5	-10.6
% Difference					
Indicated	21.3%	-7.5%	0.8%	12.3%	22.3%
Inferred	-48.0%	-9.9%	-11.8%	-53.2%	-54.1%

15.0 MINERAL RESERVES ESTIMATE

15.1 Mineral Reserves Statement

The Mineral Reserves for the PLS Property are based on the Mineral Resources with an effective date of May 17, 2022. Detailed mine designs have been generated and modifying factors have been applied. The Mineral Reserve includes a nominal amount of material above the mineralized waste cut-off of 0.03% U_3O_8 and below the incremental COG of 0.19% U_3O_8 that has been included based on the requirement to access certain mining areas or manage geotechnical conditions in a production area.

Estimates of mineralization and other technical information included herein have been prepared in accordance with NI 43-101 – Standard of Disclosure for Mineral Projects.

Table 15-1 summarizes the estimated Mineral Reserves.

Table 15-1: Mineral Reserve Statement, FCU – PLS Property

Category	Tonnes (000 t)	Grade (% U_3O_8)	Contained Metal (Mlb U_3O_8)
Probable	--	--	--
R780E Zone	2,630	1.46	84.8
R00E Zone	56	1.24	1.5
R840W Zone	322	1.04	7.4
Total Probable	**3,007**	**1.41**	**93.7**

Notes:
1. CIM Definition Standards (2014) were followed for classification of Mineral Reserves.
2. The Mineral Reserves are reported with an effective date of January 17, 2023.
3. Mineral Reserves were estimated using a long-term metal price of US$65 per lb of U_3O_8 and a US$/C$ exchange rate of 0.75 (C$1.00 = US$0.75)
4. Underground Mineral Reserves were estimated by creating stope shapes using Datamine's MSO. MSO was run at an initial COG of 0.20% U_3O_8. For longhole stoping, a minimum mining width of 4 m (including hanging wall and footwall dilution) and stope height of 20 m was used. Following MSO, the mineable shapes were further sub-divided in Deswik to produce a maximum width of 12 m (including hanging wall and footwall dilution). D/F mining is based on 5 m wide by 5 m high development shapes located in the crown pillar areas of the orebodies.
5. Mining recovery of 95% was applied to all stopes while all development mining assumes 100% extraction.
6. The density varies based on block model values. An estimated waste density of 2.42 t/m^3 was used for areas outside of the block model boundary.
7. By-product credits were not included in the estimation of Mineral Reserve as the mill is not designed to recover gold (Au).
8. Numbers may not add due to rounding.

Mineral Resource to Reserve conversion for Indicated material was 84.2% within the R840W, R00E, and R780E zones. Mining losses fall primarily within the following two categories:

- Sterilization of material located within the designed crown pillar.
- Resource blocks being excluded from designed stopes due to low grades or inadequate continuity with adjacent mineralization.

Two other zones on the PLS property, R1515W and R1620E, have not been considered for inclusion in Mineral Reserves at this time.

Mining Plus is not aware of any mining, metallurgical, infrastructure, permitting, or other relevant factors that could materially affect the Mineral Reserves estimate.

15.2 Underground Design Criteria

The primary mining method envisioned for the PLS project is longhole sublevel retreat stoping in the longitudinal direction. Stopes were generated using Datamine MSO software using the spatial parameters presented in Table 15-2 below.

Table 15-2: MSO Input Parameters – Stopes

Parameter	Value	Unit
Stope Height[1]	20	m
Stope Length[2]	10	m
Minimum Stope Width[3]	4	m
Maximum Stope Width[3]	12	m
Minimum Stope Dip	75	°
Optimization Orientation	-23.8	°
Initial COG	0.2	%

Notes:
1. Stope heights are based on floor-to-floor elevations.
2. Stope lengths are assumed to be in the longitudinal direction along strike.
3. Min/max stope widths include 1 m of dilution from both the hanging wall and footwall of the stope for a total of 2 m. The MSO stope shapes generated that were >12 m width were post-processed into adjacent stope shapes that conformed with geotechnical guidance.

Due to geotechnical constraints and proximity to the bedrock contact, the crown pillar zones above 420 m elevation of the deposits are planned to be mined via D/F. These are currently designed as 5 m wide by 5 m high development drifts utilizing cemented hydraulic backfill to extract multiple passes where the orebody thickness exceeds 5 m. D/F shapes were generated using Datamine MSO software using the spatial parameters presented in Table 15-3 below.

Table 15-3: MSO Input Parameters – D/F

Parameter	Value	Unit
Drift Height	5	m
Drift Width	5	m
Pillar	0	m
Dip	90	°
Maximum Waste	30	%

table continues…

Parameter	Value	Unit
Initial COG	0.2	%

15.3 Modifying Factors

Modifying factors are the key difference between a Mineral Resource and Mineral Reserve. These factors include dilution (internal and external) and mining recovery or extraction. Reserve planning is done utilizing these factors, the spatial constraints presented in Section 15.2, and a 0.25% breakeven U_3O_8 COG in an iterative process. The following sections will describe the modifying factors in further detail.

15.3.1 Internal Dilution

Internal dilution is the inclusion of material below COG in a final stope shape. Planned or internal dilution is commonly higher in longhole mining methods due to the inherent lack of selectivity as well as variations in geology (grades, orientation, thickness). It is often necessary to include internal dilution to build coherent stope shapes that follow the constraints required for efficient extraction. Any inferred material within a reserve stope shape has also been assigned a grade of 0% during the Mineral Reserve calculation.

15.3.2 External Dilution

External or unplanned dilution is additional material coming from the hanging wall, footwall, or sidewalls, outside of the planned stope boundary. It is generally driven by local ground or geotechnical conditions or operational factors such as drill and blast accuracy. Depending on the stope location and sequencing, it is also possible to have backfill dilution from adjacent stopes and floors.

A hybrid approach to applying dilution to longhole stoping has been used for the PLS project. While a minimum longhole mining width of 2 m is practically achievable, the MSO minimum mining width parameter was forced to 4 m to account for 2 m of combined hanging wall and footwall overbreak. A 10 m maximum mining width has also been assumed, but when accounting for 2 m of overbreak, a realistic parameter of 12 m has been used. Further dilution was then added during the scheduling phase to account for backfill dilution and spatial constraints. The dilution applied within the schedule is assumed to be at zero grade, whereas when dilution has been applied through MSO, the grade is based on block model values.

There are two parameters being used to determine the additional external dilution:

- Backfill wall exposure
  - Stopes have been manually assigned whether they will have end wall or sidewall backfill exposure to account for differences in backfill dilution. Sidewall exposure occurs in areas of the orebody where widths are more than 12 m, and multiple stopes must be mined in the transverse direction to fully extract the ore. In effect, this means once the initial or primary stope is mined, all subsequent or secondary stopes in sequence will have side walls consisting of cemented fill.
- Proximity to hanging wall and footwall contacts
  - In areas where stopes are mined within 5 m to 7 m of the Mineralized Shear Zone (MSZ) contact, it is expected that additional dilution will occur, as stopes will naturally want to break to the geological unit contacts. The risk is expected to be partially mitigated using cable bolting where required.

Table 15-4 below shows the different external dilution factors being applied according to the situation.

Table 15-4: Stope External Dilution Factors

External Dilution Factor		Number of Walls within 5 - 7 m of MSZ Contact		
		None	One	Two
Backfill Wall Exposure	End Wall/Floor	5%	8%	10%
	Sidewall	10%	12%	15%

Development, including D/F mining, has been assigned an overbreak/dilution factor of 5%, which has a 0% grade assigned to the waste dilution.

An analysis of the planned underground stope shapes has been performed and shows a net dilution of 24% for the PLS project. Mining Plus recommends that a detailed dilution study be conducted that further defines the expected performance of the various ore zones given the large range in ore strengths as part of future studies.

15.3.3 Mining Recovery

Mining recovery is a factor applied to account for potential ore losses during the mining process. The value is driven largely by the selected mining method and equipment. In longhole methods, mining recovery is largely dependent on drill and blast and mucking practices. Inaccurate drilling and poor blasting outcomes can result in material being left behind, while remote mucking performance is driven by line of sight, floor quality, and operator experience. In development mining, the selective nature of the process allows for near full extraction of the resource. This is due to the ability to visually inspect the headings allowing potential issues to be resolved.

For the PLS project, a longhole mining recovery of 95% has been applied, while development mining assumes 100%.

15.4 Cut-Off Grade

The values used in the calculation were based off benchmarking, previous studies on PLS, and any data updated and/or calculated during the FS.

Stopes were generated in MSO at a 0.20% U_3O_8 cut-off as a first pass. From this initial stope set, stopes over 0.25% U_3O_8 were filtered and selected for inclusion in the design based on their location and continuity with adjacent mineralization. The stopes were then evaluated for additional modifying factors such as external dilution inclusion. In cases where diluted stope grades were below 0.25% but above 0.15% U_3O_8, a determination was made for inclusion in the Mineral Reserve based on the practicality of mining (development requirements, mill feed blend requirements, adjacent stopes, geotechnical pillar considerations, etc.). After completion of the project economic model, the initial COGs were validated to ensure they aligned with the project economic results.

Table 15-5 shows the COG calculations.

Table 15-5: COG Estimate

Parameter	Unit	Value	Notes
Annual Mine and Mill Production	000 t	350	--
Daily Throughput	t/d	1,000	--
Uranium Grade	% U_3O_8	1.41%	--
Uranium Recovery	%	96.99%	--
Recovered Uranium	Mlb	10.6	--
Uranium Price	US$/lb	65	--
Exchange Rate	C$/US$	0.75	--
Uranium Price (Canadian)	C$/lb U_3O_8	86.67	--
SK Royalty Payments	%	7.25%	--
Net Uranium Price Realized	C$/lb U_3O_8	80.38	--
Revenue			
Net Smelter Return	C$ 000	849,796	1
Value per t	C$/t ore	2,428	--
Value per % U_3O_8	C$/% U_3O_8	1,718	--
Operating Costs			
Unit Underground Mining	C$/t proc	153.0	2
Unit Processing	C$/t proc	162.8	2
Unit G&A	C$/t proc	78.1	2
Mining	C$ 000	53,550	--
Processing	C$ 000	56,973	--
G&A	C$ 000	27,342	--
Total Operating Costs	**C$ 000**	**137,865**	--
Mining Incremental at 60% Variable	C$/t proc	91.8	3
COGs			
Operating Costs	% U_3O_8	0.23%	4
Incremental (Var. Mining, Proc. G&A)	% U_3O_8	0.19%	5
Portal Discard (Proc., G&A)	% U_3O_8	0.14%	6

Notes:

1. Assume that buyers pay for transportation to refinery and the Uranium is 100% payable, and there are no penalties.
2. Cost assumptions are based on comparable projects, previous PFS study work (RPA, Wood, and Clifton, 2019b), studies, and updated FS work.
3. Mining Plus assumes that incremental mining costs are 40% fixed annually and 60% are variable based on tonnage.
4. This COG considers the recovered metal needed to cover the total operating costs.

5. This COG considers the recovered metal needed to cover the variable mining costs, processing, and G&A costs.
6. Surface COG (often referred to as the Portal Discard) considers that the material has already been mined and brought to surface, and the metal content must be enough to cover processing and G&A.

15.5 Summary of Material Classification

The limit for what is considered benign waste is 0.03% U_3O_8. This limit is considered a rule of thumb and is accepted from a regulatory perspective. Table 15-6 summarizes how the quantities of ore, waste, and specialized waste rock were derived.

Table 15-6: Summary of Ore and Waste Classification by Underground Mining Type

Resource Block Grade – %U_3O_8	Category	Within Stopes (Grade Measured Per Stope)	Within Development (Grade Measured Per Round)
<0.03%	Waste	Blended into diluted stope grade and sent to appropriate stockpile	Sent to Waste Pile, not processed
0.03% to 0.15%	Special Waste	Blended into diluted stope grade and sent to appropriate stockpile	Sent to Special Waste Pile, not processed, available for mill grade blending or backfill if required
0.15% to 0.25%	Low Grade (LG) ore	Blended into diluted stope grade and sent to appropriate mill stockpile	Sent to LG Stockpile for blending and processing
0.25% to 4.00%	Medium Grade (MG) Ore	Blended into diluted stope grade and sent to appropriate mill stockpile	Sent to MG Stockpile for processing
>4.00%	High Grade (HG) Ore	Blended into diluted stope grade and sent to appropriate mill stockpile	Sent to HG Stockpile for processing

16.0 MINING METHODS

The PLS Project hosts the Triple R deposit, a structurally controlled east-west trending sub-vertical high-grade uranium deposit. The deposit is overlain by 50 m to 100 m of sandy overburden, with the high-grade mineralization located near the bedrock-overburden contact. Bedrock in the project area consists of discontinuous Cretaceous Mannville Group and Devonian La Loche Formation sedimentary bedrock, which overlay Archean basement rocks comprised of gneisses.

The PFS completed in 2019 considered an underground-only mine plan which was designed for extraction of the R780E and R00E orebodies. The mine access was primarily through a decline ramp, and a FAS and EAS were designed to provide both ventilation and secondary egress.

As part of the FS, the mine plan has been updated to include the R840W orebody to the southwest of the R780E and R00E zones. The mine access is planned using a decline from surface and an FAS and EAS. The FS has changed the decline position to the south of the R840W orebody and the capital lateral development has been relocated to the hanging wall of the deposits to remove the potential to interfere with mineralization located in the footwall.

16.1 Overview

The underground mining method will be primarily longhole retreat stoping using longitudinal methods based on the current geotechnical analysis and orientation of mineralization. Where the mineralization thickness allows, mining will progress across the orebody from the hanging wall drive (HWD) to footwall drive (FWD). In the longitudinal areas of mining, the lenses will be mined from the southwest to the northeast in a bottom-up sequence, except for one area in the later part of the mine life which will be mined underneath a sill pillar. Mining is planned at a nominal rate of 1,000 t/d ore.

The mine will be accessed using a decline originating to the south of the R840W deposit. The decline will include a box cut into the overburden and a portal face collared into the overburden headwall (HW). The first stage of the decline will be developed through overburden geological units for approximately 350 m on a gradient of -15%. The decline is planned to be excavated using a tunnel shield method with segmented concrete liner which will require the length of the decline to be dewatered to below the tunnel horizon. Following this, the decline will transition through a section of weak sedimentary bedrock for 65 m before it reaches competent bedrock.

The ventilation system will be a push-pull system with one primary FAS and one primary EAS. The underground ventilation network will use a series of dedicated fresh air drives, internal raises, and return air drives to distribute air to development and production areas. The primary ventilation network has been designed to keep all production zones under negative pressure to prevent air from recirculating through the network. In production headings, a negative pressure auxiliary ventilation system will be used to ensure personnel remain in fresh air and contaminants are removed at the face and directed to the return air system.

The crown pillar areas will be partially recovered using bulk artificial ground freezing techniques and an overhand D/F mining method. The artificial ground freezing reticulation is planned to be installed from an offset underground drift below the crown pillar areas in both the R00E and R780E zones. The refrigeration plants will be constructed on surface and will pump refrigerated brine solution underground via the FAS to the crown pillar areas.

16.2 Geotechnical Analysis and Design

Geotechnical analysis and design were carried out by BGC (BGC Engineering Inc. 2022e). The data set for the rock mechanics assessments includes the results of geotechnical and hydrogeological site investigations by BGC and geological exploration activities by FCU, supplemented by publicly available literature, as necessary, to address data gaps and/or low confidence data in the understanding of the project in-situ stress setting and major geologic structures.

16.2.1 Geotechnical Assessment

The areas within the proposed underground mine footprint were divided into structural domains based on location within the project area, see Table 16-1 and Table 16-2. The rock mass is dominated by foliation, which dips sub-vertically to the southeast, with occasional rotation to the south. Sub-horizontal exfoliation joints, moderately dipping cross-joints, and shallow to moderately dipping fault sets are also present.

Table 16-1: Intact Strength Properties Summary (BGC Engineering Inc. 2022e)

Structural Domain Type	Description	Unit Weight (g/cm³)	Poisson's Ratio	Young's Modulus (GPa)	M_i constant	Median RMR_{76}	Median Q'	Median RQD (%)	UCS (MPa)
Sedimentary Bedrock Unit	Devonian (Elk Point Group) siltstones and sandstones comprising either the Contact Rapids Formation or La Loche Formation underlay Till 1, and are primarily encountered in the southeastern margin of the property.	2.44	0.17	19.8	7.2	42	3.33	27	18
Shallow Basement Bedrock Unit	Consists of all basement bedrock within 10 m of the upper basement bedrock contact. This rock has been affected by paleo weathering that has been overprinted by alteration, and is weaker than its fresh condition.	2.63	0.17	19.8	7.8	55	12.4	82	31
Footwall Unit	Consists of all basement bedrock within the footwall of the MSZ.	2.70	0.19	56.6	8.2	65	23.3	95	88
Decline Unit	Consists of all basement bedrock within the Decline area.	2.69	0.21	42.5	8.3	68	25	100	49
Hanging wall QFBG-G Unit	Consists of QFBG-GN in the hanging wall of the MSZ.	2.63	0.21	48.8	8.1	64	12.5	99	45
Hanging wall SIL-S-QF Unit	Consists of silicified quartz-feldspar-biotite-garnet in the hanging wall of the MSZ.	2.68	0.30	52.6	8.1	58	10.5	93	45
MSZ Unit	Comprises potentially mineralized rock as indicated in the FCU geological model and with scintillator values greater than 300 cps.	2.71	0.17	18.8	8.1	58	12.5	89	43
Resource Unit	Consists of indicated and inferred mineralized rock as indicated in the FCU geological model.	2.46	0.17	19.8	8.8	53	5.93	74	17
Fault Zones	Comprises fault-disturbed rock.	2.46	0.17	19.8	8.8	31	1.24	0	17

Table 16-2: Soil Geotechnical Units Material Properties (BGC Engineering Inc. 2022e)

Soil Geotechnical Unit	Description	Cohesion (kPa)	Friction Angle (°)[2]	Elastic Modulus (GPa)	Poisson's Ratio	Bulk Unit Weight (kN/m³)	Material Model
Glaciofluvial Unit	Glacial deposits between 12 and 76 m thick, comprised primarily of fine to medium sand with some silt and some clay.	0	32	0.6	0.33	19	Mohr-Coulomb
Till 1[1]	Till 1 underlays Till 2. It was interpreted as a basal till predominantly composed of disturbed Cretaceous-derived siltstones and claystones which are interbedded with glaciofluvial outwash sands. The Cretaceous-derived material was described as very stiff to hard silty clay to clayey silt, with some sand to sandy composition.	0	25-29[1]	0.95	0.33	19	Mohr-Coulomb
Till 2	Comprised of gap graded sand and gravel, with up to 20% cobbles. Silt content was measured to vary form approximately 10% to 35%.	0	29	0.95	0.33	19	Mohr-Coulomb
In Situ Mudstone	--	0	12	2.0	0.33	19	Mohr-Coulomb

Notes:

1. Till 1 above the mine footprint is predominantly low-plastic. Material represented with a bi-linear failure envelope of 29° up to s' of 750 kPa, reducing to 25° at 1,140 kPa.
2. Glaciofluvial, Till 2, and Till 1 units are assigned peak strength values. In-situ mudstone is assigned residual strength values.

16.2.1.1 Underground Design Acceptance Criteria

The design acceptance criteria for the geotechnical assessments for the underground mine design is detailed as follows:

- The kinematic stability (e.g., stability of wedges formed by intersecting discontinuities) of all excavations is judged to be acceptable under static conditions when a minimum factor of safety (FOS) of 1.1 (production excavations), 1.3 (development excavations), or 1.5 (mine access excavations: FAS, EAS and decline) is demonstrated by underground wedge stability analyses with the recommended ground support designs in place.
- The overall stability of all development, mine access, and cut and fill excavations is judged to be acceptable under static conditions when two-dimensional finite element analyses indicate that excavation-induced yielding is constrained within the ground support elements and the ground support elements do not fail.
- The finalized stope dimensions and sequencing are judged to be acceptable when:
  - Empirical analyses using the Stability Graph method indicates the stopes are “stable without support” (for unsupported stopes) or “stable with support” (for stopes supported with long support).
  - The maximum acceptable average equivalent linear overbreak slough (ELOS) is 1.0 m and 1.5 m (per wall) for the unsupported design cases.
  - Two-dimensional or 3D finite element analyses indicate that excavation-induced yielding is constrained to within 3 m to 5 m of the excavation boundary (estimated limit of long support) and the ground support elements (if included in the model) do not fail.
- The overall stability of the crown pillar is judged to be acceptable under static conditions when:
  - Two-dimensional finite element analyses indicate that excavation-induced yielding is constrained to the lower 50% of the crown pillar.
  - Yielding does not extend through the frozen crown pillar to the unfrozen sedimentary bedrock or unfrozen soil above the crown pillar.
  - A minimum serviceable life of two years is indicated through analytical (rigid beam) and empirical analyses.

The acceptance criteria are based on industry standard practice with the following assumptions:

- Material shear strengths are assumed to be reasonable based on the information available.
- The geotechnical dataset is sufficient to develop an understanding of the potential failure mechanisms.
- Underground excavations will be managed during operations using observational techniques and instrumentation where necessary.
- Workers are not permitted to work under unsupported ground.
- Stopes will be mined in a longitudinal retreat sequence and will be backfilled as soon as possible.
- The crown pillar acceptance criteria are based on modification of industry standard practice due to the design conservatism necessary to minimize strain in the crown pillar. Failure or large strain of the crown pillar may induce failure, which would allow uncontrolled water inflows through the rock mass into the underground.

16.2.1.2 Hydrogeological Input to Rock Mechanics

A 3D numerical groundwater model and dewatering assessment of the Decline and underground mine was completed by BGC for the Project (BGC Engineering Inc. 2022b). The results of the groundwater model were considered in the rock mechanics assessments completed.

Two design cases for pore pressure conditions were considered. The “unmitigated” case assumed passive draw-down of the phreatic surface as the mine life advances with no probe and grout cover. The “mitigated” case assumed passive draw-down of the phreatic surface as the mine life advances with probe and grout cover mitigation implemented throughout the mine life during development of all excavations. Since there will be a tendency for grout permeability anisotropy (a specific discontinuity set will be a preferential pathway for the grout), and grouting will only occur if threshold seepage rates occur in the probe holes, rock mass strength gain due to grouting was not considered in the rock mechanics assessments.

16.2.1.3 Empirical Stope Stability Analysis and ELOS Estimate

Preliminary recommendations for stope dimensions and ground support to assist with the development of stoping design criteria have been developed using the empirical “Stability Graph” method (Hutchinson and Diederichs 1996) (Nickson 1992). The method is used to estimate acceptable mining dimensions for the proposed stope back and walls using the hydraulic radii (HR) and modified stability number (N’) calculated as follows:

$$N' = Q' \ x \ A \ x \ B \ x \ C$$

Where:

Q’ is the modified rock mass classification parameter

A is a measure of the ratio of intact rock strength to induced stress

B is a measure of the relative orientation of dominant structure with respect to the excavation surface

C is a measure of the influence of gravity on the stability of the face being considered.

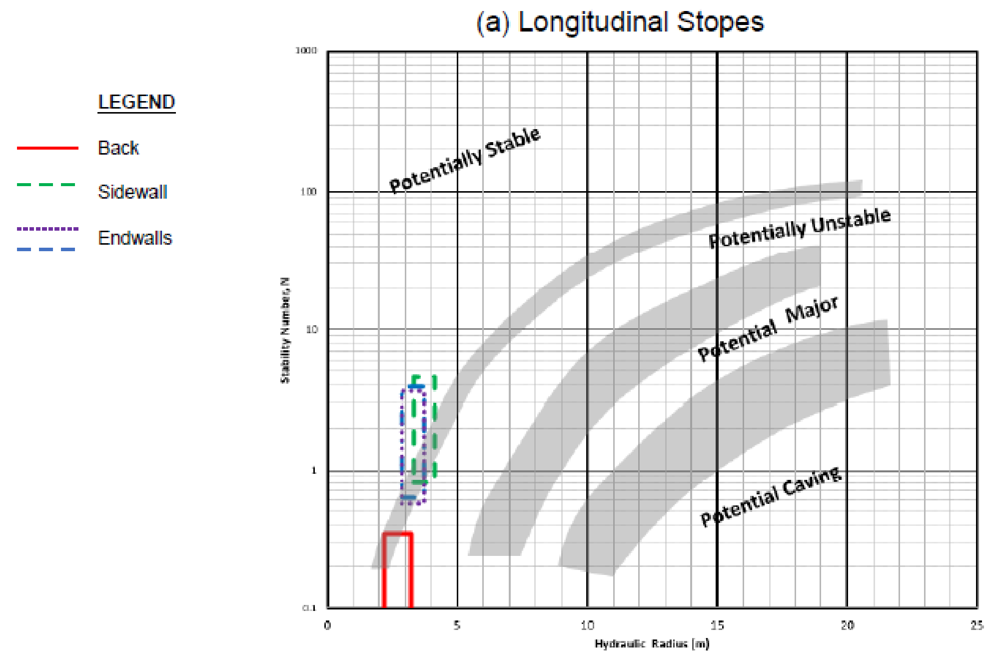
The results of the stability number calculation (N’) and the hydraulic radii (area/perimeter) for the assessed stope surface are plotted on the stability curve to estimate the achievable unsupported and supported stope dimensions. The N’ values were calculated according to the following:

- The base-case designs were estimated using the mean values for rock mass quality (Q’) and intact strength of the Resource geotechnical unit within the R840W, R00E, and R780E zones.
- The mean longitudinal stope hanging wall dip direction and dip is parallel to foliation and the general trend of the MSZ (approximate dip direction 150° and variation in dip from 75° to 90°).
- The “A” parameter was calculated using the design unconfined compressive rock strength (UCS) for each geotechnical unit, with an in-situ stress field at a representative stope depth of 200 m below ground surface and a horizontal to vertical stress ratio (K) value of 1.2. Induced stresses were estimated according to the methods of Mawdesley et al. (2001).
- The “B” parameter was calculated for the dominant design discontinuity set for each stope surface (back and walls).
- The “C” parameter was calculated based on the orientation of the design discontinuity sets, which indicated that sliding failure mechanisms will govern stability in the walls, and that gravity fall will govern stability in the back.

- The three discontinuity sets controlling stability in the stopes are a sub-horizontal to horizontal set (primarily controlling stability in the stope backs), joints and faults along foliation (dipping steeply to the southeast), and conjugate sets to foliation (dipping moderately to the northeast and southwest).

The results of the analyses for unsupported stopes were presented for the 25th percentile to 50th percentile Q' values in Figure 16-1. Maximum recommended HRs for unsupported and supported stope backs were also provided in a summary table on Figure 16-1 and were based on the 50th percentile Q' values. Maximum recommended HRs for supported stope endwalls and sidewalls were not provided due to the limited access for long support installation and the inability to support the entire wall(s).

The estimated average ELOS (Clark and Pakalnis 1997) is 1.0 m to 1.5 m (per wall) for the unsupported design cases. The N' and HR values for the screening-level designs were near the lower limits of the empirical ELOS chart. Based on work by Pakalnis on ELOS in weak rock, if the HR is less than 3.5 m, the expected ELOS is less than 1.0 m (Clark 1998).



DESIGN CASE	STOPE FACE	Q'	A	B	C	N'	MAX. HR (unsupp.)	MAX. HR (supported)
Longitudinal	Back	1 – 6	0.1 – 0.13	0.38	1	0.04 – 0.29	2.6	3.2
	Sidewalls	1 – 6	0.28 – 0.34	0.35	6.8	0.8 – 3.96	4.1	N/A
	Endwalls	1 – 6	0.23 – 0.28	0.34	8	0.62 – 4.54	3.75	N/A

NOTES:

1. THIS FIGURE SHOULD BE READ IN CONJUNCTION WITH BGC'S REPORT TITLED "PATTERSON LAKE SOUTH FEASIBILITY STUDY, UNDERGROUND ROCK MECHANICS ASSESSMENTS" AND DATED NOV. 2022
2. Q' VALUES FOR THE RESOURCE GEOTECHNICAL UNIT.
3. DESIGN DISCONTINUITY SETS ARE SUBHORIZONTAL-HORIZONTAL, STEEPLY DIPPING PARALLEL TO FOLIATION, AND MODERATELY DIPPING CONJUGATE TO FOLIATION.
4. ESTIMATED STOPING DEPTH OF 200 MBGS. STOPES ARE ASSUMED TO BE DEPRESSURIZED. K (H:V) OF 1.2. VERTICAL STOPES END WALLS AND SIDE WALLS.
5. INDUCED STRESSED ARE CALCULATED ACCORDING TO THE METHODS OF MAWDESLEY AT AL. (2001).
6. FOR THE SUPPORTED CASE IN THE BACK, LONG SUPPORT IS ASSUMED TO BE PROVIDED USING CONNECTABLE SWELLEX (SWX-PM24C) (ATLAS COPCO) OR SIMILAR, WHICH ARE LIMITED TO A BOLT LENGTH OF APPROXIMATELY 4.0 M. 4.0 M IS THE ABSOLUTE MINIMUM BOLT LENGTH INDICATED FOR SUPPORTED SPANS IN THE BACK.

PREPARED BY: AH	FIGURE TITLE: Stability Graph Results (Unsupported) Stope Dip 90°			
CHECKED BY: CJS	CLIENT: FISSION URANIUM CORP.			
APPROVED BY: CJS	SCALE: NTS	PROJECT NO: 1665-009	FIGURE NO: 7	

0 5 10 mm in ANSI B sized paper

BGC ENGINEERING INC.

Figure 16-1: Stability Graph Results for Unsupported Longhole Designs by BGC (BGC Engineering Inc. 2022e)

16.2.1.4 Numerical Stope Stability Analyses

Numerical modelling was completed by BGC to provide an indication of the potential for mining-induced stress changes to affect excavation and pillar stability (BGC Engineering Inc., 2022e). A summary of the mine planning considerations to reduce the potential for instability is provided below:

- It is recommended that the stopes within the top level (400 m to 420 m elevation) be mined only after completion of ground freezing.
- A minimum stope dip of 75° is recommended.
- Sequencing of the mining advance/retreat should attempt to maximize rib pillar thickness to minimize yielding in the rib pillars. When required, pillar widths equal to or greater than three stope widths (radially) are recommended. The modelling also indicates that when mining across multiple levels, rib pillars less than four stope widths exhibit signs of overstress and instability.
- It is recommended that no single lift (20 m) remnant sill pillars be left after mining. When required, a 20 m sill pillar should be remotely monitored for stress-induced failure and instability. Stopes with angled walls (i.e., 75° or less) will likely cause more yielding in the sill pillar than stopes with vertical walls (i.e., 90°). It is recommended that no more than two stopes be mined across the strike of the resource immediately below a 20 m sill pillar as the risk of sill pillar yielding increases with additional excavated stopes. It is also recommended that stopes beneath the planned sill pillars be mined later in the mine life after pore pressures have dissipated.
- Underhand mining may result in stope instabilities across multiple mining levels, with the potential for work under yielded ground, and therefore it is recommended that all mining occur via overhand methods wherever feasible. When underhand mining is required, high strength backfill, and optimization of stope sequencing may reduce the extent of yielding in the stope walls and backs.

16.2.2 Backfill Strength Recommendation

MineFill Services completed a study to review the different backfill alternatives for the FCU Triple R deposit based on available materials (Minefill Services 2022). The initial selection included cemented rockfill (CRF) (with processed mine waste rock), cemented aggregate fill (CAF), CHF batched with locally available sands, and cemented paste fill.

The use of CRF in underground stopes was found to be limited by the availability of waste rock over the mine life. The study shows a deficit of almost 1 Mt without the supplemental sand, and about 0.5 Mt if local sands are incorporated in the CRF mix. The use of paste fill was rejected once the paste strength test results came available. The tests showed the cemented paste performed worse than the hydraulic fills at similar binder contents.

The final selection adopted for the study was the use of CRF mixed with the locally available sand at a 75/25 ratio until the waste rock runs out in year 8. From there the stope backfill will revert to CAF utilizing locally available aggregate and sand extracted from an area north of the planned TMF. The priority for the use of CHF will be to fill the D/F excavations as it will be easier to achieve a tight fill. The D/F excavations are concentrated along the top of the deposit within the crown pillar zones. The aggregate for the CRF and CAF will be produced locally in a modular crushing and screening plant consisting of a jaw crusher and a single cone in a closed circuit. The plant will produce two products: a coarse aggregate comprising minus 75 mm to plus 10 mm, and a fine aggregate comprising minus 10 mm.

Based on the test work completed, the cemented backfill material is shown to gain the required strength to perform under a variety of mining conditions and exposures. The target UCS for the backfill mixes is:

- 360 kPa based on 20 m vertical free-standing height in longhole stopes

- 250 kPa for mobile equipment operating on the floor of overhand D/F stopes
- 1.7 MPa based on undercut spans of 4 m to 10 m for long hole stope sills

Table 16-3 outlines the estimated cement content to achieve the required strength for each mix type.

Table 16-3 Backfill Cement Content by Mix Type and Mining Method

Mix Type	D/F	Longhole Stope Wall	Longhole Stope Sill
CRF	3.0%	3.5%	5.5%
CAF	3.0%	2.5%	5.5%
CHF	8.0%	Not Required	Not Required

The indicative curing times for each backfill type is summarized below:

- CRF/CAF – 20 m vertical free standing exposure in longhole stopes – seven days
- CHF – 5 m vertical sidewall exposure in D/F excavation – seven days

The use of underhand mining to recover stopes below planned sills is limited and most extraction is planned to be bottom up. There is one level in R780E that will be mined below a planned sill pillar, however, this does not occur until late in the mine life.

MineFill recommends the following future work:

- Additional testwork on the hydraulic fills to optimize the designs. The additional testing should include rheology, additional percolation tests, and uniaxial compression tests over a range of solids contents.

16.2.3 Ground Support

Finalized ground support recommendations for each excavation type are provided in Table 16-4 (BGC Engineering Inc. 2022e). The recommended design is the most conservative suggested between the empirical and kinematic analyses conducted, with minor conservative adjustments made if needed for operational efficiency (e.g., to recommend the same bolt type, length, and/or spacing as other excavations).

As shown in Table 16-4, reinforced ribs of shotcrete (RRS) are recommended in all 4-way intersections in the Resource unit, and in areas of the R840W crown pillar where the crown pillar thickness (basement bedrock + sedimentary bedrock) is less than 20 m. For mine planning purposes, the RRS are estimated to be 45 cm thick (including additional shotcrete and the rebar stick-up) around the full drift profile (not including the floor) and comprised of a double layer of 16 mm to 20 mm diameter rebars (six rebars first layer, two rebars second layer) with 45 cm of shotcrete (including the shotcrete applied as primary support prior to RRS installation). The recommended RRS spacing is 2.5 m (centre-to-centre). The RRS installed in intersections should extend a minimum of 3 m into adjacent excavations. If the 45 cm thickness of the RRS system is prohibitive to clearance requirements, a multi-stage support system consisting of 5 cm of shotcrete, Swellex (2.4 m long on 1.2 m spacing) with welded mesh, a second 5 cm layer of shotcrete, long support of self-drilling connectable anchors (5.0 m long on 2.0 m spacing), and strapping of the intersection pillars is recommended. Short rounds lengths (1 m to 2 m) and perimeter cushion holes in each blast layout should be considered. Spiling may also be required.

Long support in stope backs and stope walls may be required to control unplanned dilution and to maintain stability in the shoulders of the topsills of adjacent stopes. The kinematic analyses also indicate the potential for unstable

(FOS < 1.2) wedges to form in stope back and walls if discontinuities are persistent on a scale of 10 m to 15 m or greater. For costing purposes, BGC recommended 10 m self-drilling connectable anchors at 2 m spacing fanned from the topsill and bottomsill ore drives for up to 20% of stopes. This estimate is based on the relative density of faults in the R840W, R00E, and R780E structural domains (whereby faults are considered most likely to be persistent on a stope-scale).

BGC considered the potential for corrosion of the ground support elements. Water quality data from water samples collected during basement bedrock pumping tests (BGC Engineering Inc. 2020; 2022a; 2022d) indicate water from within the proposed mine footprint (in both mineralized and unmineralized rock) has a pH ranging from 7.11 to 8.23 and chloride concentrations of 99.8 mg/L to 5,590 mg/L. For cost estimating purposes, BGC recommended that ground support elements for development excavations receive corrosion protection coating due to the longer required service life of those excavations.

Due to adverse orientation of some of the discontinuity sets in the rock mass, there remains a residual risk for rock fall despite the installation of the ground support summarized in this Technical Report. The risk for that rock fall increases where the discontinuities are persistent (greater than half of the excavation span) and closely spaced (less than half of the excavation span). The existing geotechnical database is insufficient to predict where these more persistent and/or closely spaced discontinuities may occur, therefore thorough geotechnical inspections, geotechnical mapping, and routine scaling of the workings throughout the development cycle should be carried out to help mitigate the residual risk.

Table 16-4: Underground Support Design Recommendations (BGC Engineering Inc. 2022e)

Development or Production Excavation	Excavation Purpose	Support Type[1,2,4]	Assumed Span (m)	Assumed Unit	Bolt Spacing (staggered) (m)	Bolt Length (m)	Shotcrete Thickness (cm)[9] or Welded Mesh	Notes
Development	Production Crosscuts	Swellex	4.0	Resource, MSZ	1.2	2.4	Mesh	--
	Decline, Main Development Headings, Internal Ramps	Resin-grouted rebar	5.0	Footwall (FW), Headwall (HW)	1.5	2.4	Mesh	Not inclusive of any fault zones, which will require special case.
	EAS[8]	Resin-grouted rebar	5.0	See note 8	2.0	3.0	Mesh	Not inclusive of any fault zones, which will require special case. For cost estimating purposes, recommended consideration of a 10 m wide fault zone with 3 m long #7 rebar on 1.5 m staggered spacing and 15 cm of FRS or MRS.
	FAS[8]	Resin-grouted rebar	6.8	See note 8	2.0	3.0	Mesh	Not inclusive of any fault zones, which will require special case. For cost estimating purposes, recommended consideration of a 5 m wide fault zone with 3 m long #7 rebar on 1.5 m staggered spacing and 15 cm of FRS or MRS.
	Intersections (3-ways)	Connectable Swellex	8.5	FW, HW	1.8	3.6	Mesh	Plus primary support of 2.4 m Swellex on 1.2 m spacing.
	Intersections (4-ways)	Connectable Swellex	8.5	FW, HW	2.0	5.0	Mesh	Plus primary support of 2.4 m Swellex on 1.2 m spacing.
	Underground Infrastructure (pump stations, etc.)	Resin-grouted rebar	6.0	FW	1.5	2.4	Mesh	--
	Underground Infrastructure (pump stations, etc.)	Self-drilling connectable anchors[4]	6.0	HW	1.5	3.6	Mesh	Plus primary support of 2.4 m Swellex on 1.2 m spacing.
	Underground Infrastructure (pump stations, etc.)	Self-drilling connectable anchors[4]	8.0	FW	2.4	5.0	5.0	Plus primary support of 2.4 m Swellex on 1.2 m spacing.
	Underground Infrastructure (pump stations, etc.)	Self-drilling connectable anchors[4]	8.0	HW	2.4	5.0	5.0	Plus primary support of 2.4 m Swellex on 1.2 m spacing.
	Shaft Stations	Resin-grouted rebar	10.0	FW, HW	2.5	5.0	10	Plus primary support of 2.4 m Swellex on 1.2 m spacing.
	Internal Raises	Resin-grouted rebar	3.0	FW, HW, MSZ	1.2	2.1	Mesh	Internal raises do not require ground support if no person-access.
Production	Ore Drives	Swellex	4.0	Resource	1.2	2.4	7.5	--
	D/F	Swellex	5.0	Resource	1.2	2.4	15.0	--
	Intersections[7]	Connectable Swellex or Swellex + RRS (see notes column)	7.5	Resource	1.5	3.6	--	Plus RRS at 2.5 m spacing, extending min. three wide into adjacent excavations. Thickness of the RRS estimated at a minimum of 45 cm including additional shotcrete and the rebar stick-up. Strapping of the pillars is also recommended.
	Stopes[3]	Self-drilling connectable anchors[4]	10.0	Resource	2.0	10	--	For primary stopes adjacent to remnant pillars, hanging wall rock, or footwall rock. Long support fanned from topsill and bottomsill, plus primary support installed as part of ore drive.

table continues…

Development or Production Excavation	Excavation Purpose	Support Type[1,2,4]	Assumed Span (m)	Assumed Unit	Bolt Spacing (staggered) (m)	Bolt Length (m)	Shotcrete Thickness (cm)[9] or Welded Mesh	Notes
R840W Crown Pillar	D/F[7]	Connectable Swellex or Swellex + RRS (see notes column)	5.0	Resource	2.0	3.6	--	Where crown pillar (basement + sedimentary) thickness is less than 20 m. Plus RRS at 2.5 m spacing extending the full drift profile (not including the floor). Additional thickness of RRS estimated at a minimum of 45 cm including additional shotcrete and rebar stick-up. RRS in addition to standard ground support for D/F Excavations.

Notes:

1. Standard Swellex unless otherwise noted.
2. Resin-grouted rebar (#7) is recommended for long-term excavations due to corrosion potential of Swellex.
3. For cost estimating purposes, only primary stope walls adjacent to waste are considered for long support. Overbreak of walls in ore is considered acceptable contingent on the stability of the adjacent topsill and bottomsill for neighboring stope production.
4. MAI SDA self-drilling anchors (connectable) R32 S or similar are recommended for stope long support.
5. Ground support recommendations are provided for cost estimating purposes only and are not to be used for construction.
6. Internal raises are assumed to be used for person-access between levels.
7. Recommendations for RRS are based on a 10 m span in the Resource unit, and corresponds to "RRS-II" support category in NGI (2015). If thickness of the RRS is prohibitive to clearance requirements, recommend primary support of 5 cm shotcrete followed by 2.4 m Swellex on 1.2 m spacing with welded mesh, 5 cm shotcrete, and 5 m long self-drilling connectable anchors on 2.4 m spacing.
8. Recommendations for the EAS and FAS are based solely on the data from PLS21-VS-002 and PLS21-VS-004, respectively. The design intent of the ground support designs presented in this Technical Report for the FAS and EAS are for LOM in shaft intervals that do not receive a concrete liner.
9. Mesh-reinforced shotcrete (MRS) or fibre-reinforced shotcrete (FRS).

16.2.3.1 Shaft Geotechnical Characterization and Ground Support Recommendations

The two main shafts in the mining footprint—the EAS and the FAS—were analyzed separately to the rest of the rock mass due to their importance as the main access/egress points for the mine and their vertical extent. Two geotechnical drillholes, PLS22-VS-002 and PLS22-VS-004, were drilled through the proposed EAS and FAS locations, respectively, and that data was assessed for rock mass quality, rock mass variability, presence of major structures, and discontinuity orientations, to design the rock support for the shafts.

The resulting ground support recommendations provided by BGC (2022e) for the two vertical shafts for LOM in shaft intervals that do not receive a concrete liner are:

- EAS
  - Stability along the EAS shaft will be governed by the formation of unstable wedges and the poorer-quality zone between the bedrock contact and at elevation 385 m. The recommended ground support consists of 3 m long resin-grouted #7 rebar on 2 m staggered spacing with 5 cm of FRS or MRS.
  - A 2 m wide fault zone is interpreted between 358.4 m and 360.4 m elevation. This fault zone may require additional ground support. For cost estimating purposes, BGC recommends a 10 m long zone of increased ground support between 355 m and 365 m elevation, consisting of 3 m long resin-grouted #7 rebar on 1.5 m staggered spacing with 15 cm of MRS. Increased inflows should also be expected through this zone.
- FAS
  - Stability along the FAS shaft will be governed by the formation of unstable wedges and the poorer-quality zone from elevation 420.6 m to 387.6 m. The recommended ground support consists of 3 m long resin-grouted #7 rebar on 2 m staggered spacing with 5 cm of MRS.
  - Two fault zones are interpreted. One occurs between 414 m and 415 m elevation and is a discrete structure with a logged aperture of 21 mm, and as such may not require any additional ground support during shaft development. The second fault occurs between 366.5 m and 367.5 m elevation and is a broken rock zone with 600 mm aperture. This fault may require additional ground support. For cost estimating purposes, BGC recommends a 5 m long zone of increased ground support between 364.5 m and 369.5 m elevation, consisting of 3 m long resin-grouted #7 rebar on 1.5 m staggered spacing with 15 cm of MRS. Increased inflows should also be expected through this fault zone.

16.2.3.2 Decline Geotechnical Characterization and Ground Support Recommendations

The decline from surface was analyzed separately to the rest of the rock mass due to its importance as one of the main access/egress points for the mine. Eighteen drill holes were advanced along the proposed Decline alignment to investigate the soil, bedrock, and hydrogeology along the alignment (BGC Engineering Inc. 2022f). Geotechnical characterization of the soils along the decline is presented in the same BGC report. The available rock data was assessed for rock mass quality, rock mass variability, presence of major structures, and discontinuity orientations, to design the rock support for the Decline.

Two major faults were interpreted by BGC to potentially be present along the decline. Fault Decline-1 dips shallowly to the east-northeast and is identified with relatively high confidence. Fault Decline-2 dips steeply to the south-southeast and is identified with low confidence. Fault Decline-1 is considered by BGC to present a greater risk to the development of the decline, because it dips shallowly and is identified with high confidence. It is estimated to intersect the decline approximately 750 m from the proposed portal, near the intersection with the mine level at elevation 400 m. This fault was intersected by PLS21-PDEC-009 (71.9 m to 77.6 m along hole), where it resulted in decreased recovery (variable from 43% to 100%) and decreased RQD (variable from 0% to 78%) with increased oxidation noticeable on some discontinuity surfaces. Long support consisting of 5 m long connectable anchors (MAI SDA or similar) on 2.5 m spacing may be required when advancing through the fault zone, and increased groundwater seepage should be expected.

Based on 2D numerical analyses of the mining activity nearest to the shafts, mining-induced displacements at the proposed alignment of the decline are expected to be on the order of 1 cm to 2 cm. The fault zones discussed above are not anticipated to affect the overall stability of the decline providing that the fault zones are mapped and characterized during construction and the ground support designs are confirmed using the mapping data prior to installation.

16.2.4 Geotechnical Inputs for the Portal and Decline Design

WSP Global Inc. (WSP) completed an updated study on the design of the portal and decline to access the Triple R deposit (WSP Golder 2022). A summary table of the testing results available are presented in Table 16-5. The data available is limited relative to what is required to assess the ground parameters with confidence, therefore conservative assumptions were made in design. There are no direct testing results available for the stiffness of the overburden units, therefore in the design calculations conservative estimates were made regarding these parameters based on descriptions and published correlations. Five UCS tests were undertaken on Sandstones (BGC 2017; BGC 2018a; BGC 2018b). Four UCS tests are available for the Gneiss bedrock. A number of point load tests have been undertaken on the sandstone, but no testing is available for the mudstones due to a lack of intact samples.

Table 16-5: Summary of Available Testing Information for Decline Design

Material	Bulk Density (g/cm^3)	Friction Angle Phi' (°)	Drained Cohesive Strength C' (kPa)	Undrained Shear Strength Cu (kPa)	UCS (MPa)	E (GPa)
Glaciofluvial Sands	1.74[1]	31–34[1] (32)	0	N/A	N/A	No direct stiffness testing on the overburden units. Design values estimated from correlations with SPT, CPT, or typical values
Till 2[2]	1.91 – 2.13 (2.0)	24–36 (30)	0	120 (n=1)	N/A	
Till 1[2] including highly weathered mudstone/siltstone	1.85–2.14 (2.0)	13.2–37.1 (25)	--	--	--	
Devonian Sandstone	2.23–2.64	N/A	N/A	N/A	1.2–63.6 (n=5)	0.1* (n=1)
Gneiss	2.71	N/A	N/A	N/A	14–160.1 (n=4)	3–77.69 (n=4)

Notes:
1. From previous 2018 testing.
2. Ground conditions for Tills 1 and 2 are very variable
* Low modulus values from a single test which also returned the lowest UCS value. Not considered representative.
* Bracketed values indicate the typical values adopted in design

Where necessary and appropriate, correlations with relevant test data or typical values based on descriptions were adopted for the design. The rock mass behaviour was estimated using Geological Strength Index and Hoek-Brown failure criteria. Using the Hoek-Brown failure criteria, equivalent Mohr-Coulomb friction angle and cohesion values have been calculated for the rock masses.

16.2.5 Box Cut and Portal Headwall Design

The main access to the mine will be by decline, utilized for vehicle access and transport of materials (WSP Golder 2022). The box cut portal will provide access to the entrance of the decline and will be formed by excavating a ramp into the Glaciofluvial sands. The sides of the ramp will be battered back to an angle such that it will be stable without any stabilization measures other than potentially a layer of shotcrete, if required. The portal face will be cut at approximately 82° as it is orthogonal to the decline angle which is approximately 15%, this face will be supported by shotcrete, mesh, and soil nails to provide temporary stability during the construction works. A steel liner plate tunnel will be constructed within the box cut excavation adjoining to the portal and backfilled to provide long term stability.

16.2.5.1 Box Cut Excavation Slope Stability Assessment and Support Recommendation

From the two boreholes near the portal location, the ground conditions near the surface comprise Glaciofluvial sands to 20 m depth, underlain by Till 2 (sands and silts) to 40 m depth. The initial depth of the portal is at approximately 14 m depth meaning it will be situated fully within the Glaciofluvial sands. The parameters of the Glaciofluvial sands and the underlying Till 2 are based on the available testing data as well as conservatively assumed parameters based on the available descriptions and borehole logs in the vicinity of the portal. In detailed design, further ground investigation and testing should be carried out locally at the portal entrance to further confirm the assumptions below.

The Mohr-Coulomb constitutive model is used to estimate the ground behavior. A summary of the parameters used for the portal slope stability assessment is shown in Table 16-6. A 10 kPa load is assumed at the top of the portal face and sides offset 2 m from the crest of the slopes as an exclusion zone for the duration of the excavation being open.

Table 16-6: Parameters for Slope Stability Assessment

Material	Unit Weight (kN/m^3)	Constitutive Model	Drained Cohesion (kPa)	Phi (°)
Glaciofluvial Sands	20[1]	Mohr-Coulomb	0	32
Till 2 (Sands and Silts)	20	Mohr-Coulomb	0	30

Notes:
1. Unit weight is increased as a conservative assumption to account for material variability.

As the portal slopes are located within sands, the portal face slope will require soil nails to ensure stability with respect to slip failures. The initial soil nailing layout is assumed to be 1.5 m c/c spacing with 12 m embedment with 350 mm x 350 mm x 20 mm facing plates as well as a facing of 200 mm of shotcrete and A142 mesh. The adequacy of this will be assessed at detailed design stages. The steel liner tunnel at the portal is to be designed and waterproofed by a proprietary manufacturer and was not covered in the study by WSP Golder. Figure 16-2 shows the box cut and portal headwall excavation design for the surface decline.

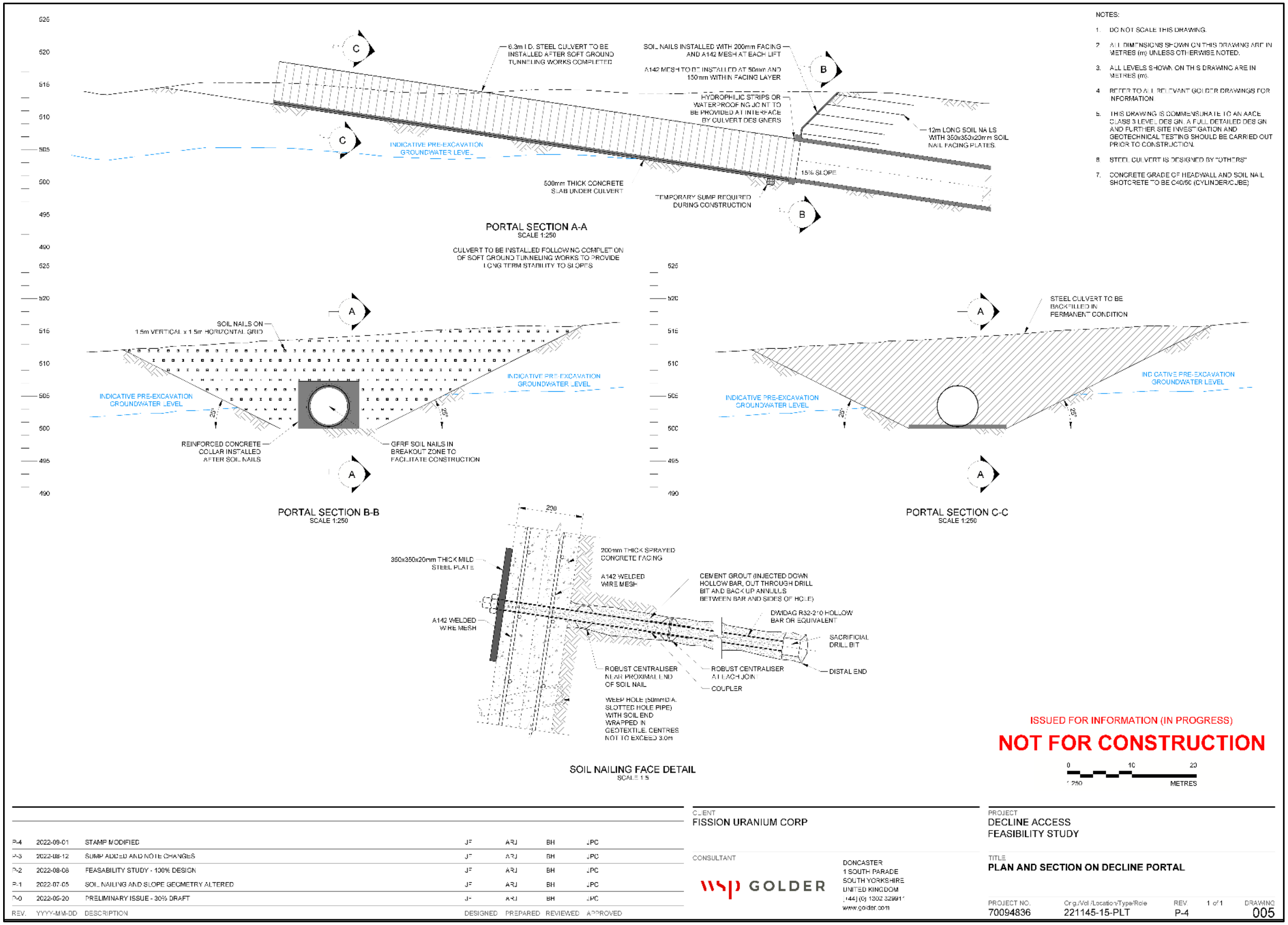


Figure 16-2: Plan and Section of Surface Decline Box cut and Headwall Portal Design

16.2.6 Surface Decline Design

A summary of the geology that the decline will encounter is in the WSP Golder report along with the full description of the study (WSP Golder 2022). The first 50 m to 60 m vertically will be situated within the glacial overburden. This is followed sequentially by zones of weathered sandstones, medium strong to strong sandstones, and gneiss bedrock, respectively. Two tunnel liner types are considered. For the glacial overburden and weathered sandstones, a segmental liner is adopted and for the hard rock zone below, rock bolt support and mesh are adopted.

16.2.6.1 Design Assumptions

- The decline internal diameter is 5.98 m.
- Tunnel is dewatered and depressurized prior to and during construction.
- The tunnelling shield provides adequate resistance to sand lenses during construction.
- The gradient of the decline is approximately 15%.
- Grouting quantities have not been assessed.
- Requirements for temporary measures to support the soft ground between excavation and jacking the segmental rings in have not been assessed.

16.2.6.2 Soft Ground Tunnelling

The following sets out the design for tunneling through the overburden material, i.e., Glaciofluvial sands, Till 2 and Till 1 as outlined in Figure 16-3. A concrete segmental liner will be utilized for the soft ground (overburden and weathered rock as appropriate). Table 16-7 outlines the soil properties adopted for the design of the soft ground tunneling liner. The structural design life of the lining is assumed to be 50 years. This will require regular maintenance throughout the asset’s lifecycle to ensure watertightness is maintained along with any areas of the lining that require repair.

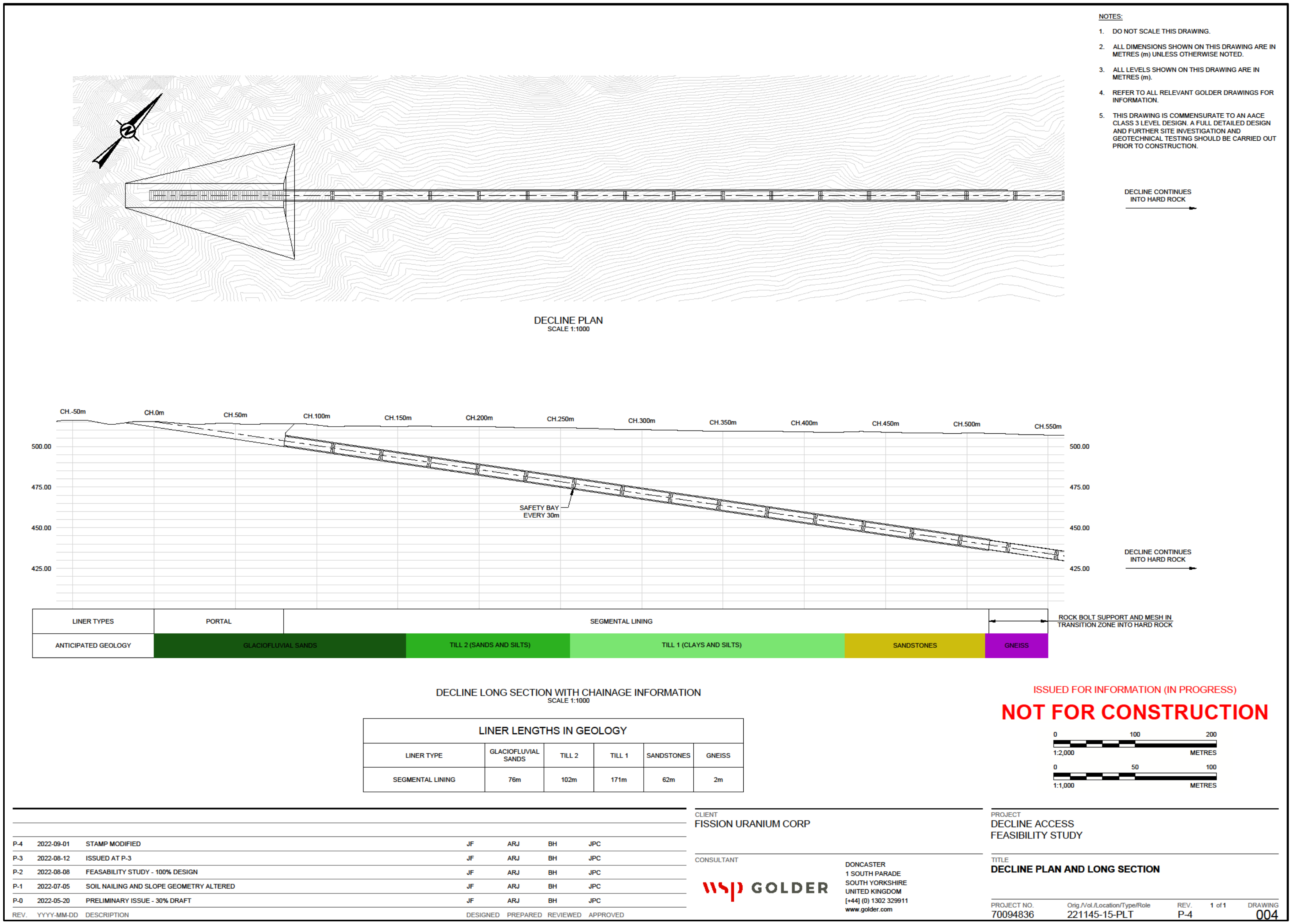


Figure 16-3: Decline and Box Cut Excavation Plan and Long Section.

Table 16-7: Assumed Parameters for Soft Ground Tunneling Design

Material	Unit Weight kN/m^3	Drained Cohesive Strength (kPa)	Friction Angle (°)	Stiffness E (GPa)	v
Glaciofluvial Sands	20[1]	0	32	30	0.35
Till 2	20	0	30	25	0.35
Till 1	20	75[2]	25	30	0.2

Notes:
1. Unit weight is increased as a conservative assumption to account for material variability.
2. Based on typical values for glacial tills

16.2.6.3 Seismic Design

No seismic analysis has been allowed for in this feasibility design study, however, underground structures generally have a good response to seismic events. From an initial review of the publicized peak ground acceleration (PGA), the value of ~0.02 g is low. Wang (1993) shows that underground structures such as tunnels generally perform well under seismic loading events and that tunnels with low PGA values experience low to no damage. Therefore, at feasibility stage this has not been considered but should be carried out in detailed design phases.

16.2.6.4 Tunnel and Liner Design

The proposed lining system is to be a reinforced concrete segmental liner. The liner system is made up of individual 500 mm thick concrete segments forming a complete ring. In-between each segment and ring will be a gasket for compliance and water resistance purposes. The loading for the liner was assessed for various loading cases and can be found in the WSP Golder report. Figure 16-4 shows the proposed segmental liner design for a tunnel typical section.

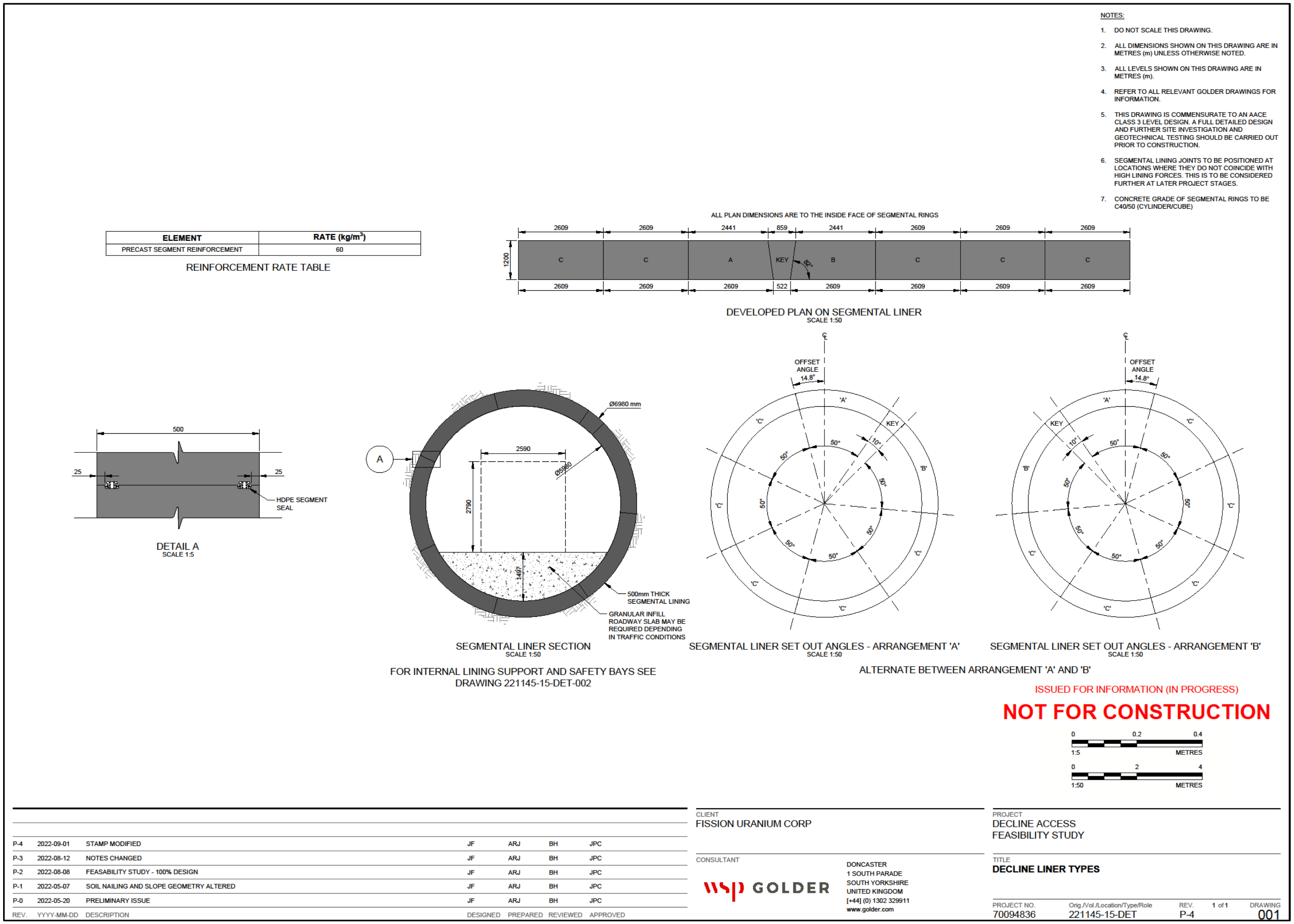


Figure 16-4: Decline Liner Section and Concrete Segment Arrangement

Figure 16-5 shows a cross section of the tunnel showing the spatial and clearance requirements for a 30 t (or similar) underground truck. Safety bays will be situated at chainage intervals of 30 m into the advancing decline. The approach for the safety bays is to situate them outside of the tunnel bore, in accordance with local regulations. The safety bays are to be constructed on the outside of the tunnel lining and require the lining to be “cut” to form an opening within it. Forming this opening breaks the structural continuity of the lining, and therefore the forces must be distributed around the opening. As the decline is anticipated to have a service life of approximately 50 years, a set of modular steel ring frames will be built on either side of each safety bay opening to properly distribute the stress around the modified liner section. The steelwork will be cast in concrete to provide protection against corrosion and impact from plant working within the tunnel. The steelwork will be cast in concrete to provide protection against corrosion and impact from plant working within the tunnel.

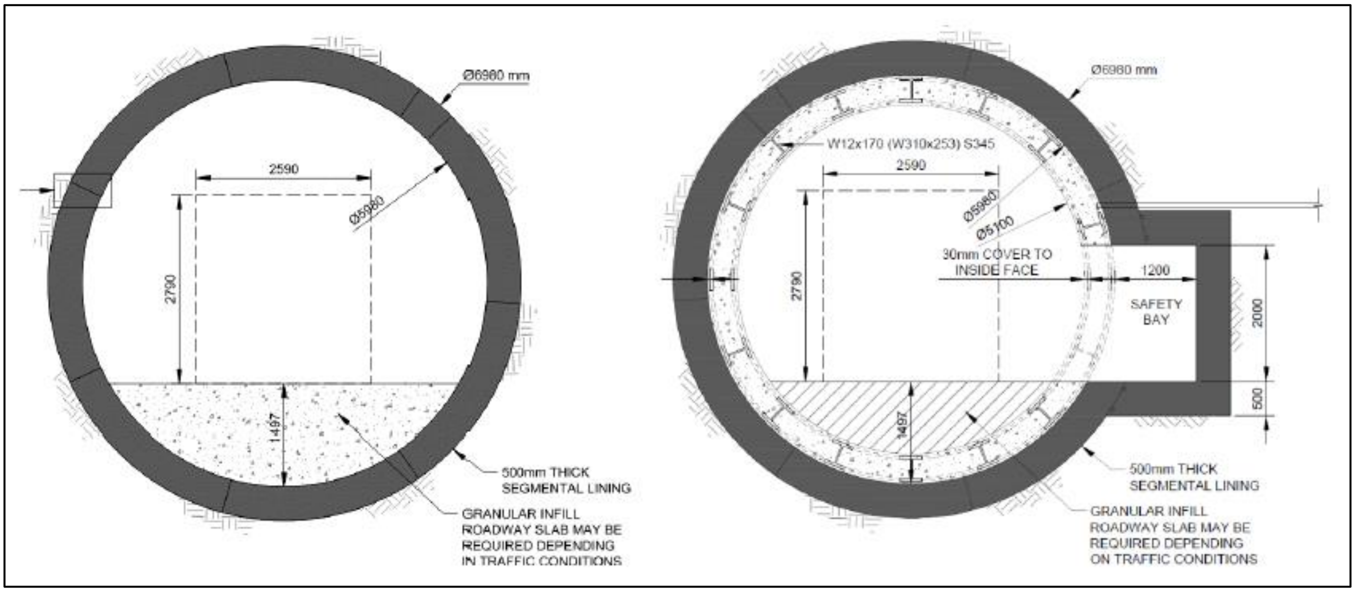


Figure 16-5: Decline Tunnel Cross Section

From the available borehole data, it is expected that there will be a transition zone within the rock between weathered rock into the competent hard rock. The tunneling shield will be required to be progressed fully into the competent rock in order for the segmental rings, which are installed behind the shield, to cover the full length of the overburden and weathered rock as well as for there to be a set amount of competent rock above the crown to ensure initial stability before the hard rock excavation can commence. The practicality of this transition should be assessed further at later stages of design as it will most likely require hard ripping or potentially hydraulic breaking to embed the tunneling shield far enough.

Based on available data, it is recommended that the shield is progressed sufficiently far into the competent Gneiss to allow 2 m of competent rock cover above the final segmental ring. This would result in approximately 2 m to 3 m of segmental liner being required from the Sandstone:Gneiss contact to transition to mesh and bolting.

16.2.6.5 Recommendations

As the project moves into more detailed execution phases, WSP Golder recommend that the following be undertaken:

- Detailed design phases to produce finalized design prior to construction.

- Further geotechnical testing and ground investigation to verify assumed parameters for design. In particular, triaxial testing of cohesive horizons to assess undrained strength, stiffness testing (e.g., pressure meter if practicable), and additional characterization of the extent of sand lenses in the decline alignment.
- Further geomechanical analysis to realize opportunities to optimize lining thickness and verify joint design.
- Engagement with regulatory authorities to understand if there are other opportunities relating to incorporation of safety bays.

16.2.7 Crown Pillar Design

The crown pillar stability analyses were based on empirical analyses and two-dimensional numerical modelling to assess the impact of rock mass strength, rock mass structure, and mining-induced stress changes on crown pillar stability (BGC Engineering Inc. 2022e). Due to the risk of inflow from Patterson Lake and the saturated overburden above the proposed underground mine, the objectives of the crown pillar stability analyses were to limit deformation of the crown pillar and reduce the potential for uncontrolled groundwater inflow through fractured (damaged) rock. For the purposes of the work completed, the "crown pillar" was defined as the thickness of bedrock (including the sedimentary bedrock, where present) below the overburden (where overburden includes the glaciofluvial and till units) and above the uppermost production level.

The results of the empirical and analytical analyses showed that the potential for shear failure along the abutments (indicated by rigid beam analyses) governs stability when crown pillar thickness is less than total span, and rock mass instability within the crown pillar governs stability when crown pillar thickness is equal to or greater than total span. Overall, the crown pillar stability is governed by the low rock mass strength of the Resource unit, the high groundwater table, and the thick overburden (soils) above the crown pillar.

The crown pillar design recommendations based on analytical, empirical, and numerical analyses for each zone are summarized below. If the results of the analytical and empirical analyses did not align with the results of the numerical analyses, the analytical and empirical designs were carried forward because the numerical model was uncalibrated.

- R840W
  - For variation of total span from 5 m to 15 m (i.e., up to three D/F excavations) on a single level, a crown pillar thickness (sedimentary and basement bedrock) of 20 m is recommended.
  - Total spans of 20 m (i.e., four D/F excavations on a single level) or greater are not recommended.
- R00E
  - For total spans up to 5 m, a minimum crown pillar thickness of 5 m is recommended.
  - For total spans from 5 m to 10 m, a minimum crown pillar thickness of 10 m is recommended.
  - For total spans from 10 m to 15 m, a minimum crown pillar thickness of 15 m is recommended.
  - For total spans from 15 m to 2 0m, a minimum crown pillar thickness of 20 m is recommended.
  - To recover more than 20 m of mineralization on a single level, a rib pillar of 10 m is recommended between adjacent D/F areas when total spans greater than 20 m are required.

- R780E
  - For the western frozen portion:
    - For total spans up to 5 m, a minimum crown pillar thickness of 5 m is recommended.
    - For total spans from 5 m to 10 m, a minimum crown pillar thickness of 10 m is recommended.
    - For total spans from 10 m to 15 m, a minimum crown pillar thickness of 15 m is recommended.
    - For total spans from 15 m to 20 m, a minimum crown pillar thickness of 20 m is recommended.
    - Total spans exceeding 20 m are not recommended.
  - For the eastern unfrozen portion:
    - For total spans less than 10 m, a minimum crown pillar thickness of 15 m is recommended.
    - For total spans from 10 m to 15 m, a minimum crown pillar thickness of 20 m is recommended.
    - For total spans from 15 m to 20 m, a minimum crown pillar thickness of 35 m is recommended.
    - Total spans exceeding 20 m are not recommended.

The excavations immediately beneath the crown pillar should be supported with pattern bolting (Swellex), and 10 cm of MRS. Higher-capacity ground support (reinforced shotcrete ribs, steel sets, or lattice girders) is recommended where rock mass quality is very poor (Q < 1), and in the R840W when probe drilling indicates the sandstone thickness is absent or thin (< 5 m).

Depressurization prior to excavation of the crown pillar will reduce the potential for instability. Excavations should be tight-filled as soon as possible after excavation. Filling all gaps between the cured backfill and the back of the excavation with shotcrete may improve overall stability of the crown pillar and allow for increased recovery. The crown pillar should be mined at the end of the mine life in a retreat sequence to reduce the potential for sterilization of the entire crown pillar if a portion of the crown pillar becomes unstable and/or unmanageable groundwater inflow occurs. Mineralization within 10 m of the basement bedrock contact should be considered as having a low potential for recovery, regardless of the sedimentary bedrock thickness.

The crown pillar areas of all three zones (R840W, R00E, and R780E) should be isolated from the remaining mine excavations using hydrostatic bulkheads after completion of mining within each respective zone. Pre-mining construction of hydrostatic mine doors at each access point to the D/F areas for each zone should also be considered as an inflow risk mitigation measure.

16.2.8 Geotechnical Uncertainties and Related Recommendations for Further Work

This section summarizes BGC's perception of the data gaps and geotechnical uncertainties associated with the underground rock mechanics assessment presented in their report (BGC Engineering Inc. 2022e). Recommendations to address remaining gaps in information, increase the reliability of the assessments, and evaluate mitigation options for risks identified are also presented.

16.2.8.1 Geotechnical Units, Structural Domains, and Discontinuity Strengths

Optimization of the proposed mine plan, including crown pillar thickness and stope dimensions, would be possible with additional refinement of the geotechnical units, structural domains, and design discontinuity sets. Geotechnical drill holes to collect additional geotechnical information should be used to confirm the geological interpretations and the geotechnical parameters of the rock mass and discontinuities. The drilling program should include packer testing above, below, and across/within faults or geologic contacts. Areas requiring additional geotechnical information include the Resource geotechnical unit to better understand the variability of that unit, the Sedimentary Bedrock unit to further define the distribution and variability of that unit (particularly where it forms part of the crown pillar), and the rock mass in the northeast of the R780E footwall where footwall development excavations are proposed and where lower quality rock and a potential major geologic structure have been identified. Drilling to target the Underground Infrastructure Area and an inferred major structure (fault) in proximity to the FAS are also recommended.

The drill holes should be oriented to minimize blind zones in the combined structural data set and target the inferred large-scale structures based on work presented in this Technical Report. Televiewer surveying should be conducted in all drill holes to collect oriented discontinuity data and to assist with the identification and interpretation of large-scale structures, and piezometers should be installed to monitor groundwater pressures both before and during mining. Additional laboratory testing including UCS, Brazilian tensile strength (BTS), triaxial, frozen creep, and frozen UCS testing should be completed.

The scheduling of these investigations relative to the overall project schedule should be considered with respect to the potential operational and financial risks presented by deferral of the work. Some or all of the recommended work could be done early in development from the underground, when collected data and observations about ground performance could be used to calibrate the existing numerical models.

16.2.8.2 Structural Geological Model

The structural geologic model remains poorly defined at this stage of study, therefore there is uncertainty regarding the location, orientation, and geotechnical characteristics of large-scale structures (faults) across the project area. Because large-scale structures have the potential to affect stability of the crown pillar, stopes, EAS and FAS, lower decline, internal ramps, and mine development drives, BGC has generated a preliminary model based on available data.

The current model developed by BGC has been developed through a desktop study of available geotechnical and geological data. It is recommended that the high confidence faults with the potential to impact major infrastructure be targeted in future drilling campaigns.

The reliability of the rock mechanics assessments could be improved if further geological modelling work could confidently define the location, orientation, and geotechnical characteristics of geologic structures.

16.2.8.3 Three-Dimensional Geotechnical Model

The strength and quality of the Resource geotechnical unit is highly variable. The preliminary variability assessment conducted as part of the current study should be updated as additional data is collected during detailed design, and as mining commences and ground performance can be assessed. Increased drilling through the Resource geotechnical unit with a focus on geotechnical parameters and/or clay intensity and friability could help better define the higher-quality and lower-quality zones within the Resource.

Laboratory UCS samples are lacking in the weakest intervals of the Resource unit (there are none in either of the more friable bin groups to compare to less friable rock), therefore more intact UCS samples should be taken, if possible, to improve this database. Less invasive sampling methods such as Pitcher sampling or Shelby tube sampling or alternative constitutive models for the weakest intervals of the Resource unit should also be considered.

The preliminary variability assessment should be used as the starting point for a mine-wide 3D geotechnical model, which can be continually updated as mine development occurs and would allow calibration and refinement of the rock mechanics assessments presented in this Technical Report.

16.2.8.4 Mine Planning Considerations

The mine plan should account for the potential for open historical drill holes, particularly those collared on Patterson Lake, because they present an inflow risk. BGC recommended that FCU compile a "vertical opening register" that summarizes the as-built and completion data for all historical drill holes, and is kept updated as additional drilling and other vertical development occurs. Prior to excavation, the surveyed downhole path of all drill holes should be plotted on driving layouts to anticipate drill hole intersections, and necessary contingency devices such as packers or stem pipes should be made available to manage potential water inflows.

The crown pillar areas of all three zones (R840W, R00E, and R780E) should be isolated from the remaining mine excavations using hydrostatic bulkheads after completion of mining within each respective zone. Pre-mining construction of hydrostatic mine doors at each access point to the D/F areas for each zone should also be considered as an inflow risk mitigation measure.

BGC recommended that the crown pillar design assumptions be confirmed using a probe and grout program prior to excavation within 15 m of the inferred basement bedrock contact. Accurate and precise confirmation of the basement bedrock and sedimentary bedrock units are critical to crown pillar stability, and a poor understanding of those contacts increases the potential for crown pillar instability.

16.3 Hydrogeology

BGC was retained by FCU to undertake hydrogeological assessments to support mine planning activities for the FS (BGC Engineering Inc. 2022b). The main purpose of this work was to provide (a) an updated conceptual hydrogeological model and numerical groundwater flow model, (b) FS-level estimates of the range of dewatering rates and number of wells needed for dewatering and depressurization for construction of the proposed decline, and (c) FS-level estimates of the range of groundwater seepage that could be encountered in the underground development during the LOM.

The conceptual model previously developed by BGC in 2019 was updated with results from drilling investigations and groundwater level data collected by BGC up to June 2022. No substantial changes were made to the conceptual model, but the additional borehole logs, hydraulic conductivity data, groundwater monitoring locations, and extended hydraulic head records provided an expanded dataset for building and calibrating the numerical model.

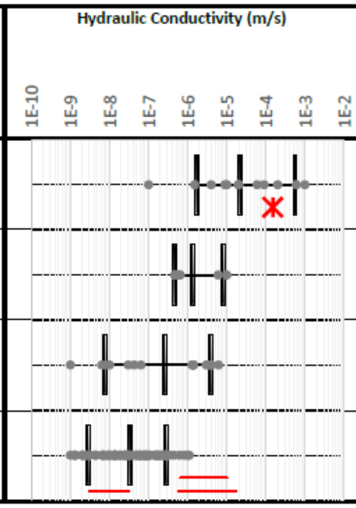
16.3.1.1 Hydrogeological Parameters and Hydrostratigraphy

A hydrostratigraphic unit (HSU) comprises one or more geologic units with similar hydrogeologic properties (e.g., hydraulic conductivity). The hydrostratigraphic interpretation of the PLS site was developed from field data collected between 2016 and 2021 and builds on the geologic and hydrogeologic interpretations presented in previous works by BGC (BGC Engineering Inc. 2018c; 2019b; 2022d; 2022f). Results from in-situ hydraulic testing data (i.e., 83 packer tests, 26 single-well response tests, and 4 pumping tests), hydraulic head data (i.e., 84 vibrating wire piezometers [VWPs], 10 monitoring wells), and borehole logs were also used to inform interpretations of the HSUs presented here.

Table 16-8 and Figure 16-6 show a summary of hydraulic conductivity estimates for in-situ hydraulic testing conducted at the PLS site for each HSU. The high variability in the hydraulic conductivity and thicknesses for each HSU is reflective of the complex glacial history of the region.

Four HSUs are defined in the PLS site (from shallowest to deepest): Upper Aquifer (glaciofluvial sands), Lower Aquifer (Till 2), Aquitard Complex (primarily Till 1, with Cretaceous derived siltstones and claystones, Devonian siltstones and sandstones, and Cretaceous mudstones), and Basement Rock (granitic and metasedimentary gneisses, which host the mineralization).

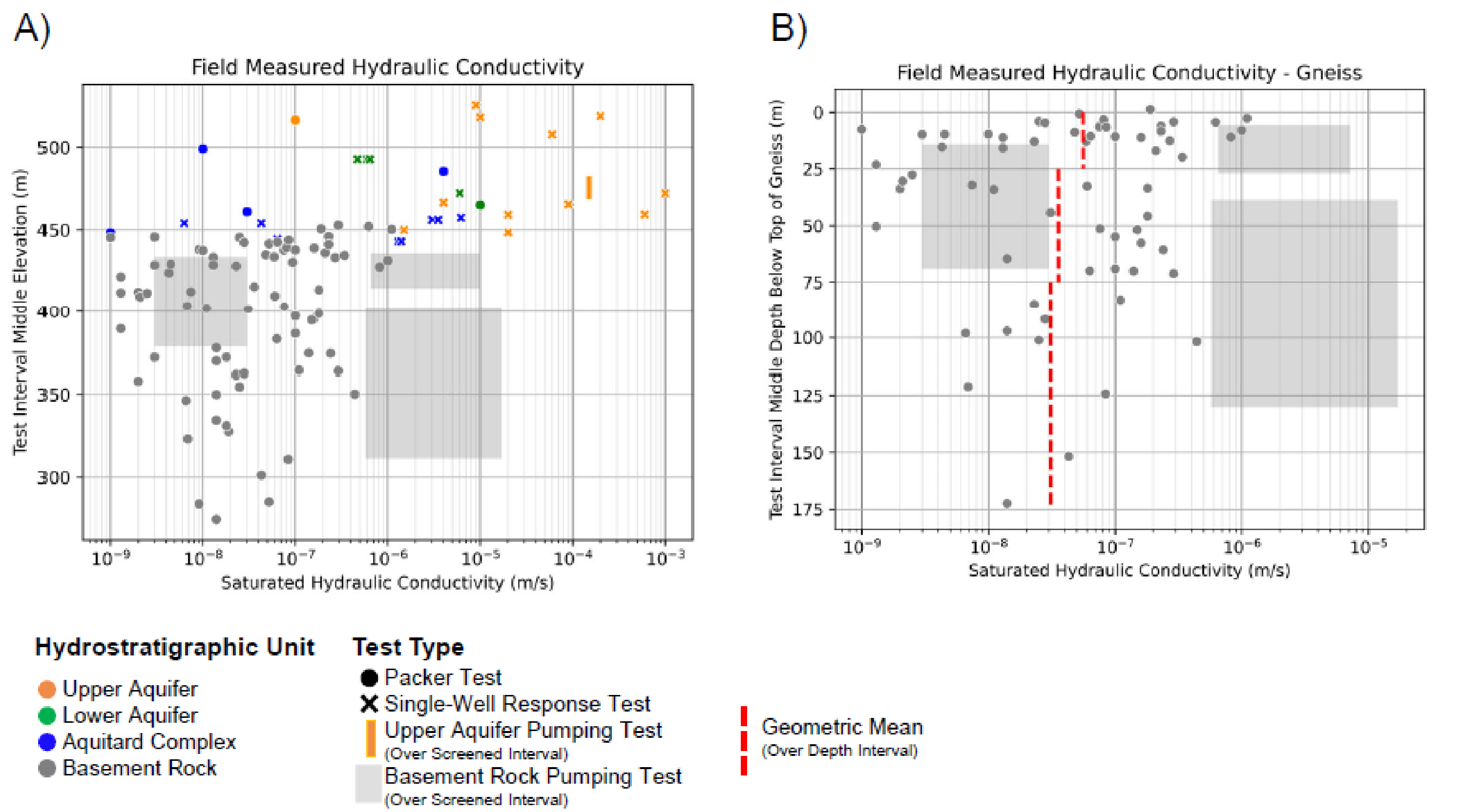
Table 16-8: Summary of Hydraulic Conductivity Data



Hydrostratigraphic Unit	Hydraulic Conductivity (m/s)	Field Data Summary			
		10th Percentile (m/s)	Geometric Mean K (m/s)	90th Percentile (m/s)	Test Count
Upper Aquifer		1.8E-06	2.2E-05	5.6E-04	12
Lower Aquifer		4.7E-07	1.3E-06	8.0E-06	6
Aquitard Complex		7.4E-09	2.5E-07	3.9E-06	14
Basement Rock		2.8E-09	3.4E-08	2.8E-07	77

Notes:

1. Values from hydraulic testing by BGC Engineering Inc. (2017; 2018a; 2018b; 2018c; 2019c; 2019d; 2020; 2022d; 2022f) and Clifton (2019).
2. Red symbols represent values and ranges of hydraulic conductivity estimated from pumping tests (BGC Engineering Inc. 2022d; 2022f; 2020).
3. Statistics for all units do not include pumping tests due to the differences in scale between packer and slug tests and pumping tests.



NOTES
1. THIS FIGURE SHOULD BE READ IN CONJUNCTION WITH BGC'S REPORT TITLED "HYDROGEOLOGICAL ASSESSMENT FOR THE DECLINE AND THE UNDERGROUND DEVELOPMENT", AND DATED NOV 14, 2022.
2. VERTICAL DATUM IS CANADA GEOID MODEL HTV2.0
3. GEOMETRIC MEAN DOES NOT CONSIDER PUMPING TESTS.

PREPARED BY:	FIGURE TITLE:		
KJH	HYDRAULIC CONDUCTIVITY DATA		
CHECKED BY: WZ	CLIENT: FISSION URANIUM CORP. PATTERSON LAKE SOUTH		
APPROVED BY: RT	SCALE: AS SHOWN	PROJECT NO: 1665009	FIGURE NO: 2-1

0 5 10 mm in ANSI A sized paper

BGC ENGINEERING INC.

Figure 16-6: Hydraulic Conductivity Data by BGC

16.3.1.2 Observed Hydraulic Heads and Inferred Hydraulic Gradients

The predominant lateral hydraulic gradient in all the HSUs in the Project area is from the high elevation areas in the west towards Patterson Lake in the east, averaging 0.01 m/m.

Vertical gradients at the PLS site are relatively small in comparison to the dominant lateral gradient described above. Vertical hydraulic gradients across the PLS site are summarized below:

- Vertical hydraulic gradients between the ground surface and the Upper Aquifer suggest that flow is generally directed downward. The vadose zone ranges in thickness with estimated water table depths ranging from 4.1 m (BH-BGC16-RD-03, near Patterson Lake) to 16 m below ground surface (PLS19-TMF-08, near the primary proposed TMF area).
- Below the water table and within the Upper Aquifer, vertical hydraulic gradients are variable, with downward gradients in areas of higher elevation (e.g., PLS18-TMF-01, BH-BGC16-RD-02) and upward gradients at lower elevations near Patterson Lake (e.g., BH-BGC16-RD-05).
- Vertical hydraulic gradients between the upper and Lower Aquifer and the Aquitard Complex suggest that vertical flow is spatially variable (i.e., both upwards and downwards, depending on location).
- Vertical hydraulic gradients within the Basement Rock suggest that vertical flow is spatially variable (i.e., both upwards and downwards, depending on location).
- Vertical hydraulic gradients between the Upper Aquifer and the Basement Rock (across both the Lower Aquifer and Aquitard Complex) suggest that vertical flow is spatially variable.

16.3.1.3 Groundwater Recharge and Discharge

Water that enters the saturated zone or otherwise is "added" to the groundwater reservoir is called groundwater recharge, whereas water that is removed from the groundwater reservoir is called groundwater discharge.

The primary source of groundwater recharge to site is diffuse meteoric recharge. Meteoric recharge occurs as the percolation of rain or snowmelt through the unsaturated zone directly into the Upper Aquifer. Small surface water bodies, such as ponds and ephemerally saturated areas can also act as focused sources of recharge. These are normally located in isolated topographic lows and are fed by snowmelt and surface runoff, which subsequently percolates through the unsaturated zone into the water table. No focused studies on groundwater recharge rates have been conducted at the PLS site; however, after the calibration process for the 2022 PLS site numerical hydrogeological model update (BGC Engineering Inc., 2022b) a groundwater recharge value of 80 mm/yr was adopted.

Groundwater can also be recharged through inter-aquifer or cross-formational flow, whereby groundwater from a larger, more distal flow system flows into the project area laterally or from below (Healy 2010). This concept is supported by the upward vertical gradients observed within the Upper Aquifer and between the Basement Rock and Upper Aquifer.

The groundwater within the PLS site primarily discharges to Patterson Lake, which subsequently discharges to downstream lakes. The average annual outflow from Patterson Lake is approximately 3.2 x 10^7 m^3/year (Canada North Environmental Services Limited Partnership 2022), which for the area estimated as the watershed for the Patterson Lake outlet (i.e., 2.54 x 10^8 m^2; [BGC Engineering Inc., 2019, January 31]) equates to approximately 125 mm/year. With annual precipitation and annual potential evapotranspiration being 490 mm/year and 550 mm/year, respectively (BGC Engineering Inc., 2019a), it is possible that the groundwater catchment (e.g., the contributing area or groundwater capture area) for Patterson Lake is larger than the surface water catchment.

Groundwater flow systems in landscapes with low-relief and high hydraulic conductivity, such as the PLS site, are much less likely to have groundwater divides underlying topographic highs and therefore the regional water table likely is a gently sloping surface from one large surface water body to the next.

16.3.1.4 Conceptual and Numerical Hydrogeological Model Update

The general direction of groundwater flow is interpreted to follow topography eastward towards Patterson Lake. The PLS site is interpreted to be recharged by meteoric water and by adjacent/larger groundwater flow systems, and to discharge primarily into Patterson Lake. In areas with standing water and shallow water tables, groundwater also leaves the system through evapotranspiration. The majority of this flow is interpreted to occur within the Upper Aquifer due to its continuity, considerable thickness, and hydraulic conductivity that is high relative to the hydraulic conductivity inferred for the other HSUs.

An updated 3D groundwater model was developed using MODFLOW-USG, with grid refinement around the proposed decline and underground development. The model was calibrated to 49 steady-state hydraulic head targets, and 3 transient pumping tests (one test in the overburden and two tests in the Basement Rock). Calibration results from this assessment yielded a normalized root mean square error of 6.0% for the steady-state targets and reasonable match to transient responses, indicating the groundwater flow model is sufficiently calibrated.

16.3.1.5 Dewatering Assessment for the Decline

Construction of the decline through the overburden will require dewatering or depressurization of those materials along the alignment before the hydrostatic liner is constructed and sealed. The calibrated numerical groundwater model was used to predict the number of wells and associated pumping rates that may be required to sufficiently dewater the decline prior to construction, with the objective of lowering the pressure head in these units to near 0 kPa.

Two parallel lines of pumping wells that straddled the decline were considered in the assessment, with dewatering wells completed in the Upper Aquifer and a second set of deeper depressurization wells completed in the lower-permeability Lower Aquifer and Aquitard Complex present along the current decline alignment. The resulting system consisted of 16 pumping centres with 2 to 4 wells located at each pumping centre and straddling the decline, as follows:

- One pumping centre comprising a pair of dewatering wells straddling the decline.
- Eleven pumping centres comprising a pair of dewatering wells and a pair of depressurization wells.
- Four pumping centres comprising a pair of depressurization wells.

The near steady-state (e.g., longer term) total pumping rate for all pumping centres combined was approximately 15,000 m^3/day (2,700 gpm), with the dewatering wells in the Upper Aquifer contributing more than 90% of this flow. The Upper Aquifer was predicted to reach steady-state conditions after approximately 90 days of start of pumping, while the Lower Aquifer and Aquitard Complex approached steady state conditions in approximately 120 days and a year, respectively. At the early stages of operations, total pumping rates were as high as approximately 16,500 m^3/day (3,050 gpm) and 1,500 m^3/day (200 gpm) for the dewatering and depressurization wells, respectively. This dewatering scenario resulted in reductions in pressure heads to near 0 kPa along the decline in the overburden.

A potential means of reducing the required pumping rates could be to install a slurry or bentonite cut-off wall or "hydraulic barrier wall" through the overburden materials between the decline and Patterson Lake. The effectiveness of a hydraulic barrier wall was assessed conceptually in the model, and it was shown to reduce the total predicted pumping rate for the dewatering/depressurization wells by an average of 65% relative to the rates predicted in the absence of such a wall. The decline cut-off wall was therefore carried forward as part of the mine access construction.

16.3.1.6 Underground Development Seepage Assessment

Transient simulations were performed using the calibrated groundwater flow model to evaluate seepage rates into the underground workings and hydraulic head distributions around the workings for 3D mine plan files received from Mining Plus on June 20, 2022 (17 years total in mining plan, including construction years). Two key model scenarios were considered in the analysis:

- Unmitigated Groundwater Seepage: In this scenario, it was assumed that all mine workings receive seepage from the surrounding rock mass, and that the seepage is unimpeded by any mitigation attempts.
- Mitigated Groundwater Seepage: In this scenario, it was assumed that an extensive cover grouting program is implemented for all mine development (e.g. accesses, drifts, drives, stopes, etc.), such that potential seepage from discrete zones of enhanced permeability is mitigated.

In the base case unmitigated scenario, total simulated groundwater seepage to the underground mine workings increases from less than 1,000 m^3/day (150 gpm) early in the mine life to 14,000 m^3/day (2,550 gpm) when the mine reaches its ultimate depth of 220 m elevation, and gradually decreases to 13,000 m^3/day (2,350 gpm) in the last years of mining. Due to its position under the lake, during years in which both mine areas are active, seepage into the R00E and R780E mine areas comprised between 93% and 98% of total seepage, with flow into the R840W mine area only reaching between 500 m^3/day (50 gpm) and 1,000 m^3/day (200 gpm). Predicted groundwater seepage to R840W reaches its peak when the R840W reaches its ultimate depth of 340 m elevation.

Under mitigated conditions that includes cover grouting of all underground openings, the total mine seepage was predicted to increase to approximately 3,000 m^3/day (600 gpm) when the mine reaches its ultimate depth and then gradually decrease to 2,500 m^3/day (500 USgpm) in the last years of mining. Overall, under mitigated conditions, total predicted seepage to the underground development was lower by approximately 80% when compared to the un-mitigated seepage, which is considered reasonable assuming that a continuous zone of grouted rock mass is created around all mine openings. Under unmitigated conditions, groundwater seepage to the freeze drilling drives averaged approximately 100 m^3/day (20 gpm). This represents less than 1% of total seepage rates to the underground workings. Simulating pre-grouting resulted in predicted seepage rates to the freeze drilling drives being reduced from the un-mitigated seepage by an average of 45% and 99%, using conventional and specialized grout, respectively. The lower bound estimate of mine seepage rates in the absence of cover grouting was 10,000 m^3/day (1,850 gpm) when the mine reaches its ultimate depth.

16.3.1.7 Recommendations

BGC offers the following recommendations based on the hydrogeological assessment for the decline and the underground development planned at the PLS site:

- It is recommended that the decline dewatering/depressurization system is fully commissioned and operational at least six months prior to the start of decline construction, and that progress of dewatering/depressurization is closely monitored within this six-month period and thereafter. This will require a network of multi-level piezometers that will need to be installed along the decline alignment. Data from this monitoring system would be used to determine if additional infill dewatering and depressurization wells are needed. Should the monitoring system indicate that sufficient depressurization is not being achieved, it is recommended that a contingency of 25% for installation of additional wells be carried forward. It is assumed that 15 such piezometer nests located mid-way between neighboring pumping centres would be necessary, as follows:

- Completed as grouted-in installations in a 6-inch Inner Diameter (ID) borehole.
- With four VWP tips per nest, one tip located in each hydrogeologic unit of interest (in the Upper Aquifer, Lower Aquifer, Aquitard Complex, and shallow portion of the Basement Rock).
- Equipped with dataloggers for continuous head monitoring.
- Ranging in depth between 15 m and 80 m below ground surface.

- The simulated seepage rates to the underground development reported are highly dependent on the proximity of the underground workings to the overburden-Basement Rock contact (i.e., within 15 m), the presence of Patterson Lake above the underground development, and the hydraulic conductivity of the modelled Basement Rock surrounding the mine under the lake. The model was calibrated to a pumping test performed in the Basement Rock under the lake (BGC 2020) which yielded relatively high hydraulic conductivity estimates (i.e., 5×10^{-7} to 2×10^{-5} m/s) for the Basement Rock. Given the relatively high predicted seepage rates and the sensitivity of the analyses to the bedrock hydraulic conductivity derived from the pumping tests, BGC recommends that FCU complete two additional pumping tests in the location of the proposed mine under Patterson Lake. Ideally, one pumping test would be conducted in each of the hanging wall and footwall of the deposit, and two observation VWP nests would be installed for each test. These pumping tests will provide better estimates of bulk hydraulic conductivity in and around the proposed underground development, which will expand the Basement Rock hydraulic property data set and increase confidence in seepage estimates.
- As further hydraulic testing is complete and additional data collected (e.g., pumping tests, packer tests and single well tests, hydraulic head data), the numerical model should be re-evaluated. This should include updating the steady state and transient calibrations of the model and revising model predictions.
- Opportunities exist to further optimize the current mine plan to provide additional reduction in mine seepage rates. For example, this could include: 1) sequential mining of individual ore bodies followed by hydraulic isolation of mined-out areas using tight backfill and strategically placed bulkheads, 2) accelerated mining in areas where higher seepage is expected, followed up by hydraulic isolation, and 3) sequencing development of accesses and drifts so it coincides with stope development. It is recommended that these and other options are further explored during subsequent engineering studies.
- It is understood that FCU decommissioned boreholes drilled beneath Patterson Lake during their past exploration campaigns by grouting. However, if unknown open boreholes are present or if some of the known boreholes were poorly decommissioned, they could result in significant mine inflow if encountered during mine advancement. It is thus recommended that historical records for any exploration programs in the area by others together with FCU’s borehole abandonment records be reviewed, and results incorporated into future mine planning.

16.4 Radiation Protection

In preparing the mine design, the radiological protection of underground and surface employees played a critical role. In uranium mining there are several methods of radiation exposure, gamma rays, alpha particles, beta particles, radon gas, and the decay of radon gas into radon progeny. Among these, gamma radiation and radon progeny are the primary concern for underground mining operations. The CNSC provides strict standards for the amount of radiation a worker can be exposed to over a specific period (typically five years). Based on this, a company is required to establish annual, quarterly, monthly, weekly, and daily radiation limits that a worker is allowed to receive.

There are four main tenets used to minimize the exposure to radiation. These are time, distance, shielding, and ventilation.

- **Time:** As radiation exposure is time dependent, limiting the amount of time a worker spends in an area of increased radioactivity is critical.
- **Distance:** Maximizing the distance a worker needs to be from a radiation source limits exposure.
- **Shielding:** Increased shielding (equipment, shotcrete ground support, etc.) can limit a worker's exposure to gamma radiation.
- **Ventilation:** Design a ventilation system that can efficiently remove airborne contaminants. This is particularly important for radon gas and progeny.

The mine plan and design has taken the four tenets of time, distance, shielding, and ventilation all into consideration. The ventilation system is designed to employ a "single-pass" methodology. This approach means that all fresh air reports to the exhaust leg of the ventilation system as soon as it encounters mineralized headings and stopes. This results in no ventilation being reused once it comes into contact with contaminants. For waste development headings, air is allowed to be reused provided that it meets the quality standards required.

Radiation shielding has been considered in both mobile equipment and ground support standards. The ground support in mineralized zones allows for a minimum of 50 mm of shotcrete floor to floor and a poured concrete floor which will provide effective shielding against the radiation bearing mineralization. The mine design has been performed in a manner that minimizes the time and maximizes the distance an employee is in contact with the radiation bearing mineralization. For example, mobile equipment has been selected with remote operation capability where practicable. This includes provisions for remote operation of production LHD's during stope mucking and backfilling activities as well as remote operation of roadheader equipment during tunneling operations.

16.5 Mine Design

16.5.1 FS Underground Trade-off Study Workshop

The initial phase of FS work involved conducting a trade-off study and workshop to select the optimal mine access, material handling system and mining method to proceed with into the main feasibility study phase. The FS underground trade-off study process was separated into four major areas with each using a qualitative ranking matrix to first assess any fatal flaws and then rank the various options. The four major areas are shown below:

- Mine Access and Development Upper Trade-off Study.
- Mine Access and Development Lower Trade-off Study.
- Material Handling Trade-off Study.
- Mining Methods Trade-off Study.

The Mine Access and Development Upper Trade-off Study assessed the following lateral and vertical options to access the mine from surface as well as the proposed construction method.

- Option 1 - Lateral Decline - Dewatering & Tunnelling Shield Method (Concrete Liner).
- Option 2 - Lateral Decline - Dewatering & Tunnelling Shield Method (Steel Liner & grout).
- Option 3 - Lateral Decline - SEM Development with Steel Liner & grout (poling plates).
- Option 4 - Lateral Decline - SEM Utilizing Grout Injected Lances.
- Option 5 - Vertical Shaft - Hoisting (6.5m Internal Diameter), Blind sink with freeze & Concrete liner.

- Option 6 - Vertical Shaft - Hoisting (6.5m Internal Diameter), Blind sink with freeze & Steel liner.

The trade-off study resulted in Option 1 - Lateral Decline - Dewatering & Tunnelling Shield Method (Concrete Liner) being selected for the next phase of the FS.

The Mine Access and Development Lower Trade-off Study evaluated the various equipment and technology available to complete lateral development both in unmineralized and mineralized hard rock zones:

- Option 1 – Conventional Drill and Blast.
- Option 2 – Road Header.
- Option 3 – Mobile/Continuous Miner.
- Option 4 – Mechanized Cutting (Disc cutting).

The Mine Access and Development Lower Trade-off Study resulted in Option 1 and Option 2 being selected for the next phase of the FS.

The Material Handling System Trade-off Study evaluated the various options to transport ore, waste and backfill material streams from underground and on surface. This included:

- Option 1 - Truck Haulage Surface Decline (Diesel).
- Option 2 - Decline Conveyor (Back Mounted).
- Option 3 - Production shaft at 6.5 m diameter w/ double drum hoist.
- Option 4 - Production shaft at 6.5 m diameter w/ single drum hoist.

The Material Handling System Trade-off Study resulted in Option 1 being selected for the next phase of the FS.

The Mining Methods Trade-off Study evaluated the various mining methods in the 2019 PFS and their suitability to the PLS deposit and options for optimization. This subset of methods that passed the initial suitability tests included:

- Option 1 - Primary and Secondary Open Stoping (Cemented Backfill) - Transverse
- Option 2 - Stoping Longitudinal (Cemented Backfill)
- Option 3 - Drift and Fill (D&B Overhand Method) with cemented backfill
- Option 4 - Drift and Fill (Mechanical Excavation Overhand Method) with cemented backfill

The Mining Methods Trade-off Study resulted in Option 2 and Option 4 being selected for the next phase of the FS.

Based on the trade-off study workshop and discussions, the decision was made to maintain primary access to the mine via surface decline and allow for a larger diameter and deeper fresh air shaft compared to the PFS for a number of reasons:

- Reduction of project construction risk.
- Better reflect the relatively small tonnages of daily material movement.
- Highlight the fact that the deposit is shallow compared to similar properties.
- Allow for additional airflow to support the R840W zone.

- Simplify the overall mine design.
- Simplify the day to day operations of the mine as a decline was viewed as less constrictive than a vertical shaft.

The following key points were decided upon by the FS Trade-off Study team:

- In order to de-risk the project, the surface fresh air shaft will be designed at a diameter and depth, that if required, will allow it to function as the primary material handling system, fresh air intake and transport personnel/materials.
- The surface decline will be removed from the critical path for the initial underground project construction period. This involves sinking the fresh air shaft to its final depth and then lowering the required underground development fleet into the mine through the fresh air shaft. Once the shaft station has been developed, the lateral development fleet can complete underground development from the fresh air shaft and develop key infrastructure and access prior to the completion of the decline. During this period, all waste development material will be hoisted through the fresh air shaft utilizing the temporary headframe and hoist.
- The surface decline excavation will be de-risked by reducing the overall profile to a minimum feasible cross-section required for mobile equipment clearance and egress.

These strategies work together to mitigate the construction risk of the surface decline through the weak overburden.

16.5.2 Mine Access and Development

Mine access will be established through use of a decline and two ventilation shafts. The primary access throughout mine life will be the decline, however during the initial construction period access to mine development will be limited to the ventilation shafts until the decline is fully constructed and connected to the lateral development on the 400 level.

16.5.2.1 Box Cut and Portal Excavation Methodology

To establish mine access and construction of the portal, a box cut will be excavated to establish the portal headwall location. The box cut excavation requires that the area has been dewatered by pumping the water table below the floor of the box cut. When the local dewatering has been confirmed the surface earthworks for the box cut can begin. In order to improve the dewatering timeline and reduce the inflow of water to the decline area, a cut-off slurry wall is planned to be installed prior to excavation of the decline. The cut-off slurry wall will assist in reducing overall flows reporting to the decline area during construction.

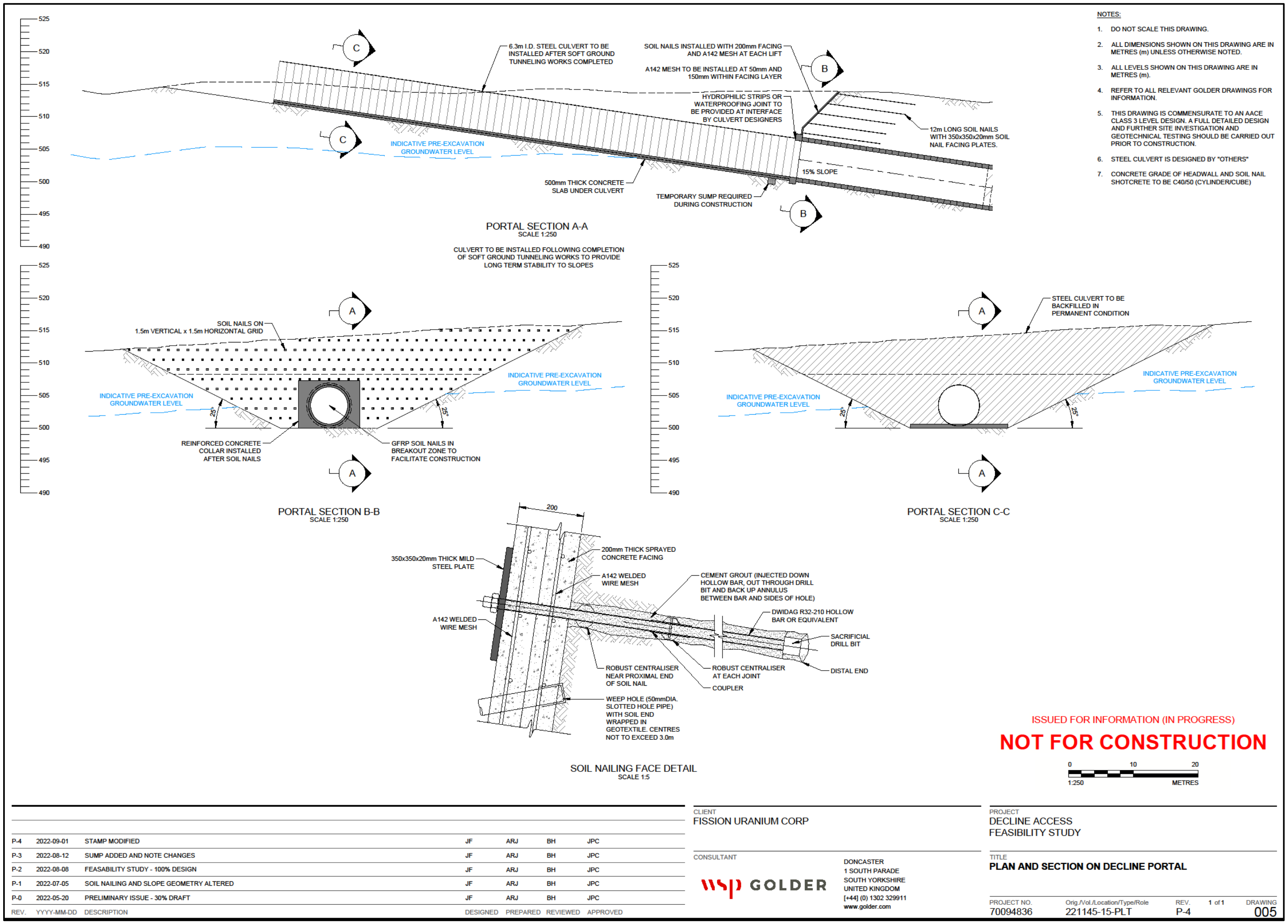
The box cut side walls will be shallow and will not require support or conditioning. The headwall must be supported by a ground support system that consists of wire mesh screen, reinforced shotcrete, and soil nails. The excavation and the establishing of the headwall must be done in 1.5 m lifts, and each lift of the headwall must be fully supported before beginning the next lift.

As the box cut is developed, the decline roadway will be created. The roadway sub-base will be sourced from excavated aggregate material. The roadway will be poured in lifts to reach the complete designed thickness of 500 mm. The roadway concrete will be formed and poured after the completion of the box cut as haul trucks need to access the box cut via the roadway base. Once the box cut is complete, the roadway can be formed and poured starting at the portal and working up grade.

As the box cut is excavated and the headwall is on the last lift, the designed portal entrance will be marked on the headwall and preparation for collaring can commence.

The portal collar will be established on the headwall of the box cut starting with the establishment of the tunnel shield. A launch frame will be built behind the tunnel shield anchoring into the concrete slab. The launch frame will provide a rigid backstop that the tunnel shield can press against to propel itself into the headwall to begin the tunnel excavation.

A box cut culvert liner will be installed on the concrete road which leads from the top entrance of the box cut to the portal entrance. This liner will not be installed until after the decline excavation (including the safety bays) is complete. A scissor lift, telehandler, and other surface equipment will be used to install the modular pieces of the culverts and they will be bolted into place. When the culvert is complete, the material from the overburden stockpiles will be backfilled in compacted lifts around the completed culvert. The culvert is planned to be hydrostatic and the operation will benefit from the culvert installation as the mobile equipment will not be required to navigate steep or slippery ramp gradients and the box cut slopes will not need to be maintained over the life of the mine. Figure 16-7 shows the box cut and portal headwall excavation design and associated ground support.



Source: Golder, 2022

Figure 16-7: Surface Decline Box Cut and Portal Excavation Design

16.5.2.2 Decline Excavation Methodology

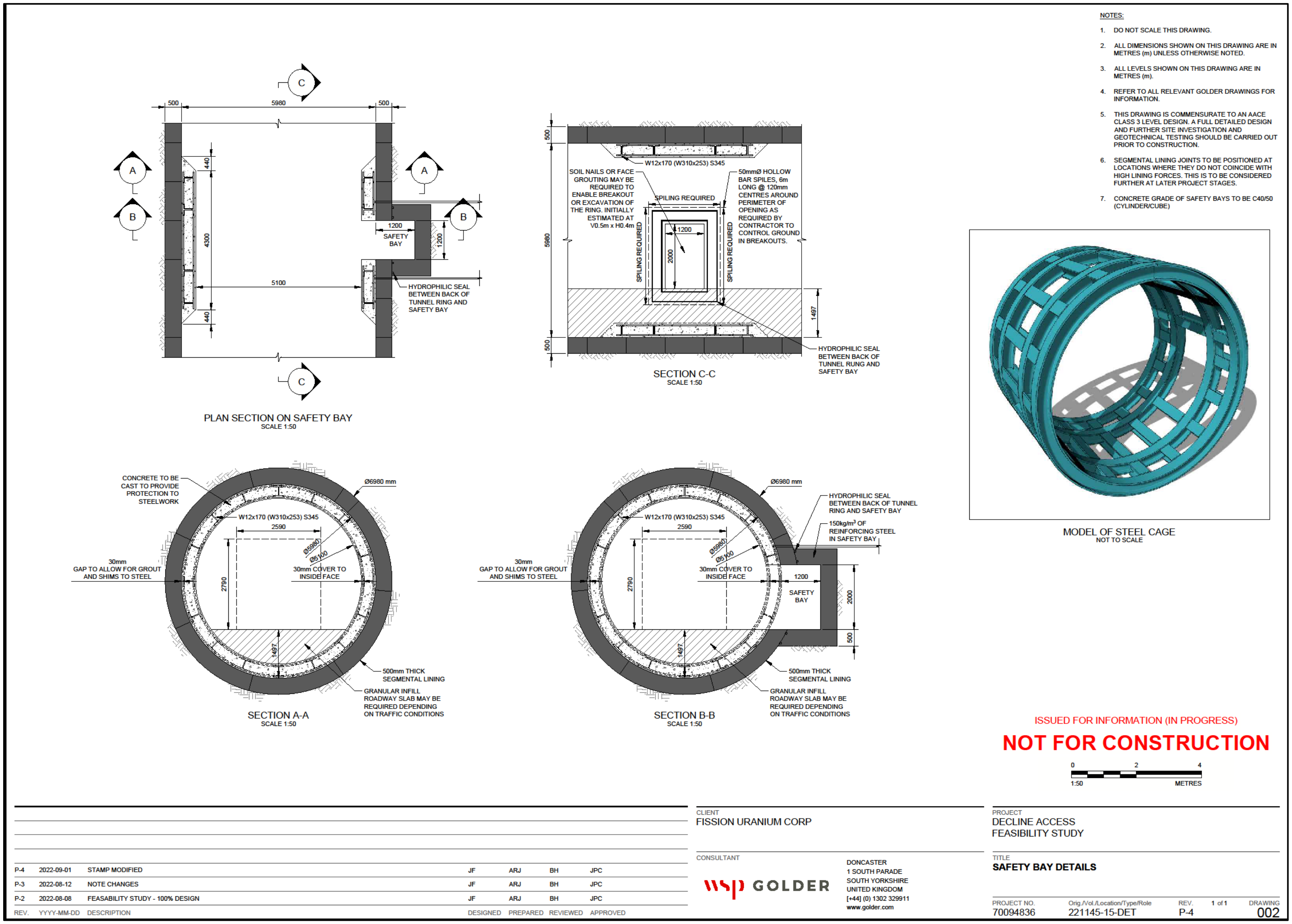
The decline is planned to be excavated using a specialized soft and hard ground method called a tunneling shield. This excavation method will require the ground water table to be lowered to below the tunnel horizon so that the local water pressure is at, or near, 0 kPa. The tunnel shield will advance from the launch frame into the host sands using hydraulic jacks. The tunnel liner segments will be erected within the shield as it advances. The material that is generated by the excavation will be hauled from the back of the tunnel shield by load haul dump (LHDs) and taken to the waste stockpile on the surface. The concrete tunnel liner segments will be brought into the tunnel by LHD and will be put into place by a segment handling arm within the tunnel shield. As the shield advances, the completed liner rings will no longer be within the tunnel shield and will be left in place to support the overlying strata. Following the advancement of the tunnel shield, grout will be pumped to fill the void behind the tunnel liner. The pre-cast segments are designed to incorporate gaskets and other hydrophilic seals so that following construction the tunnel is fully hydrostatic. The mine services consisting of ventilation duct and power cable will be advanced behind the tunnel shield as the tunnel is excavated. Figure 16-8 shows a typical tunnel shield excavator used in similar applications.

Source: Mining Plus, 2022

Figure 16-8: Typical Tunnel Shield Equipment Setup for Soft Ground Excavation

The tunnel shield will advance along the decline through the soft sections excavating material and erecting tunnel liners. When the basement unit is encountered, a cutting head will be required at the face of the tunnel shield to excavate the harder basement unit.

When the decline has reached the end of the transition zone, the safety bays will be installed. The safety bays require an inner steel ring to be built in place at each safety bay location. After the steel ring is built, spiling bars are installed and grouted in place at the safety bay location. This will support the sand unit before the liner segment is removed. Once the existing liner segment is removed, new safety bay segments will be installed. Once complete, they will be grouted in place. Figure 16-9 shows the decline cross-section and long section through a typical safety bay construction area.



Source: Golder, 2022

Figure 16-9: Decline – Standard and Safety Bay Cross Sections

16.5.2.3 Primary Ventilation Shafts

The construction of an FAS and a primary EAS is planned with the FAS having a modular escapeway installed as secondary mine egress (Thyssen Mining 2022). The FAS will contain the mine services that will be mounted to the shaft liner. The mine services include two sets of DN250 mm freeze pipes that will deliver and return brine for underground crown pillar freeze circuit. The EAS will contain only the mine services that will be used for sinking. They will be stripped from the shaft after the sink is complete, to be reused elsewhere on the project.

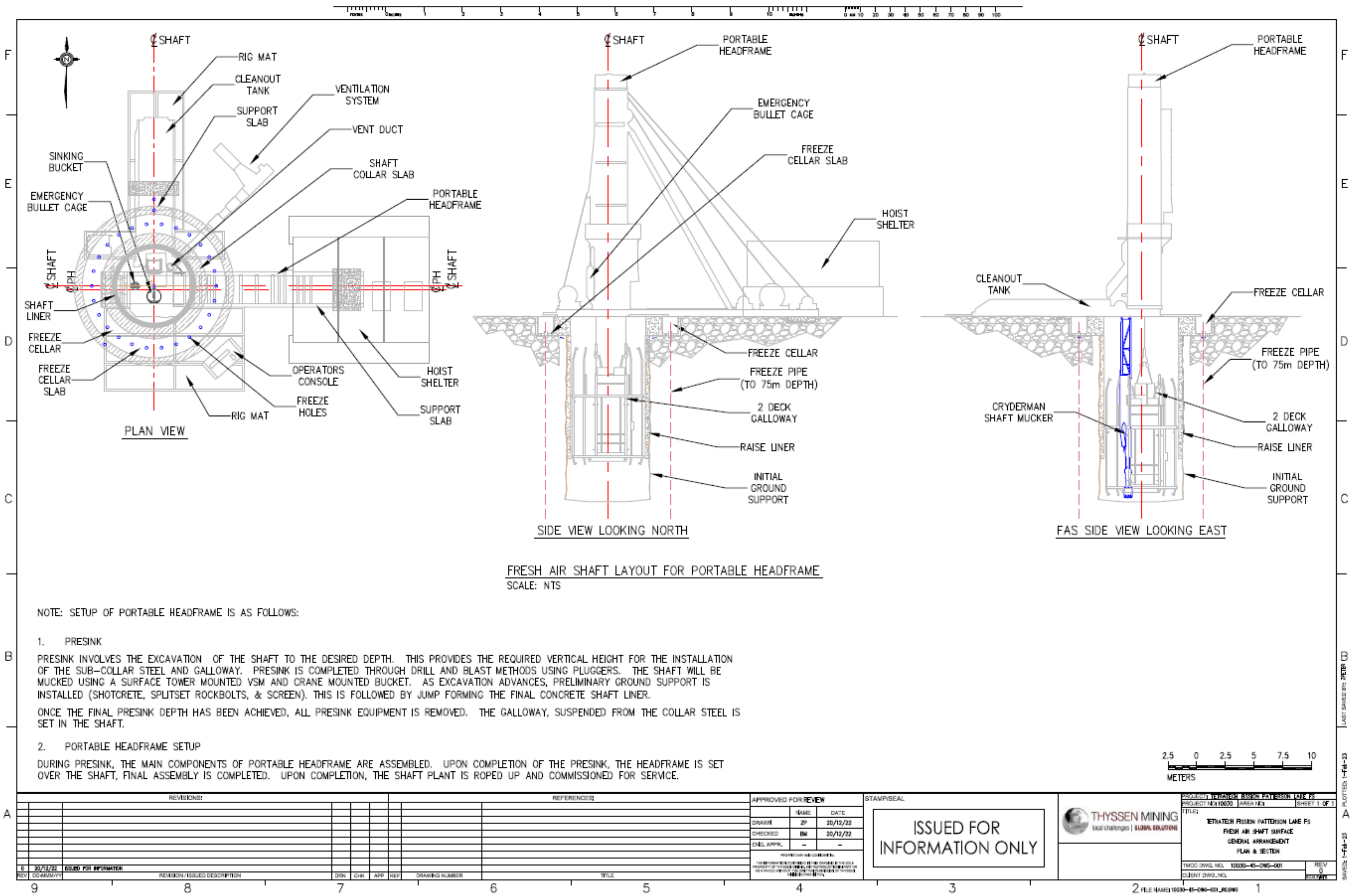
The FAS will be constructed to a finished internal diameter of 6.5 m and a depth of 120 m with a concrete hydrostatic liner extending to the 400 level. The nominal concrete liner thickness will be designed to 460 mm. The main lateral level will be 'stubbed in' at the 400 level and the sink will continue to the 120 m depth. The sinking Galloway will be 'parked' below the 400 level where it will remain *in-situ*. The bottom of the shaft will act as a sump over the LOM. The top of the shaft will be capped and intake fans inline with natural gas heaters will be installed on the ventilation plenum to supply tempered air to the mine workings.

The overall FAS design allows for the optionality to accommodate shaft production hoisting infrastructure to reduce the construction risk associated with the main decline. The shaft diameter and ventilation plenum design at the collar has been completed on the basis that it would be able to provide adequate air flow as well as contain production hoisting infrastructure that would meet the needs of the operation.

The exhaust shaft will be constructed to a finished diameter of 5.0 m and a depth of 100 m with a concrete liner extending to the full depth of the shaft. The nominal concrete liner thickness will be designed to 250 mm. The finished shaft will later be outfitted with a cap and ventilation fans and infrastructure to pull air from the underground and down the FAS. Both shafts will use a sinking or temporary headframe and a Galloway to do the shaft sinking. The headframe and Galloway will remain in place until all of the sinking activities are completed.

16.5.2.4 Ventilation Shaft Construction Methodology

The FAS and the EAS will be excavated in series with the FAS excavation occurring first followed by the EAS. The sinking plant for the FAS will be set up and commissioned following completion of a 15 m pre-sink and artificial ground freezing program, after which a sinking headframe will be built at the FAS collar. The contractor may use a portable sinking headframe if appropriate. Artificial ground freezing is a common technique used to control water inflow when developing shafts through saturated overburden material, the ground freeze will remain intact for the duration of the shaft construction. The FAS will be excavated using a conventional blind sinking method and will utilize a poured concrete liner with an inner diameter of 6.5 m. Figure 16-10 shows the planned sinking plant infrastructure to be used for the FAS, the EAS will also use a similar set-up.

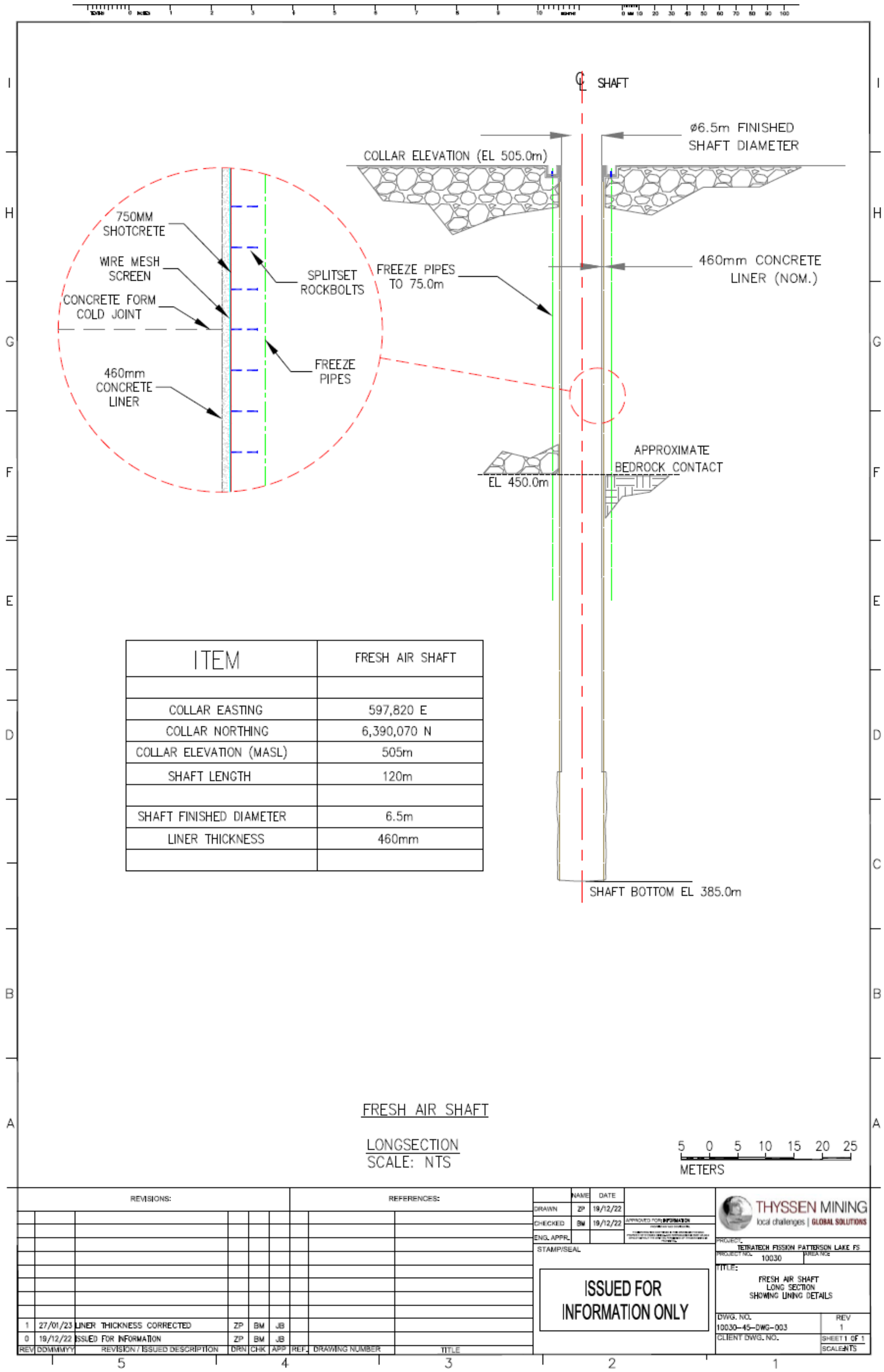


Source: Thyssen, 2022

Figure 16-10: Shaft Layout for Portable Headframe During Sinking Operations for FAS

The depth of the FAS is relatively shallow with a planned depth of 120 m. During shaft sinking, stub outs for shaft stations will be completed at the 400 level. Initial station collaring will be completed using the shaft jumbo and/or jackleg. A small skid steer will be lowered to station level and used to muck out the heading and load buckets. The depth of collaring at each station will be approximately 25 m. The FAS will have a shaft station stub excavated at the 400 level underground which will initially be driven 25 m from the station during the sinking phase. The future lateral development during project construction will take place from this shaft station. Waste rock handling infrastructure will be established at the shaft station to support skipping the muck that is generated from the lateral development.

Figure 16-11 shows the long section of the FAS at the completed depth of 120 m.



Source: Thyssen 2022

Figure 16-11: FAS Long Section Showing Liner Details

The lateral development will target establishing a connection to the surface decline excavation as well as other key infrastructure areas on the 400 level. The muck will be hoisted to surface via the sinking buckets which will run on crossheads attached to the Galloway ropes. Following the lateral development connection to the surface decline, the sinking Galloway will assist with the installation of a modular escapeway and any required services within the shaft. The Galloway will be chaired below the shaft station temporarily during the lateral development on 400 level and then permanently following the manway installation, grouting, and permanent services installation.

For the FAS, the permanent mine services will be:

- Brine freeze pipes: 10 inch pipes, flanged connection, covered in 4 inches of insulation x 2
- Backfill lines: 8 inch x 2, backfill lines will also serve as contingency water lines if required
- Mine water supply line: 6 inch
- Mine water discharge line: 6 inch x 2
- Mine water ring collection: 4 inch
- Electrical power and ground cables, communication cables, blast cable
- Secondary Egress Self Contained Manway

Matcher grouting of the FAS concrete liner will be conducted during the permanent services installation following a 220 day thaw period. The grouting behind the shaft liner will assist with reducing water inflows and ensure the stress is adequately redistributed on the concrete liner.

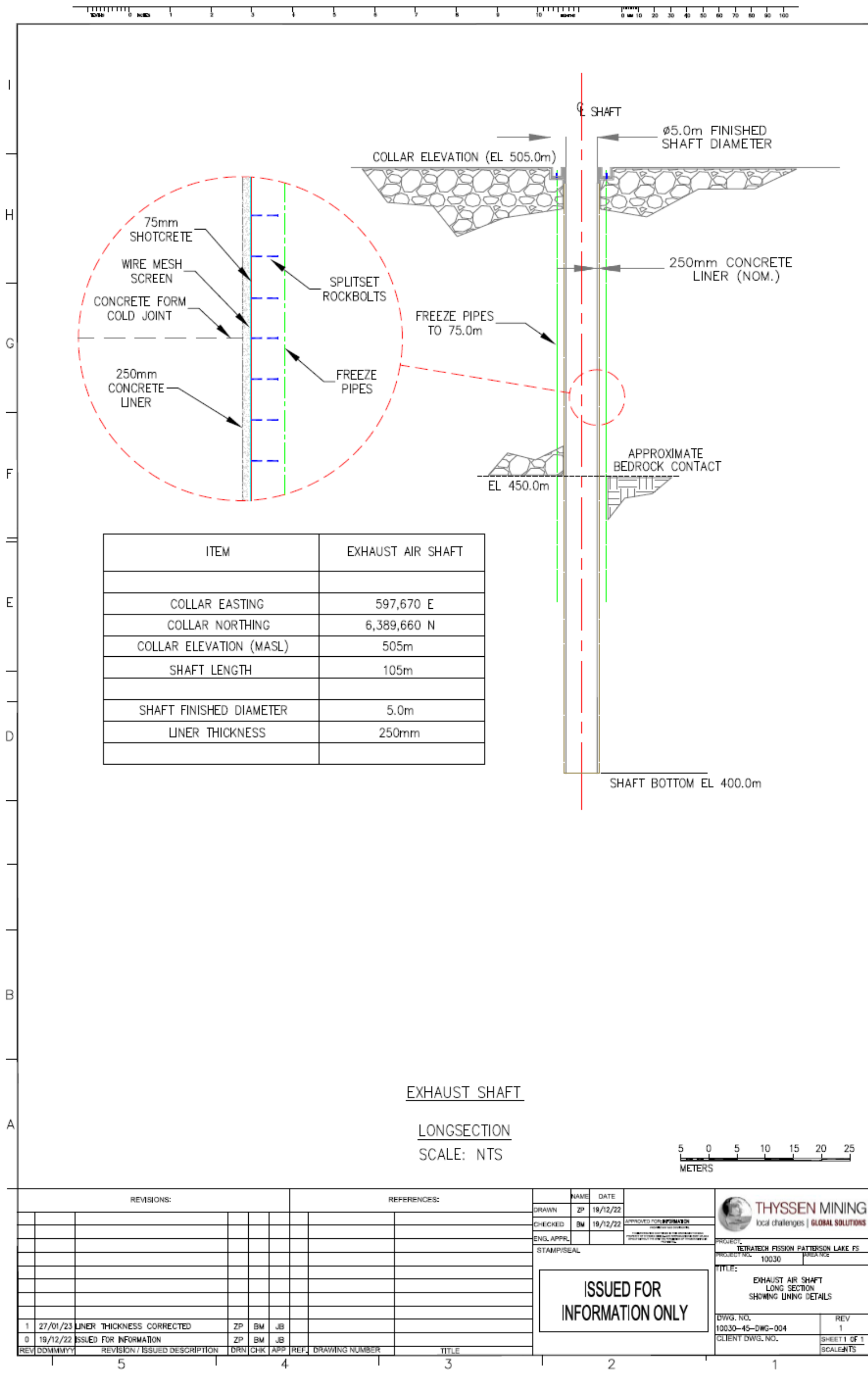
When the matcher grouting of the FAS liner is complete, the sinking plant infrastructure will be moved to the EAS where it will be set up and commissioned to begin the blind sink of the EAS.

The EAS will also be sunk via traditional blind sinking which will utilize artificial ground freezing. The shaft sinking will be completed to a depth of 105 m. The exhaust shaft does not require a shaft station as the shaft bottom will be at the 400 level. At the shaft bottom there will be a stub excavated where the future lateral development will later break into, rather than development breaking directly into the shaft where there is a risk that the shaft liner could be damaged.

For the exhaust shaft, the temporary mine services will be:

- Compressed air 6 inch pipe
- Mine water supply 6 inch pipe

Matcher grouting in the EAS will be conducted following a 220 day thaw period. The matcher grouting will be completed from a purpose-built platform as the headframe will have been demobilized prior to the grouting operations. The reason for this is that it allows the surface exhaust air ventilation system to be commissioned earlier in year -1 of construction. Figure 16-12 shows the long section of the EAS at the completed depth of 105 m.



Source: Thyssen 2022

Figure 16-12: EAS Long Section Showing Liner Details

Following completion of the matcher grouting, the remaining shaft sinking infrastructure at the EAS will be removed and packaged for demobilization. This milestone will mark the end of the shaft sinking and decline development contractor scope.

16.5.2.5 Secondary Egress Escapeway

A modular escape way will be installed in the FAS. The raise will be installed following the shaft sink and 400 level lateral development. It will provide a secondary means of egress from the mine. The escape way will be installed from the top down and it will be anchored to the shaft liner. The modular pieces will be lowered from surface and the pieces will be assembled and anchored to the shaft wall from the Galloway.

16.5.3 Lateral Development

Lateral development will be mined using a mix of conventional drill and blast and mechanical excavation methods, with the crown area and mineralized development being completed by roadheader. The primary reason for using conventional drill and blast in waste development headings instead of roadheader is due to the high abrasivity of the waste rock mass. The profile and development types are shown in Table 16-10 and Table 16-10. The development profile varies for purpose built underground infrastructure such as the permanent refuge station, main pump and dewatering stations and compressor bay.

Table 16-9: Lateral Development Profile Type with Associated Development Types

Profile Type	Development Type
6.0 mW x 5.0 mH	Foot Wall Drive
5.0 mW x 5.0 mH	Ramps, Remucks, Truck Loadout, Sump, Sub Station, Ore Lifts, Attack Ramp, Magazine, RAD, FAD,
4.5 mW x 4.5 mH	Level Accesses, HWD, Cross-Cut
4.0 mW x 4.5 mH	Ore Drives

Table 16-10: Lateral Development Meters by Development Type

Development Type	m
Ramps	3,092
Remuck	711
Level Accesses	856
Truck Loadout	1,504
Sump	935
Main Sump	388
Sub Station	112
HWD	4,340
Foot Wall Drive	744

table continues…

Development Type	m
Ore Drive	10,431
Ore Lift	4,162
Cross-Cut	1,426
Attack Ramp	595
Magazine	36
RAD	4,637
FAD	1,792
Washbay, Permanent Refuge Station, Compressor Bay	82
Pump Station	85
Total	**35,928**

16.5.4 Vertical Development

Vertical development will consist of the primary ventilation shafts and internal FAS and EAS. The primary shafts will be mined via a method of conventional blind shaft sinking as previously described. Internal raises will be mined via a traditional long hole drop-raising technique. Table 16-11 and Table 16-12 provide a summary of vertical development dimensions and quantities.

Table 16-11: Vertical Development Profile and Description

Profile Type	Development Type
6.5 m Diameter	FAS
5.0 m Diameter	EAS
3.0 m x 3.0 m	Internal Raises

Table 16-12: Vertical Development Meters by Development Type

Development Type	m
FAS	140
Return Air Shaft	97
Internal Raises	759

16.5.4.1 Internal Egress and Escapeways

The internal raises mined will be used for both ventilation and escapeways for secondary egress. The escapeways will be a modular fully enclosed structure inclusive of ladders and internal landings. The structure will be installed within the raise and will allow for mine ventilation to travel through the enclosed escapeway.

A long-hole drill will be used to drill the full length of the raise, the raise will then be blasted in sections, starting at the undercut and progressing towards the overcut. Muck under the blasted raise will be drawn down for each blast.

Once the full length of the raise has been safely blasted, the raise will be surveyed and assessed for any potential tight areas prior to installation.

Once the working platform is set up the enclosed escapeway will be installed from the overcut of the raise in pieces. Strand jacks will be used to lower the escapeway pieces into place in the raise. The escapeway will be subsequently lowered with each additional piece until reaching the undercut of the raise. This method does not require conditioning or a working platform to install ground support in the raise.

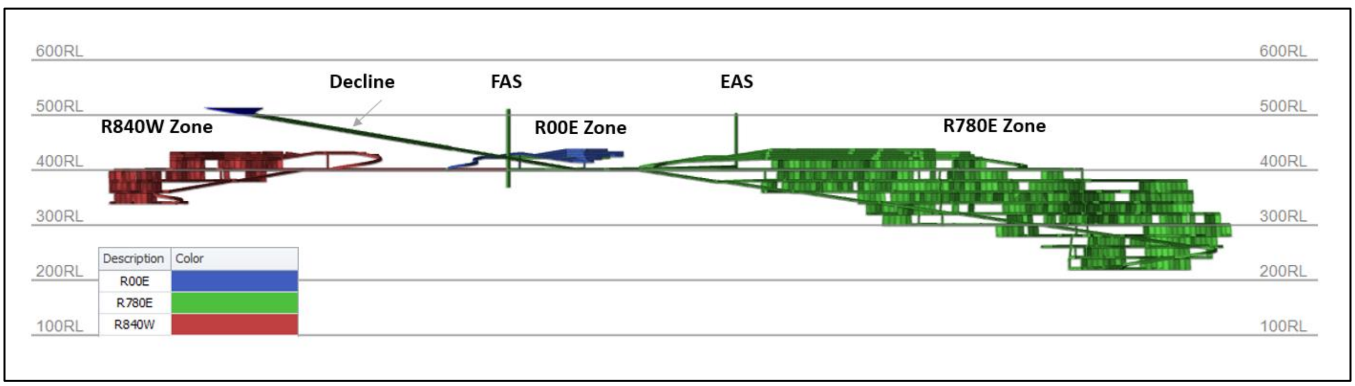
16.6 Mining Method and Sequencing

16.6.1 Mining Zones

The mine design includes the following primary zones:

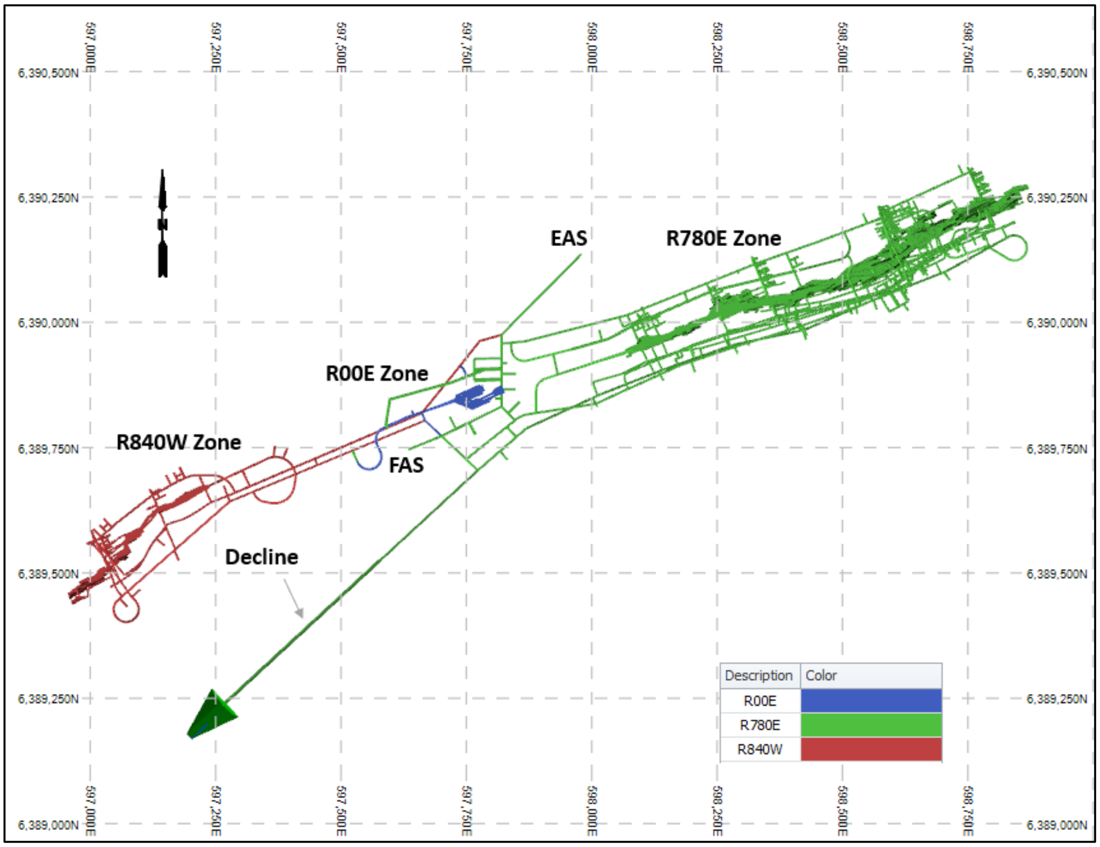
- R780E
- R00E
- R840W

Figure 16-13 and Figure 16-14 show a longitudinal section and plan view of the mine plan with the three main zones and the associated mine access development.



Source: Mining Plus 2022

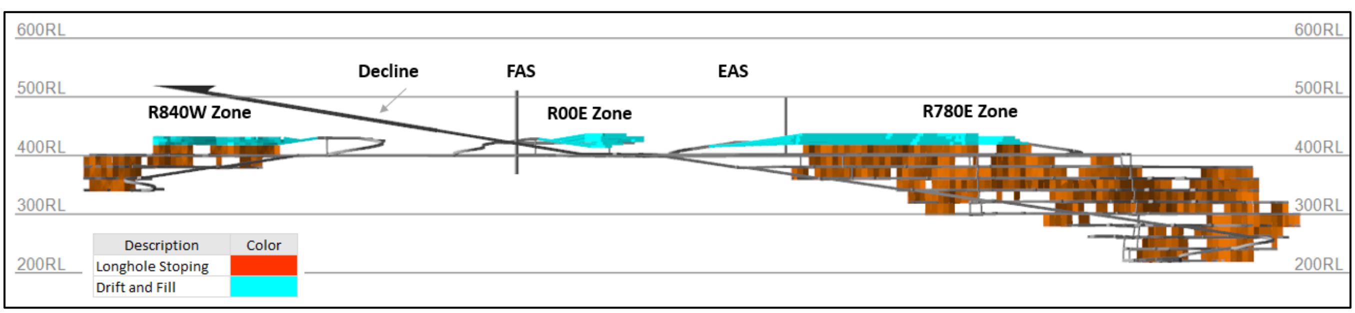
Figure 16-13: Longitudinal View of Primary Mine Zones and Mine Access (looking northwest)



Source: Mining Plus 2022

Figure 16-14: Plan View of Primary Mining Zones and Access

The primary mining method selected is longhole stoping using longitudinal retreat. The crown pillar area which comprises 15-30 m of the R840W and R780E and entire R00E are planned to be extracted via mechanized D/F methods due to geotechnical constraints and proximity to the overlying basement contact. Figure 16-15 shows the mine plan with a legend by mining method.



Source: Mining Plus 2022

Figure 16-15: Longitudinal View of Mine Design Showing Mining Method (looking northwest)

16.6.2 Longhole Stoping

The primary mining method for the underground will be longhole retreat mining in a longitudinal orientation. The mining will progress from the bottom levels to the top, and from the southwest to northeast. Mining is planned at nominally 1,000 t/d ore. The overcut and undercut levels will be driven at 20 m sub-level spacing and are planned to be mined using mechanical excavation with a roadheader where ground conditions permit. These drives are located in the Resource unit and geotechnical testing has indicated that there is a wide range of rock strengths expected with a UCS value of 17 MPa used for design purposes. Mining within the Resource unit will be primarily completed using mechanical excavation to reduce ground control issues and reduce mining dilution related to drill and blast techniques. A jumbo will be used in locations where the Resource unit is more competent and will benefit from drill and blast development techniques.

Stope lengths are 10 m in strike and have variable widths ranging from a minimum of 4 m to maximum 12 m inclusive of footwall and hanging wall dilution. The dimensions used are within BGC's recommended geotechnical parameters. Stopes were designed using MSO and the parameters used to create the stopes are outlined in Table 16-13 below. The values used in the COG calculation were based off benchmarking, previous studies on PLS, and any data updated and/or calculated during the feasibility.

Table 16-13 MSO Input Parameters – Stopes

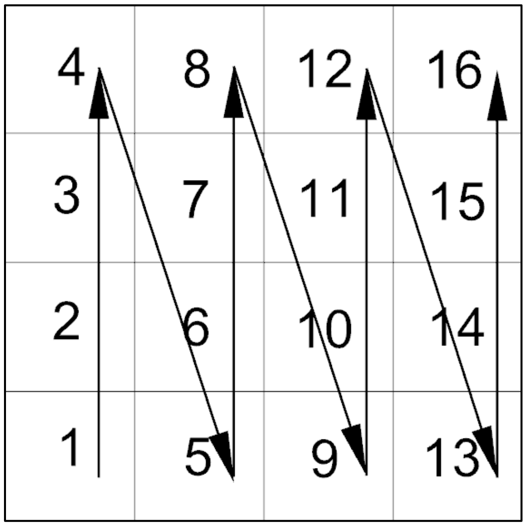
Parameter	Value	Unit
Stope Height[1]	20	m
Stope Length[2]	10	m
Minimum Stope Width[3]	4	m
Maximum Stope Width[3]	12	m
Minimum Stope Dip	75	°
Optimization Orientation	-23.8	°
Initial COG	0.20	%

Notes:
1. Stope heights are based on floor-to-floor elevations.
2. Stope lengths are assumed to be in the longitudinal direction along strike.
3. Min/max stope widths include 1 m of dilution from both the hanging wall and footwall of the stope for a total of 2 m. The MSO stope shapes generated that were >12 m width were post-processed into adjacent stope shapes that conformed with geotechnical guidance.

Stopes were initially generated in MSO software at a 0.20% U_3O_8 cut-off as a first pass to assist with mine design and avoid sterilization of near cut-off mineralization. From this initial stope set, stopes over 0.25% U_3O_8 were filtered and selected for inclusion in the design based on their location and continuity with adjacent mineralization. The 0.25% U_3O_8 is approximately the break-even COG. The stopes were then evaluated for additional modifying factors such as external dilution inclusion based on their location relative to geological contacts and exposure to previously backfilled walls during extraction.

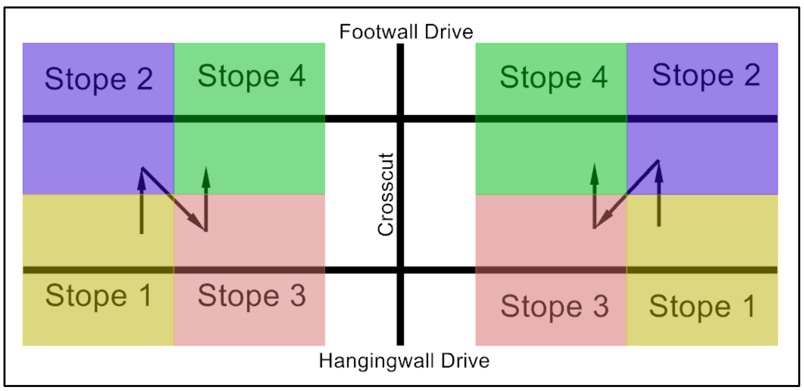
Stopes will be drilled off using a in-the-hole (ITH) longhole drill. ITH drills produce more accurate holes than top hammer drills, reducing unintended dilution and ensuring consistent blasting of stopes. Blastholes will be 102 mm (4 inches) in diameter and the drill will be equipped to collect drill cuttings to reduce personnel exposure to mineralized material. Blastholes will be loaded with Ammonium Nitrate Emulsion, this is due to the emulsion's ability to resist desensitization from water inflows. Blastholes will be primed with nonelectric detonators with standard delays hooked to a blast line which will be connected to a remote initiation system for secured and safe blasting.

Sequencing is shown in long view in Figure 16-16 with the order being a bottom-up sequence over four levels. Figure 16-17 shows the sequencing in plan view with mining starting at the end of the ore drive and retreating to the access crosscut starting with stopes nearest to the HWD.



Source: Mining Plus 2022

Figure 16-16: Example of Stope Panel Sequencing – Long Section View



Source: Mining Plus 2022

Figure 16-17: Typical Level Stope Sequencing – Plan View

16.6.3 Mechanized Drift and Fill

The upper portion of the deposit will be mined using roadheader based mechanized D/F. This is due to geotechnical constraints and proximity to the basement contact which is covered in the crown pillar section of 16.2. Mining within the crown pillar area of the R780E and R00E zones will require a bulk freeze using refrigerated brine solution to be formed prior to extraction. The artificial ground freezing will create a bulk freeze that targets the first 10 m of the overlying overburden material and 15 m of the crown pillar vertically. The freeze is planned to extend out laterally to the geological contact called the MSZ. Each drift will be backfilled and cured before mining is allowed beside or above, which will limit the maximum open span to 5 m. D/F shapes were generated using MSO software using the spatial parameters presented in Table 16-14 below.

Table 16-14 Input Parameters – Mechanized D/F

Parameter	Value	Unit
Drift Height	5	m
Drift Width	5	m
Pillar	0	m
Dip	90	°
Maximum Waste	30	%
Initial COG	0.2	%

16.6.4 Ventilation

The PLS ventilation system will be established by use of an FAS, EAS, and decline portal. Air will down-cast through the FAS, and up-cast through both the portal and EAS as exhaust. The overall intake for the system is designed to be 350 m^3/s.

Table 16-16 shows the build-up of the required airflow for the project during steady state operations. Airflow per heading is built around the assumption that there will be continuous instances of two pieces of equipment in the same heading at the same time and are built around selected equipment engine sizes. Airflow is distributed between the various locations in the mine relative to the anticipated active headings or infrastructure in each zone. As airflow that comes in contact with mineralized zones and dirty water sumps cannot be reused, there is a limit on the number of active headings and airflow must be managed diligently by the use of the bulkhead regulators and ventilation on-demand (VOD) system. Saskatchewan Mines Regulation cS-15.1 Reg 8 s16-8 stipulates that the minimum required airflow per rated kilowatt of a diesel engine underground is 3.8 m^3/min, which is calculated to be 0.063 m^3/s. Table 16-15 shows the build-up of the required airflow for a single heading ventilation circuit in a scenario with two pieces of production equipment operating.

Table 16-15: Single Heading Ventilation Circuit Buildup

Equipment Type	Engine Size (kW)	Required Airflow (m^3/s)
Truck	235	14.9
LHD	160	10.1
Total	**--**	**25.0**

Table 16-16: Ventilation Requirements Buildup

Work Headings	Quantity	Airflow per Heading (m^3/s)	Total Airflow (m^3/s)
R780E	**8**	**75**	**150**
Development (waste)	2	25	50
Development (ore)	2	25	50
Production	4	25	100
R840W	**2**	**75**	**50**
Development (waste)[1]	0	25	0
Development (ore)	1	25	25
Production	1	25	25
R00E or R780E D/F	**0**	**25**	**0**
Development or Production[1]	0	25	0
Surface Decline	**2**	**15**	**30**
Trucks in Decline	2	15	30
Idle Flow (0.5m/s * 5x5 Arch)	**15**	**52.5**	**63**
Fixed Facility (mag, office, other)	2	12.5	25
Freeze Drive Level	1	12.5	12.5
Intake/ Exhaust Airdoor interface	1	12.5	12.5
Contaminated Water Sumps	1	12.5	12.5
Inactive Ore Drives (unsealed)	10	2.5	25
Leakage	**20%**	**--**	**58**
Total	**--**	**--**	**350**
FAS Flow	--	--	350
EAS Flow	--	--	320

Notes
1. Required airflow shown as zero due to limit on total development and production areas online at the same time.

Due to the location of the mine (Northern Saskatchewan, Canada), the ventilation system requires a heating system that can raise the air temperature by a maximum differential of 48°C (-43°C to +5°C). For this purpose, a 22.5 MW direct fire heater fueled by natural gas has been selected. It will be integrated with the primary intake fans. Table 16-17 shows the expected monthly consumption of LNG during steady state operations. The LNG will be supplied to the primary intake fans by a pipeline from the main power plant area. Heating is typically required between the months of October and April.

Table 16-17 Primary Ventilation Heating Requirements – LNG Consumption by Month

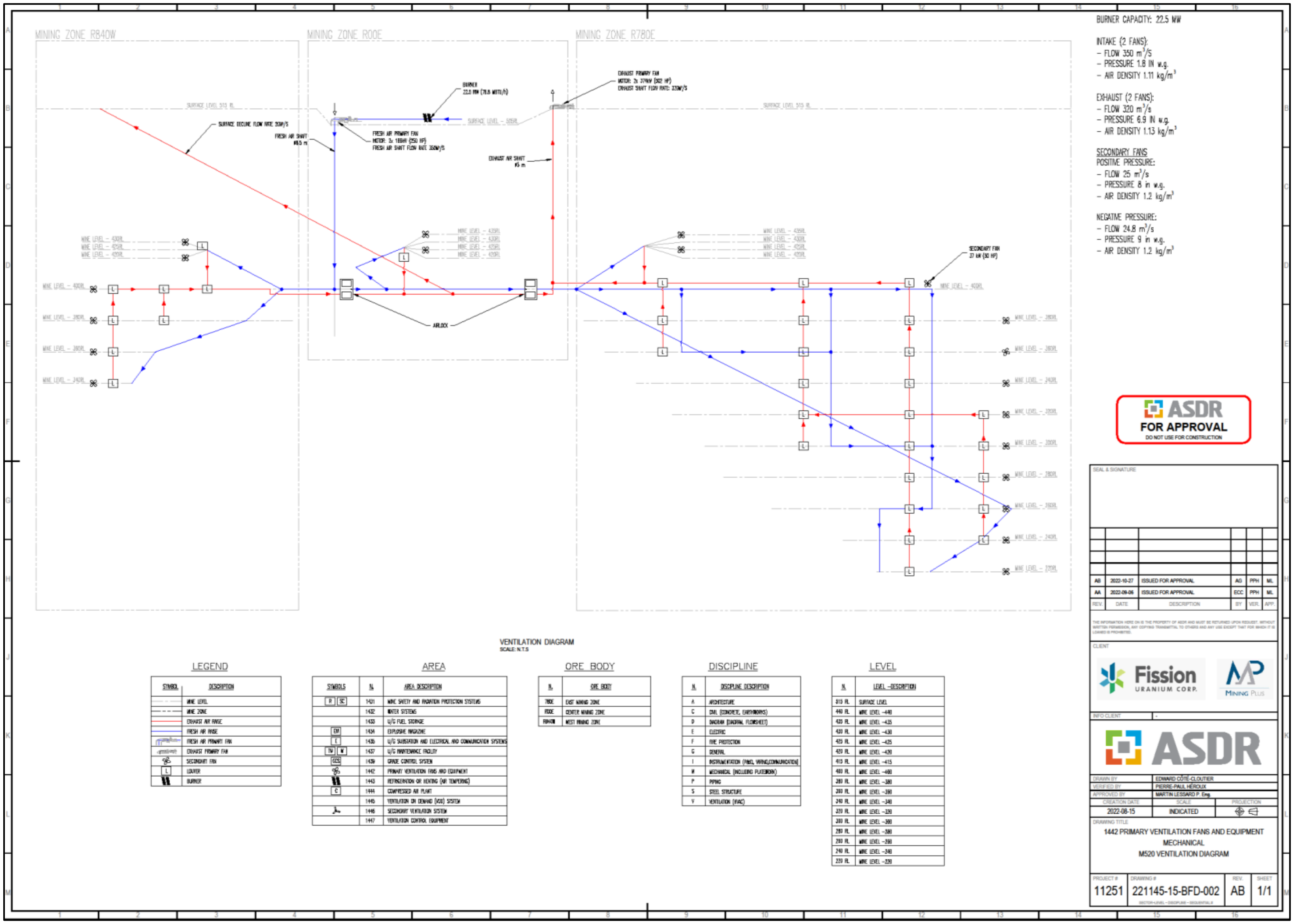
Description	Units	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Mean Temperature[1]	°C	-21	-17	-11	-1	6	13	17	14	8	0	-9	-18
Airflow Volume	m^3/s	350	350	350	350	350	350	350	350	350	350	350	350
Heat Required	MBTU	28,258	21,329	17,163	6,030	--	--	--	--	--	5,198	14,520	24,418
LNG	L	1,344,045	1,014,460	816,331	286,823	--	--	--	--	--	247,229	690,631	1,161,384

Notes
1. Reference Station: Cluff Lake Weather Station, Saskatchewan, Canada

The primary ventilation system intake fans are low duty and act more like a blower to add pressure to the system to make the surface decline exhaust. This fan set-up is envisioned as a parallel two fan set-up on surface drawing air directly through the mine air heaters.

The primary system exhaust fans are envisioned as a parallel two fan set up on surface. These fans carry the primary ventilation load and ensure that the overall system remains under negative pressure to remove the potential for recirculation in the underground workings.

The primary ventilation system is held under negative pressure by the surface exhaust fans. Flow to each level is controlled by louvers installed in each of the bulkheads on the return side of the production level cross-cuts. The proposed size of the louvers is 4 m^2 equivalent (2 m x 2 m). The louvers are automated and will be controlled remotely at the mine control centre. A full schematic of the ventilation system is shown in Figure 16-18.

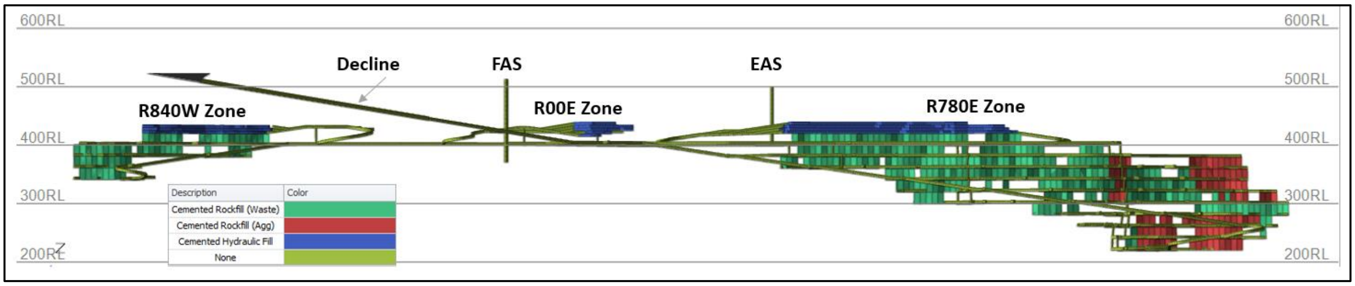


Source: ASDR, 2022

Figure 16-18: Primary Ventilation System Schematic

16.6.5 Backfill

Backfill is an integral part of the production cycle which will allow for a higher overall resource extraction. The engineered backfill methods chosen for the project are CRF and CHF. The use of cement in varying quantities will allow for mining in close proximity to backfilled voids while controlling dilution and rock mass stability. The CRF material will be composed of a blend of development waste rock and locally available sand using a 75:25 ratio. The CHF will be composed of locally available sand. CRF is planned to be used as backfill in stopes while CHF will be used as backfill in the D/F excavations located in the crown pillar area. The backfill material balance completed for the project shows that, even with the use of blending, there is a deficit of waste rock from underground sources to satisfy the backfill needs in the stoping areas. The waste rock stockpile is consumed by the end of year 8 and the remaining stopes will require aggregate materials that are sourced from a nearby location north of the TMF. The aggregate source near the TMF will have been initially excavated during the TMF construction and will have adequate supply to meet the backfill needs of the operation. Figure 16-19 shows a longitudinal view of the planned backfill method by area with CRF using local aggregate sources identified as a different backfill type.



Source: Mining Plus, 2022

Figure 16-19: Longitudinal View of Mine Design Showing Backfill Method (looking northwest)

The aggregate and sand for the CRF will be produced locally in a modular crushing and screening plant consisting of a jaw crusher and a single cone in a closed circuit. The plant will produce two products: a coarse aggregate comprising -75 mm to +10 mm, and a fine aggregate comprising -10 mm. The CRF will be batched in a pre-engineered packaged CRF plant consisting of three aggregate bins (one for local sand, one for coarse aggregate, and one for fine aggregate), a single cement silo and dosing system, and a twin shaft concrete mixer. A bottom discharge in the mixer will deliver the batched CRF directly into the back of 30-t trucks for haulage underground. Due to short hauls, the CRF demand can be met with one truck full time and a second truck part time. A 20% allowance for backhaul has been included for trucks that are available to bring CRF backfill underground as part another haulage cycle such as ore or waste hauling. The use of engineered backfill instead of ROM backfill will assist controlling segregation during backfill placement and reduce the impact of dilution during extraction.

The CRF plant has been designed to allow the batching of concrete for construction as needed.

The CHF plant is designed to be integral with the CRF plant and shares facilities such as the aggregate (sand) handling, and the cement dosing system. The CHF slurry is delivered down the main decline in a 90 mm steel reinforced composite pipe (SRCP) piping. The final run into the crown pillar stopes will be serviced with high-density polyethylene (HDPE) piping. The R00E and R780E will be supplied by the CHF using the pipeline in the decline, however due to higher pressure requirements, R840W will require a temporary surface line of about 450 m (including 40 m vertical pumping head) feeding a 100 m long borehole to the R840W crown pillar area. This line will need to be buried or wrapped in heat trace to prevent freezing during the winter months.

CRF backfill will be hauled from the surface backfill plant near the waste stockpile to the underground production areas where it will be picked up by LHDs and placed into stopes. Bumper blocks will be used to control access to stope voids and once the backfill has been filled to the natural rill angle, the CRF will be placed into the stope void using remote LHD operation.

MineFill recommends the following be addressed in further design stages:

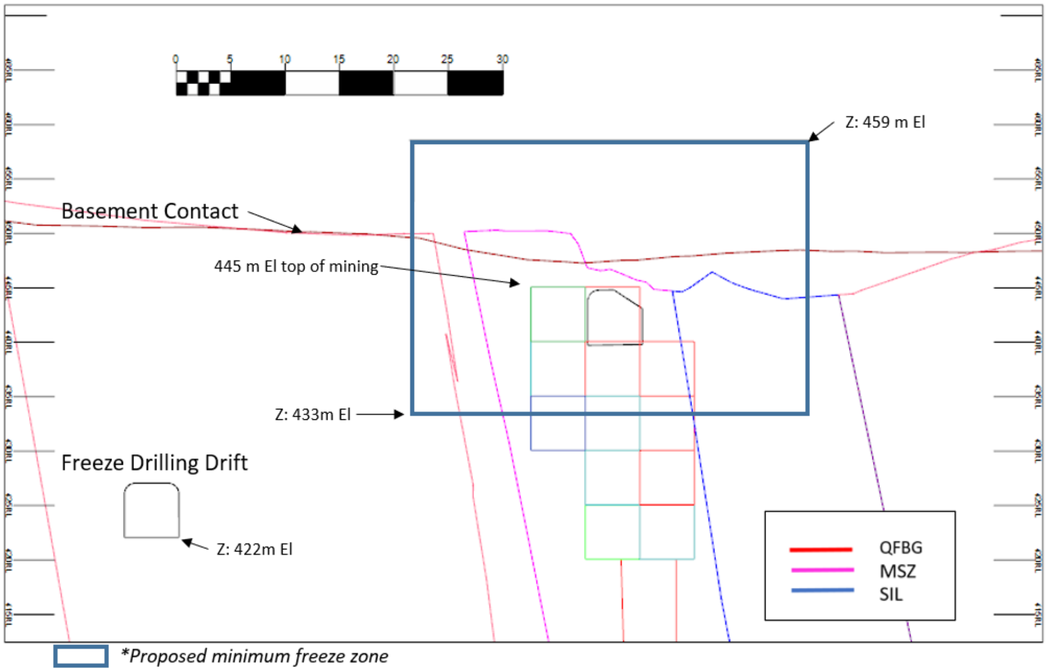
- The choice between the risks and logistics of a surface line and borehole through the sandy overburden, versus the addition of a fourth stage of centrifugal pumps and higher pipe wear, should be addressed in detailed engineering.

16.6.6 Artificial Ground Freezing

Artificial ground freezing will be used to allow for partial extraction of the crown pillar in the R00E and R780E zones. The R840W zone geotechnical analysis has shown that the area can be mined without the need for any artificial ground freeze techniques which results in improved financial performance for that zone. The artificial ground freeze design has been supported by Newmans Geotechnique Inc. (NGI), a recognized expert in industrial applications of artificial ground freezing. NGI evaluated the ground freezing needs of the project and calculated the number and spacing of the drill holes required and the resulting refrigeration plant capacity.

NGI devised a freeze layout that allows for drilling and installation of freeze circulation pipes on 4 m spacing from an underground drift on the footwall side of each orebody. Each drill station will consist of four drill holes fanned out to target the bulk freeze zone. Once the underground drilling is complete and the brine circulation network is installed, the freezing is started. The project has been designed to allow for a two-year period for each crown pillar area to circulate the brine and create a bulk freeze of -5°C or colder. The surface refrigeration plant has an installed 1,625 t capacity and would output brine at -35°C. The overall design also incorporates instrumentation and safety devices to ensure that the target freeze area is reaching design temperatures and that any intercepts of water bearing structures during drilling can be controlled.

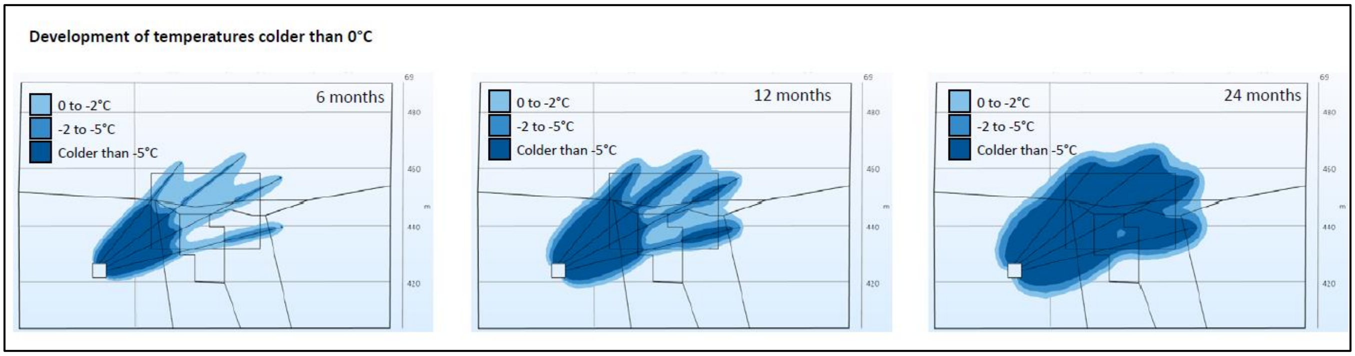
Figure 16-20 shows a typical section in the R780E crown pillar zone. The target freeze area was determined by BGC and allows for a bulk freeze to extend laterally to the geological contact called the MSZ. The bulk freeze is shown to extend 10 m vertically into the saturated overburden material and 15 m vertically down into the Basement contact. During the extraction of the upper crown pillar levels, the mining will intercept some lower freeze circulation piping, however, this can be managed operationally as the upper freeze pipes will maintain some stabilization and water barrier effect.



Source: Mining Plus, 2022

Figure 16-20: R780E Crown Pillar Section with Artificial Bulk Ground Freeze Target Profile (looking northeast)

Figure 16-21 shows a section of the R780E crown pillar and the progression of the bulk freeze over the two-year freeze timeline. The dark blue area is shown as having reached design freeze temperatures of cooler than -5 °C.



Source: NGI, 2022

Figure 16-21 R780E Artificial Bulk Ground Freeze Progression Over Two Years Section

Table 16-18 breaks down the freeze and monitoring hole drilling requirements by crown pillar area.

Table 16-18 Artificial Ground Freeze and Monitoring Holes Drill Summary

Hole Function	Quantity	Average Length (m)
R00E Freeze	176	80
R00E Monitor	8	70
R780E Freeze	504	73
R780E Monitor	16	63

16.7 Mine Productivity and Schedule

16.7.1 Shift Effective Hours

The build up for the effective hours used in unit rates for the cost model is shown in Table 16-19 along with the shift length used.

Table 16-19: Shift Effective Hours Breakdown Used for Unit Rate Calculations

Activity	Time	Unit
Shift change, travel to and from working area	1	hrs
Lunch time and Breaks	1	hrs
Equipment inspection	0.25	hrs
Available work time	9.75	hrs
Shift Length	12	hrs

16.7.2 Development Productivity

The unit lateral and vertical development rates were built up from first principles and calculated based on the cycle times for the selected equipment. The lateral development will be completed by a contractor using a mechanized diesel fleet consisting of the following units:

- 2 Boom Jumbo or Roadheader
- Explosives Chargeup Vehicle
- LHD – 7 t, Mineralized Heading / LHD – 10 t, Waste Heading
- Bolter
- Shotcrete Sprayer (where required)
- Truck – 30 t
- Auxiliary Support Equipment (e.g., Scissor Deck)

The ground support designs supplied by BGC vary by tunnel profile and location. An example cycle time buildup for a 5.0 m x 5.0 m development profile with shotcrete is outlined in Table 16-20.

Table 16-20: Cycle Time Breakdown for a 5.0 m x 5.0 m Development Profile with Shotcrete

Item	Cycle Time	Unit
Drilling Cycle	3.0	hrs
Blasting Cycle	2.0	hrs
Shoot and Ventilate Heading	0.5	hrs
Mucking Cycle	2.8	hrs
Ground Support Cycle	3.7	hrs
Shotcrete Support Cycle	3.0	hrs
Services Install	0.9	hrs
Total Cycle Time	15.9	hrs

The following rates in Table 16-21 were used for the LOM schedule, which aligns with the lateral and vertical development productivity calculated through the first principles build-ups.

Table 16-21 Mine Schedule Lateral and Vertical Development Rates

Item	Rate	Unit
Lateral Development Rate (per active heading)	3.5–4.0	m/day
Vertical Development Rate (Internal Raises)	2.0	m/day

16.7.3 Sub-Level Stoping Productivity

The PLS FS mine design comprises approximately 630 sub-level stopes being included in the LOM schedule. The stope size analysis showed that the average stope has a width of 6.7 m, a length of 10.0 m, and a vertical spacing sub-level spacing of 20.0 m. The stoping cycle time was built up from first principles using cycle times based on the equipment selected. The stoping cycle will be completed by the owner mining team and requires the following mobile equipment:

- Bolter (Cable Support)
- Longhole Drill
- Explosives Chargecar
- LHD – 7 t
- Truck – 30 t

The cycle time breakdown for an average stope is shown in Table 16-22.

Table 16-22: Cycle Time Breakdown for an Average Longhole Stope with Cable Support

Item	Cycle Time (hrs)	Shifts
Ground Support Cycle	23.8	3
Drilling Cycle	56.8	7
Blasting Cycle	18.7	2
Mucking Cycle	51.1	6
Backfilling Cycle	53.8	7
Cure Time – CRF	168	14[1]
Total time per stope	372.2	20 days

Note:
1. Cure time – CRF is based on a 24 hour day

The following rates in Table 16-23 were used for the LOM schedule. In some cases, the rates applied in the schedule were more conservative than the calculated first principles rates to ensure that the overall mine plan allowed for some flexibility during production activities. Other scheduling tools were used to improve the robustness of the mine plan, such as limiting the number of concurrent activities from a single mine area. This adds optionality to the mine plan and ensures that there is some contingency built into the production schedule.

Table 16-23 Mine Schedule Production Rates

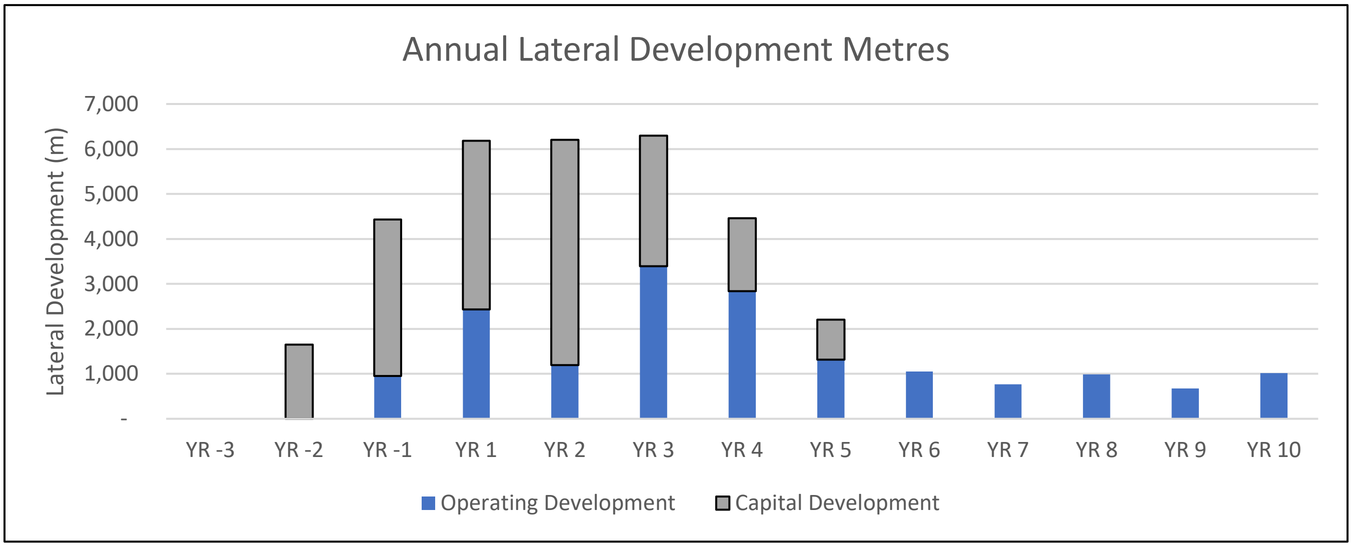
Task	Rate	Units
Production Drilling Rate	120	m/day/stope
Stope Mucking Rate	500	t/day/stope
Backfilling Rate – Stopes	250	m^3/day/stope
Backfilling Rate – D/F	300	m^3/day/stope

16.7.4 Development Scheduling

Lateral development peaks in year 1 through year 3 at approximately 6,300 m annually. Operating development in the later years of mine life, from year 7 onwards, is entirely comprised of D/F development accessing the upper crown pillar zones of the orebody. Table 16-24 and Figure 16-22 show a summary of the lateral development metres required for the project over the LOM.

Table 16-24: Annual Capital and Operating Development

Development Type	Year of Mine Life													Total
	YR -3	YR -2	YR -1	YR 1	YR 2	YR 3	YR 4	YR 5	YR 6	YR 7	YR 8	YR 9	YR 10	
Operating Development (m)	-	-	949	2,429	1,197	3,392	2,837	1,315	1,055	765	985	674	1,017	16,615
Capital Development (m)	-	1,651	3,482	3,757	5,008	2,903	1,624	890	-	-	-	-	-	19,314
Total	**-**	**1,651**	**4,431**	**6,185**	**6,205**	**6,294**	**4,461**	**2,206**	**1,055**	**765**	**985**	**674**	**1,017**	**35,928**



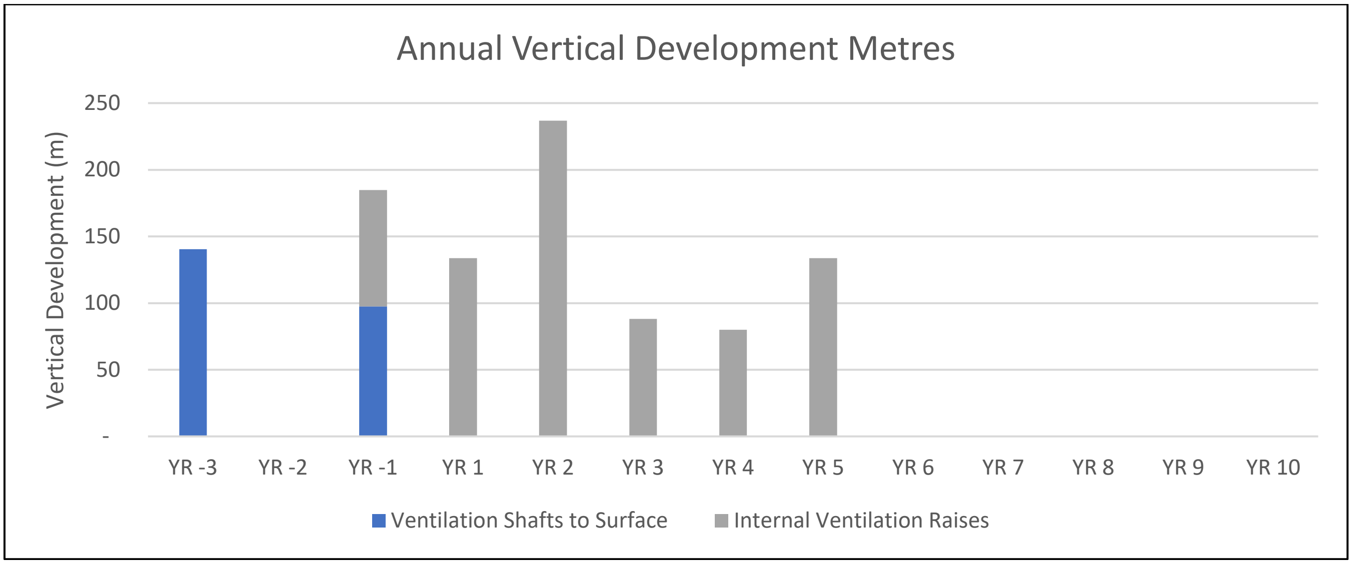
Source: Mining Plus, 2022

Figure 16-22: Annual Lateral Development Schedule

Vertical development begins with sinking of the FAS in year -3 and EAS in year -1. Vertical development peaks in year 2 and is completed by year 5 in line with the completion of capital lateral development. Table 16-25 and Figure 16-23 show a summary of the vertical development metres required for the project over the LOM.

Table 16-25: Annual Vertical Development Schedule

Development Type	Year of Mine Life													Total
	YR -3	YR -2	YR -1	YR 1	YR 2	YR 3	YR 4	YR 5	YR 6	YR 7	YR 8	YR 9	YR 10	
Surface Shafts (m)	140	--	97	--	--	--	--	--	--	--	--	--	--	238
Internal Raises (m)	--	--	87	134	237	88	80	134	--	--	--	--	--	759
Total (m)	**140**	**--**	**185**	**134**	**237**	**88**	**80**	**134**	**--**	**--**	**--**	**--**	**--**	**997**

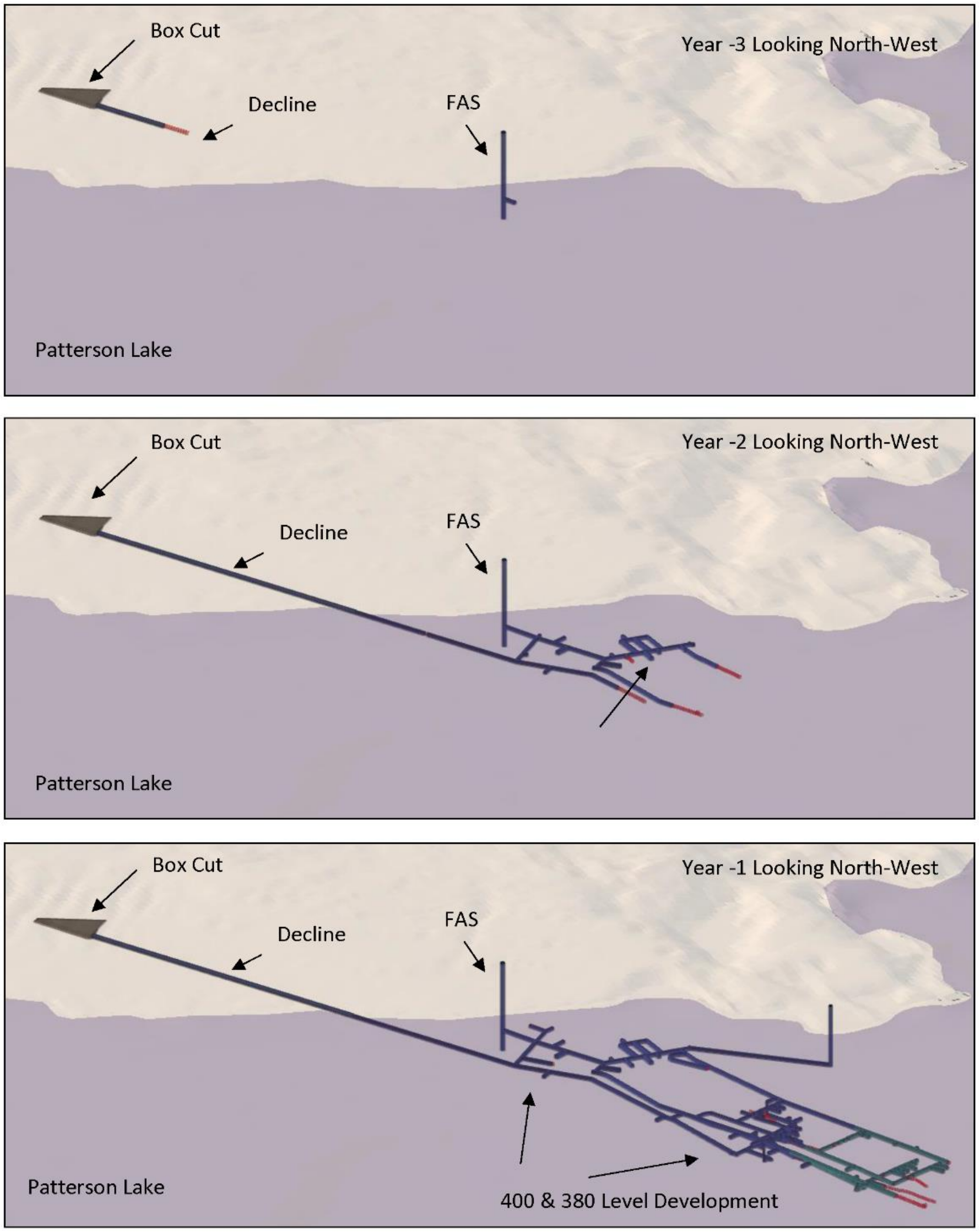


Source: Mining Plus, 2022

Figure 16-23: Annual Vertical Development Schedule

16.7.5 Construction Schedule

A three-year construction schedule is envisaged for the project. The shaft sinking activities are scheduled to be completed in series with the FAS being sunk first in year -3. The critical path for construction involves establishing a connection between the surface decline excavation and the underground lateral development. The decline initially waits until the box cut and dewatering have been completed and then develops toward the 400 level in year -3 and year -2. While the decline is under construction, capital lateral development is proceeding on the 400 level and the waste rock material is hoisted using the FAS sinking infrastructure. Once a connection has been established to the 400 level through the decline at the end of year -2, the FAS is outfitted with permanent services and the sinking infrastructure is mobilized to the EAS. In year -1 the EAS is sunk to the 400 level and the off-shaft development breaks into the shaft bottom. At this point, the sinking infrastructure is demobilized, and the permanent exhaust air fans are installed and commissioned. Once the primary negative pressure ventilation system is set up the lateral development proceeds into the R780E orebody to prepare for stoping. Figure 16-24 shows the underground mine sequence during the three-year construction period.



Not to Scale (Mining Plus, 2023)

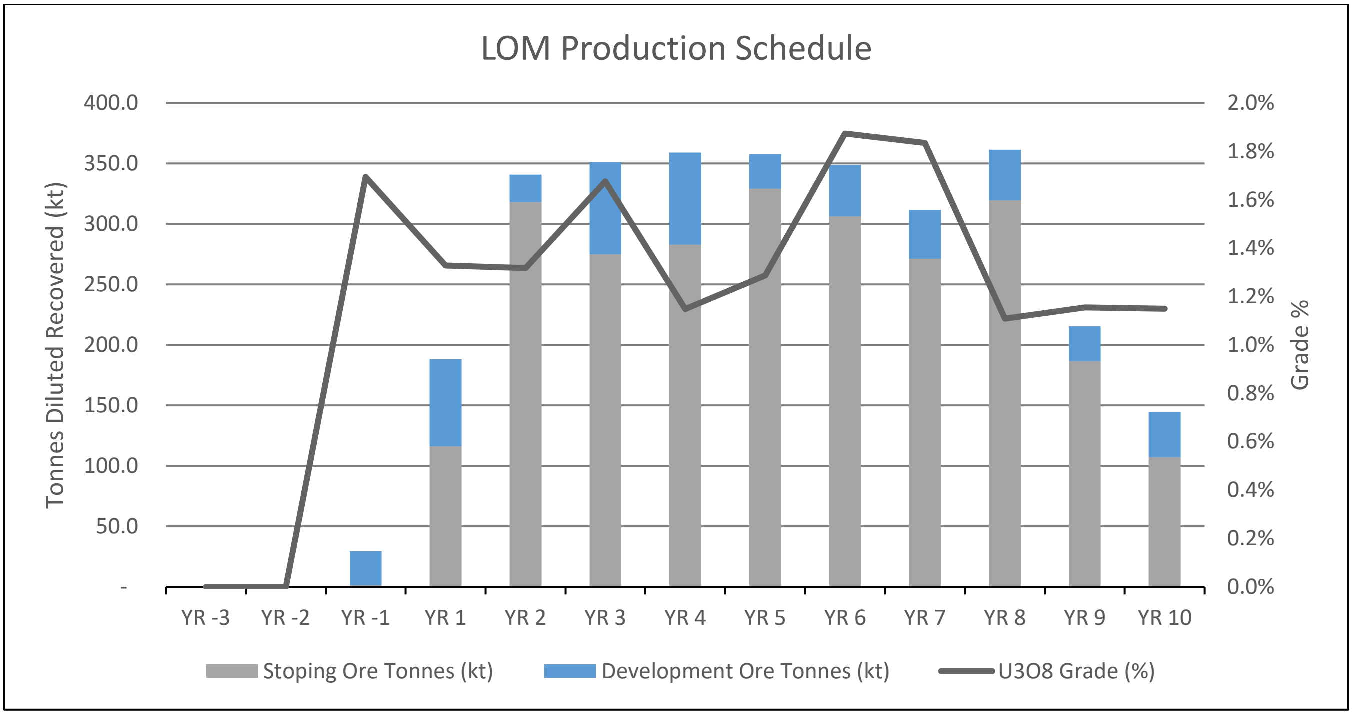
Figure 16-24: PLS Project Construction Sequence

16.7.6 Production Scheduling

The LOM waste and ore production schedule is shown in Table 16-26. Figure 16-25 and Figure 16-26 summarize in the LOM ore production and the annual metal contained in underground production delivered to the mill stockpile.

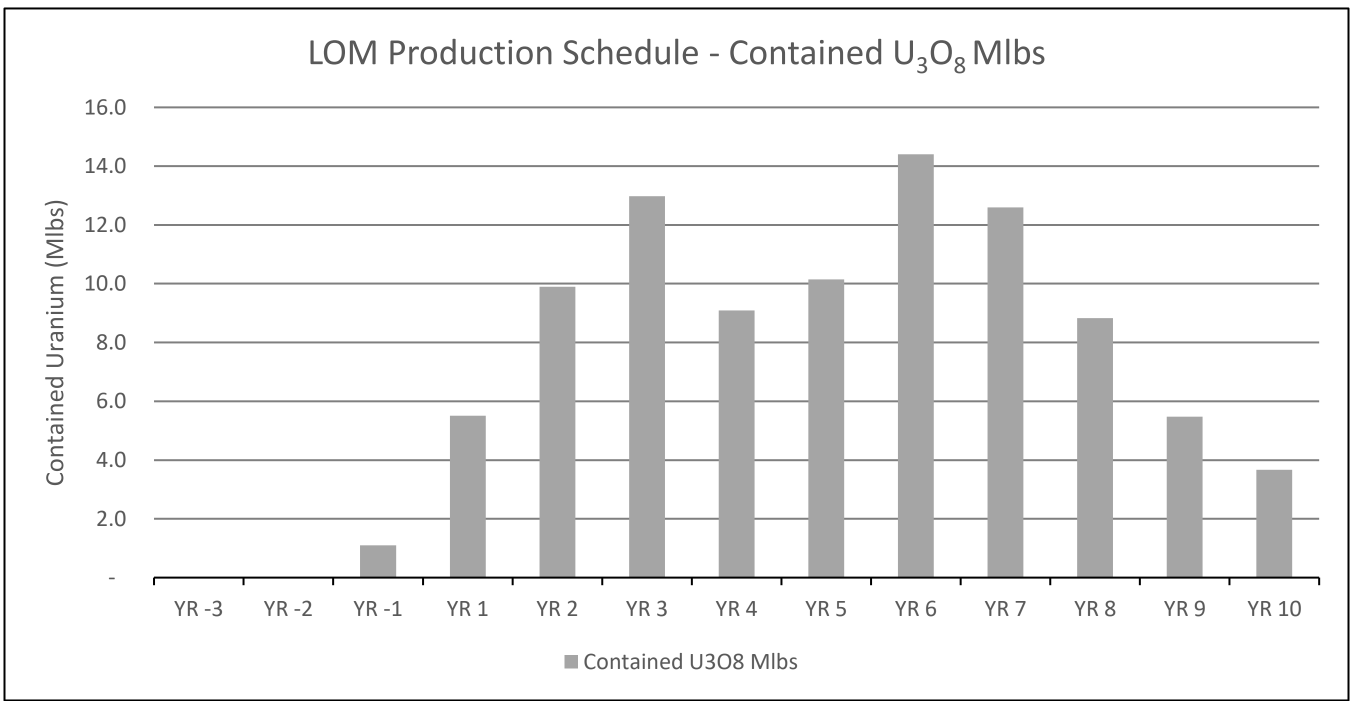
Table 16-26: Mine Production Schedule

Material	Total	YR -3	YR -2	YR -1	YR 1	YR 2	YR 3	YR 4	YR 5	YR 6	YR 7	YR 8	YR 9	YR 10
Waste Tonnes (kt)	1,488.0	27.4	143.8	222.7	260.9	327.1	227.9	144.2	83.5	7.3	2.9	10.3	8.5	21.4
Stoping Ore Tonnes (kt)	2,513.1	-	-	1.3	116.1	317.9	275.0	282.8	329.2	306.4	271.2	319.6	186.4	107.2
Development Ore Tonnes (kt)	494.4	-	-	28.0	71.9	22.8	76.2	76.2	28.5	42.3	40.4	41.8	28.8	37.6
Total Ore Tonnes (kt)	3,007.5	-	-	29.3	188.0	340.7	351.2	359.0	357.7	348.7	311.6	361.4	215.3	144.8
U_3O_8 Grade (%)	1.41%	0.0%	0.0%	1.7%	1.3%	1.3%	1.7%	1.1%	1.3%	1.9%	1.8%	1.1%	1.2%	1.1%
Contained U_3O_8 Mlbs	93.7	-	-	1.1	5.5	9.9	13.0	9.1	10.1	14.4	12.6	8.8	5.5	3.7



Source: Mining Plus, 2022

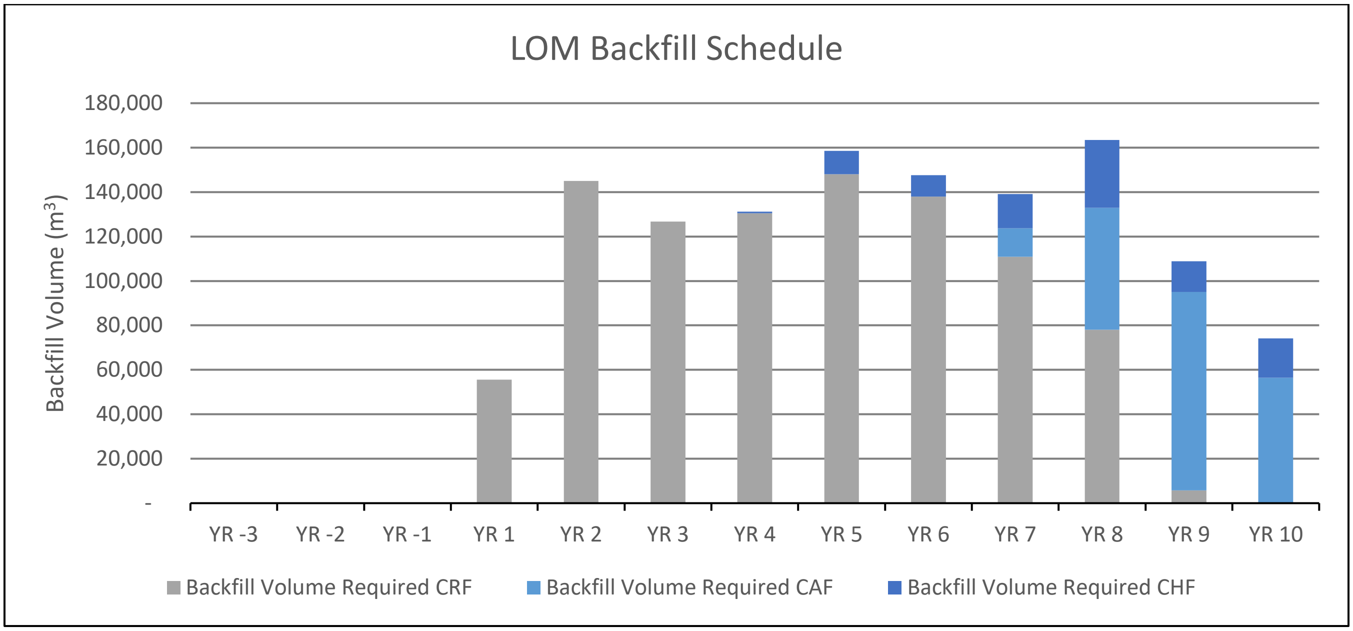
Figure 16-25: Production Tonnes and U_3O_8 Grade by Year



Source: Mining Plus, 2022

Figure 16-26 Annual Production Schedule – Contained Metal

The LOM backfill schedule is shown in Figure 16-27 below. Backfill demand is approximately 130,000 m^3/year peaking at approximately 160,000 m^3 in year 8.



Source: Mining Plus, 2022

Figure 16-27 Annual Backfill Demand by Fill Type

16.7.7 Mine Labour

Mine labour includes the following functions covering the project construction period and operations. Table 16-27 summarizes the annual mine labour by area.

- Owner employees for mine management, technical services, and production mining.
- Underground contract mining crews for capital and operating development with associated administrative and maintenance personnel.
- FAS and EAS contractor construction crews.
- Decline construction contractor crews.
- Refrigeration plant and ground freezing system installation labour.
- Underground infrastructure construction labour.
- Underground mine engineering, procurement, and construction management (EPCM) labour.

Table 16-27: Annual Total Labour Count – Mining

Labour Type	Year of Mine Life												
	YR -3	YR -2	YR -1	YR 1	YR 2	YR 3	YR 4	YR 5	YR 6	YR 7	YR 8	YR 9	YR 10
Owner Employees													
Mine Mgmt. & Tech Services	--	14	14	25	25	25	25	25	25	25	25	14	14
Production Mining	--	--	22	30	44	44	44	44	44	44	44	44	30
UG Contract Mining													
Administration	3	3	3	3	3	3	3	3	3	3	3	3	3
Mine Crews	--	14	35	39	39	44	32	27	15	15	15	15	15
Maintenance Crews	--	14	14	14	24	24	24	24	24	24	24	14	14
FAS/EAS Labour													
Shaft Crew	36	36	36	--	--	--	--	--	--	--	--	--	--
Shaft Support	8	8	8	--	--	--	--	--	--	--	--	--	--
Decline Development													
Decline Development Crew	28	28	28	--	--	--	--	--	--	--	--	--	--
Supporting Crews	38	38	38	--	--	--	--	--	--	--	--	--	--
Project Staff	19	19	19	--	--	--	--	--	--	--	--	--	--
Crown Pillar Freezing Labour													
Crown Pillar Freezing Labour	--	--	--	12	36	20	30	--	--	--	--	--	--
UG Infrastructure Labour													
Construction Labour	--	40	40	--	--	--	--	--	--	--	--	--	--
EPCM													
EPCM Labour	10	10	10	--	--	--	--	--	--	--	--	--	--
Total	**142**	**224**	**267**	**123**	**171**	**160**	**158**	**123**	**111**	**111**	**111**	**90**	**76**

16.7.8 Mobile Equipment

The mine equipment fleet selected and requirements by year is shown in Table 16-28.

Table 16-28: Annual Underground Mobile Equipment Fleet

Equipment Type	Annual Equipment Requirements												
	YR -3	YR -2	YR -1	YR 1	YR 2	YR 3	YR 4	YR 5	YR 6	YR 7	YR 8	YR 9	YR 10
Prod/Dev Equipment													
Haul Trucks – 30 t	--	--	3	4	5	5	4	4	3	3	3	3	2
LHD – 10 t – Dev	--	1	2	2	2	2	1	1	--	--	--	--	--
LHD – 7 t – Prod	--	--	1	2	3	3	3	3	3	3	3	3	2
Jumbos	--	1	1	1	1	1	1	1	--	--	--	--	--
LH Drills	--	--	1	1	2	2	2	2	2	2	2	2	1
Roadheaders	--	--	1	1	1	2	2	1	1	1	1	1	1
Bolter	--	1	1	2	2	2	1	1	1	1	1	1	1
Explosive Loaders – Dev	--	1	1	1	1	1	1	1	1	--	--	--	--
Explosive Loaders – Production	--	--	1	1	1	1	1	1	1	1	1	1	1
Shotcrete Sprayers	--	--	1	1	1	1	1	1	1	1	1	1	1
Auxiliary Equipment													
Lube Truck	--	--	1	1	1	1	1	1	1	1	1	1	1
Flat Deck Truck w. Crane	--	--	2	2	2	2	2	2	2	2	2	2	2
Transmixer	--	--	1	1	1	1	1	1	1	1	1	1	1
Personnel Carrier	--	--	2	2	2	2	2	2	2	2	2	2	2
Scissor Lift	--	--	3	3	3	3	3	3	3	3	3	3	3
Light Vehicles	--	--	6	6	6	6	6	6	6	6	6	6	6
Grader	--	--	1	1	1	1	1	1	1	1	1	1	1
Boom Lift	--	--	1	1	1	1	1	1	1	1	1	1	1
Fuel Truck	--	--	1	1	1	1	1	1	1	1	1	1	1
Wheel Loader	--	--	1	1	1	1	1	1	1	1	1	1	1

Mobile equipment replacements were determined based on the useful life of the equipment, as supplied by the equipment manufacturer or that of a similar equipment model. Once the threshold is crossed where the equipment is used for more hours than its useful life it is scheduled to be replaced within that year. Table 16-29 shows the mobile fleet replacements required by year.

Table 16-29: LOM Cumulative Replacements Required by Equipment Type

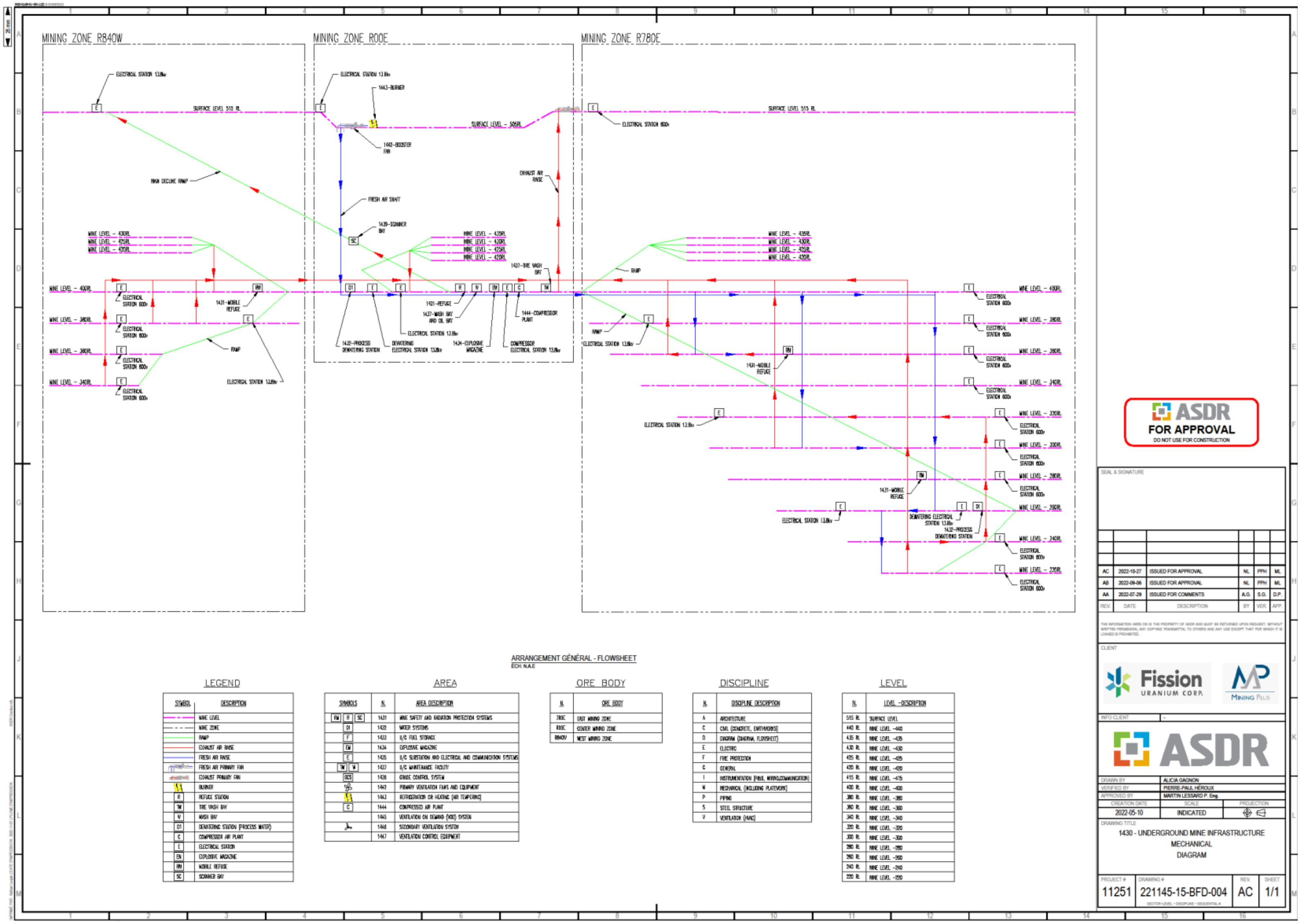
Equipment Type	Equipment Replacements By Year												
	YR -3	YR -2	YR -1	YR 1	YR 2	YR 3	YR 4	YR 5	YR 6	YR 7	YR 8	YR 9	YR 10
Mobile Equipment													
Haul Trucks – Contractor	--	--	--	--	--	--	--	--	--	--	--	--	--
Haul Trucks – Owner	--	--	--	--	--	--	--	--	--	--	1	--	--
LHD - 10.2t - Dev	--	--	--	--	--	--	--	--	--	--	--	--	--
LHD - 6.7t - Prod	--	--	--	--	--	--	--	--	--	--	1	--	--
Jumbos	--	--	--	--	--	--	--	--	--	--	--	--	--
LH Drills	--	--	--	--	--	--	--	--	--	1	--	--	--
Roadheaders	--	--	--	--	--	--	1	1	--	--	--	--	--
Bolter	--	--	--	--	--	--	--	--	--	--	1	--	--
Explosive Loaders - Dev	--	--	--	--	--	--	--	--	--	--	--	--	--
Explosive Loaders - Production	--	--	--	--	--	--	--	--	--	--	1	--	--
Shotcrete Sprayers	--	--	--	--	--	--	--	--	--	--	--	--	--
Auxiliary Equipment													
Lube Truck	--	--	--	--	--	--	--	--	--	1	--	--	--
Flat Deck Truck w. Crane	--	--	--	--	--	--	--	--	--	1	--	--	--
Transmixer	--	--	--	--	--	--	--	--	--	--	--	--	--
Personnel Carrier	--	--	--	--	--	--	--	--	--	1	--	--	--
Scissor Lift	--	--	--	--	--	--	--	--	--	--	--	--	--
Light Vehicles	--	--	--	--	-	-	1	1	1	1	1	--	--
Grader	--	--	--	--	1	-	-	-	1	--	--	--	--
Boom Lift	--	--	--	--	--	--	--	--	--	--	--	--	--
Fuel Truck	--	--	--	--	--	--	--	--	--	1	--	--	--
Wheel Loader	--	--	--	--	1	--	--	--	1	--	--	--	--

16.8 Underground Mine Infrastructure

The mine underground services and infrastructure considered for the PLS project include the following:

- Mine Safety and Refuge Stations
- Dewatering System
- Process Water System
- Explosive and Detonator magazine
- Electrical, Substation, and Communication Systems
- Wash Bay Facility
- Grade Control System
- Primary Ventilation Fans and Equipment
- VOD System and Secondary Ventilation System
- Compressed Air Plant
- Backfill Plant
- Refrigeration Plant

A general view of the underground infrastructure is presented in Figure 16-28.



Source: ASDR, 2022

Figure 16-28: PLS Underground Mine Infrastructure Schematic

16.8.1 Mine Safety and Refuge Stations

Refuge stations will be constructed underground at the PLS project that meet the local mining regulations. The primary refuge area consists of one underground permanent refuge station capable of accommodating 30 people which is the total amount of personnel expected to be underground during a shift. The primary refuge location on 400 level has been selected so that it remains in fresh air at all times and is located more than 100 m from the explosives and detonator magazines as per local regulations. There are also three mobile refuges for eight people each that will be placed as needed while mining advances in the various zones.

The primary refuge has all the equipment and supplies needed to sustain 30 people for 36 hours during an emergency event. It also has a kitchen and three offices for technical and supervisory staff to be used on a regular basis.

The mobile refuges can be moved to follow advancing development, otherwise they will be distributed in the mine as follows. There will be one located on the 400 Level of the mining zone R840W, another on the 360 Level of the mining zone R780E and the last one is on the Level 280 of the mining zone R780E. The mobile refuges will provide refuge for workers within a reasonable distance of all locations in the mine. These refuges can sustain eight people for 36 hours should an emergency occur, which is the number of personnel expected for one development and one production crew.

16.8.2 Mine Dewatering and Process Water Systems

16.8.2.1 Decline Dewatering System

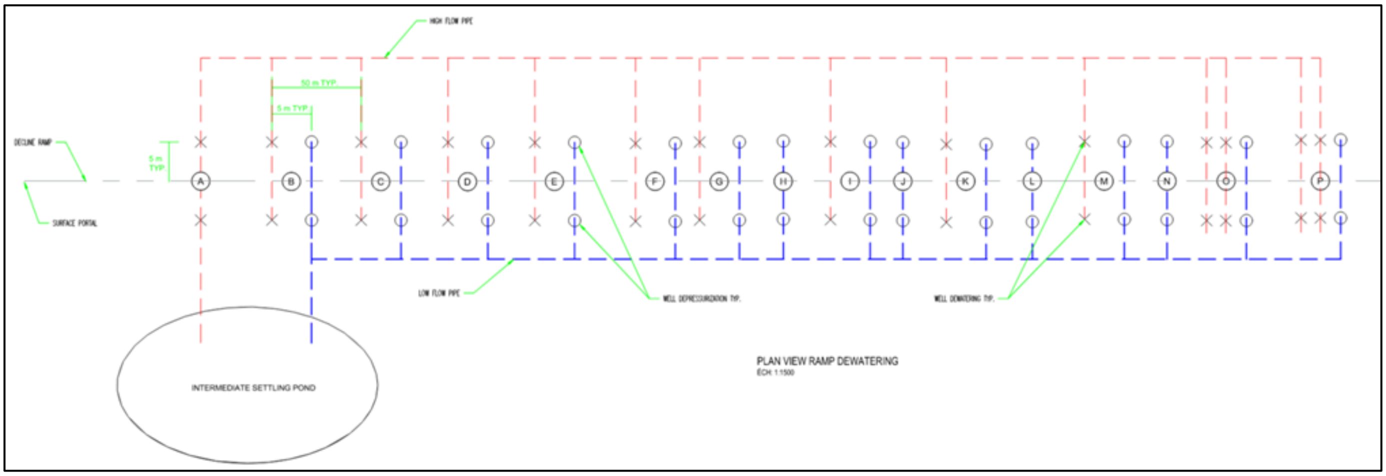
The decline dewatering system will be commissioned six months before the beginning of decline excavation. The goal is to reduce water pressure to near 0 kPa within the overburden geological material units to allow the excavation process of the decline. BGC was responsible for the hydrogeological study and determined the estimated water volume for pumping, the dewatering hole/well sizing, and the arrangement of the dewatering holes/wells (BGC Engineering Inc. 2022b). A total of 58 pumps will be needed for the decline dewatering system. There are two systems required, the dewatering system (28 pumps) will handle most of the near surface water, while the depressurization system (30 pumps) will be used to lower the water pressure at depth below the decline. Due to large differences in flow and pressure, each set of pumps has its own piping system that reports to the intermediate settling pond (ISP). From the ISP, the water will be pumped to the water treatment facility. The decline dewatering system will operate for the entire duration of the decline excavation. Upon decline completion, the system is shut off and decommissioned.

For the dewatering system, 20 pumps will each collect 110 m^3/day with a 25 m total dynamic head (TDH), and 8 pumps will each collect 416 m^3/day with a 35 m TDH. In total, 5,515 m^3/day will be sent to the ISP for further treatment.

For the depressurization system, 24 pumps will each collect 32 m^3/day with an 80 m TDH, and 6 pumps will each collect 58 m^3/day with an 80 m TDH. In total, 1,120 m^3/day will be sent to the ISP for further treatment.

The vertical or underground piping will be a Boreline hose while the horizontal or surface piping will be HDPE and carbon steel (at the well heads). Each pump line will be equipped with check and ball valves.

A schematic of the decline pumping system is presented in Figure 16-29 below.



Source: ASDR, 2022

Figure 16-29: Decline Dewatering System Schematic

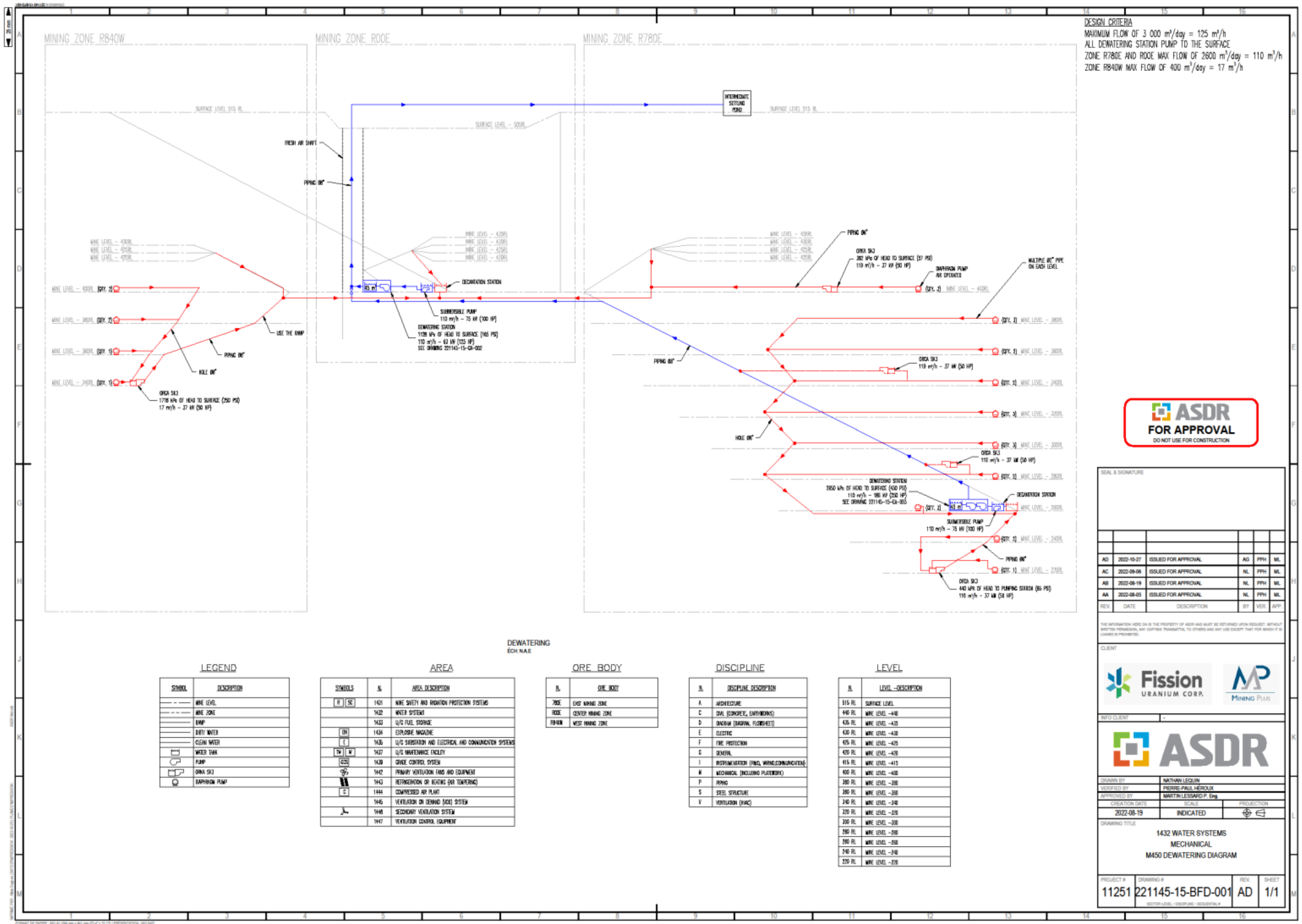
16.8.2.2 Main Underground Dewatering System

The main underground mine dewatering system, or the permanent dewatering system, was designed to be able to process a total of 125 m³/h (3,000 m³/day) as stated by BGC for the mitigated decline dewatering case (BGC Engineering Inc. 2022b). The pumps and piping have been sized accordingly.

First, the water is collected at the working faces using air-operated diaphragm pumps. Depending on proximity, water is sent directly to either the nearest settling station or booster pump skid. The booster pump skid can receive water from multiple diaphragm pumps. There are two planned settling and pumping stations, one located on the 400 level of the R00E Zone and the other on the 260 level of the R780E Zone.

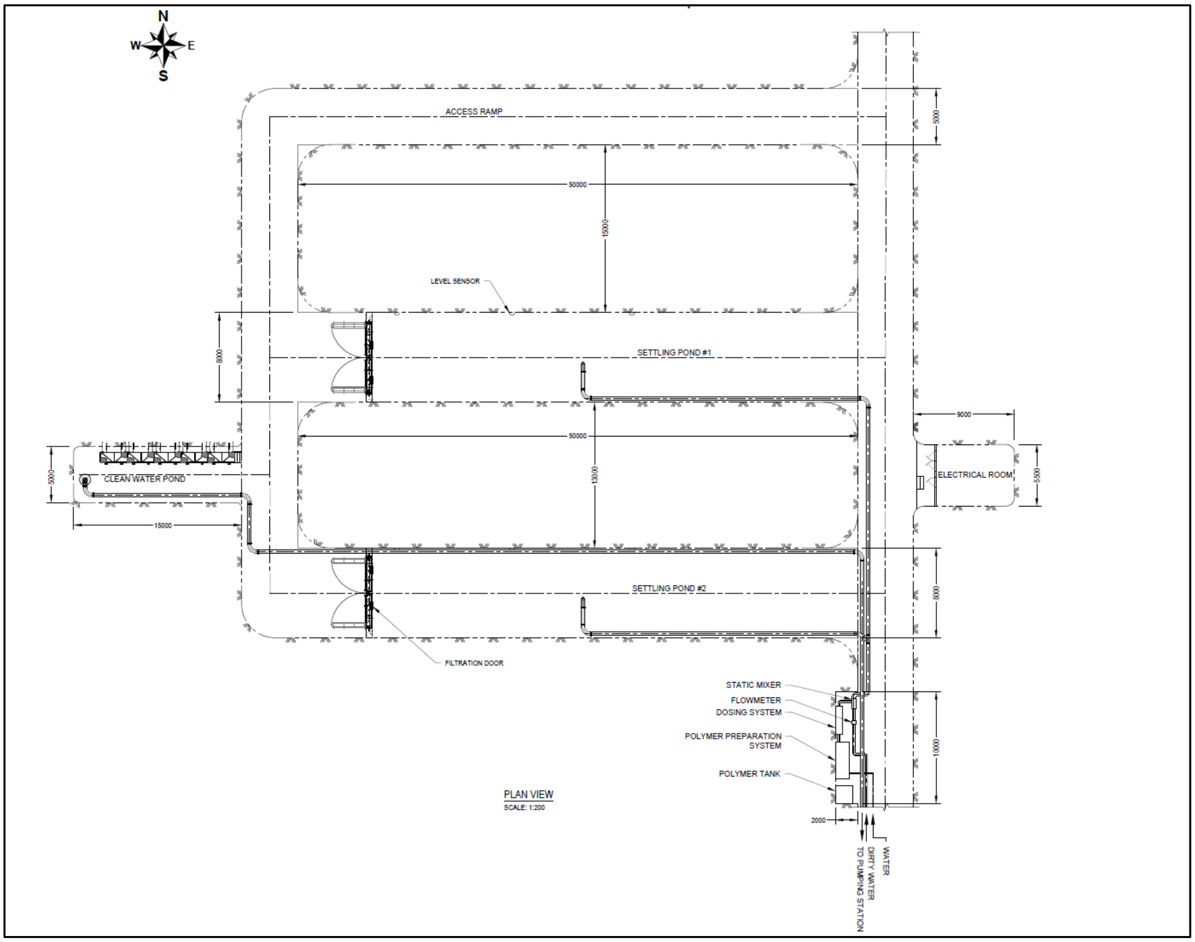
When the water arrives at a settling station, it is mixed with a flocculating agent to assist in separating water from solid materials. The dirty water is then sent into a settling sump to allow for the completion of the decant process. Once the water is separated from the solids, it goes through a geomembrane and is collected in a clean water sump via gravity. The preliminary mesh size of the membrane is set to 200 microns, but this will be finalized in the detailed engineering phase. In this arrangement, the clean water accumulates in a separate sump and can be sent to the primary pumping station while the solids fill up the settling sump until it is full. When this happens, the dirty water is redirected to a second settling sump to complete the same process while the solids in the first sump dry before getting emptied.

The decanted water is sent to the surface ISP by the pumping station via the FAS. Due to the difference in elevation, the station at the 260 level requires additional pumping stages, and therefore more power than the one at 400 Level, both use the same pump models. The first station that is going to be installed is the one on 400 Level. Due to this, it needs to be able to process all the water until the second station at 260 Level can be commissioned. Both stations are designed to be capable of pumping 110 m³/h (2,600 m³/day). The pumps are operated with VFDs and operating both sets in parallel can allow up to 315 m³/h with the current selection. The final sizing will have to be confirmed in the detailed engineering phase. All the pumping stations are equipped with a secondary standby pumping system to ensure that water can be sent to surface without interruption due to breakdown or maintenance. Figure 16-30 and Figure 16-31 show the general arrangement of the planned dewatering system and 400 level settling sump respectively.



Source: ASDR, 2022

Figure 16-30: Mine Dewatering Schematic



Source: ASDR, 2022

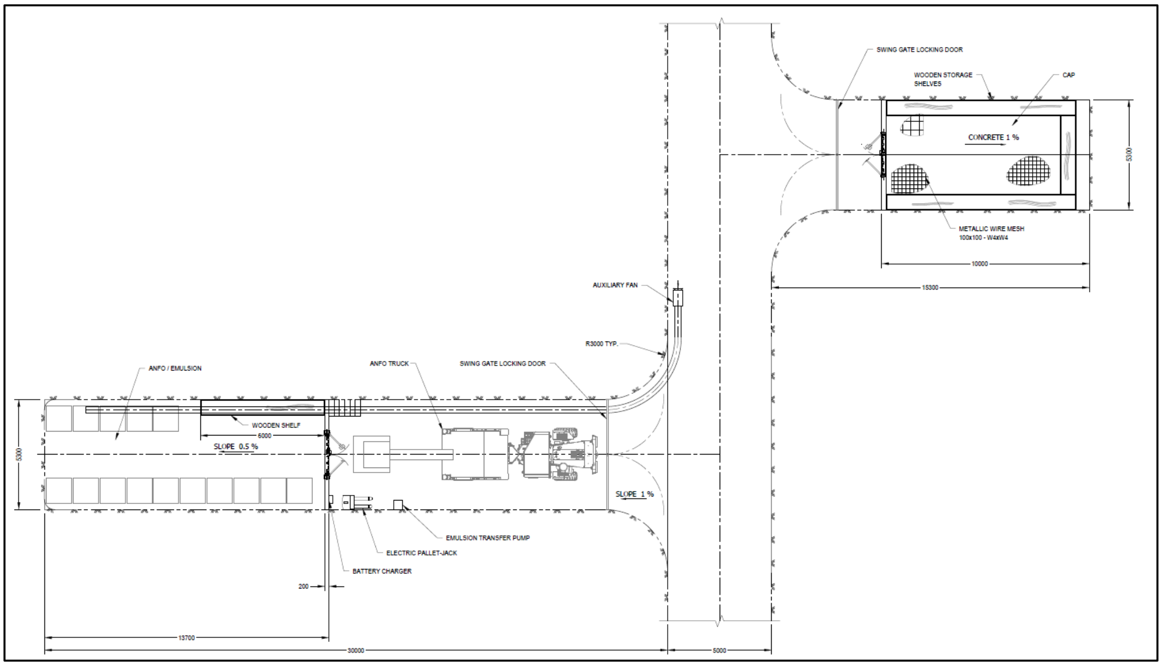
Figure 16-31: Sump Decantation Station Arrangement

16.8.2.3 Process Water System

A process water system is planned to supply the underground mining operations. This water is routed from surface and goes into the mine to supply equipment and infrastructure. On surface, near the FAS collar, there is a 10 m^3 water tank that supplies water to each level of the mine by gravity. To reduce the line pressure, two different systems are used: the first one is an open (non-pressurized) tank where a mechanical float controls the tank level to maintain a sufficient volume. The second system is a dual pressure relief valve system, where two valves are placed in parallel. This ensures that process water will always be available even when a valve needs to be replaced. After the water has been used, it gets collected by diaphragm pumps and returns to the surface via the dewatering system described in the previous section.

16.8.3 Explosive and Detonator Magazines

The explosive and detonator magazines are in the R00E Zone on the 400 Level. The explosive and detonator magazine have been designed to meet local regulations and have been isolated from the main fresh air circuit. For safety purposes, an auxiliary fan has been installed to circulate air in the explosive magazine and a locking swing gate will prevent unauthorized access to both rooms. The explosive magazine is large enough to allow a truck to park inside the swing gate. The general arrangement is presented in Figure 16-32 below.



Source: ASDR, 2022

Figure 16-32: Explosive and Detonator Magazine Plan View

16.8.4 Electrical, Substation, and Communication Systems

This section covers the electrical equipment and instrumentation for the 13.8 kV and 600 V rooms that are distributed over the three mining zones, and all the materials required to connect the electrical equipment.

The primary feed to the mine is 13.8 kV which is stepped down to 600 V for mobile equipment and infrastructure use. In total, there are 9 x 13.8 kV electrical rooms feeding 14 x 600 V electrical rooms.

The 13.8 kV electrical rooms contain the following equipment:

- Electrical substation with a 1.5 MVA transformer
- Switchgear in some rooms (the two rooms on the surface and the room on level 400 RL R00E)
- 600 V distribution panel
- Two power take-off, three plugs in some rooms (they will be moved between the rooms as required by mining needs)
- Cable tray
- Heating 30 kW
- Electrical service 120/208
- Electrical room hardware
- Ventilation starter (as needed)

The 13.8 kV electrical rooms contain the following equipment:

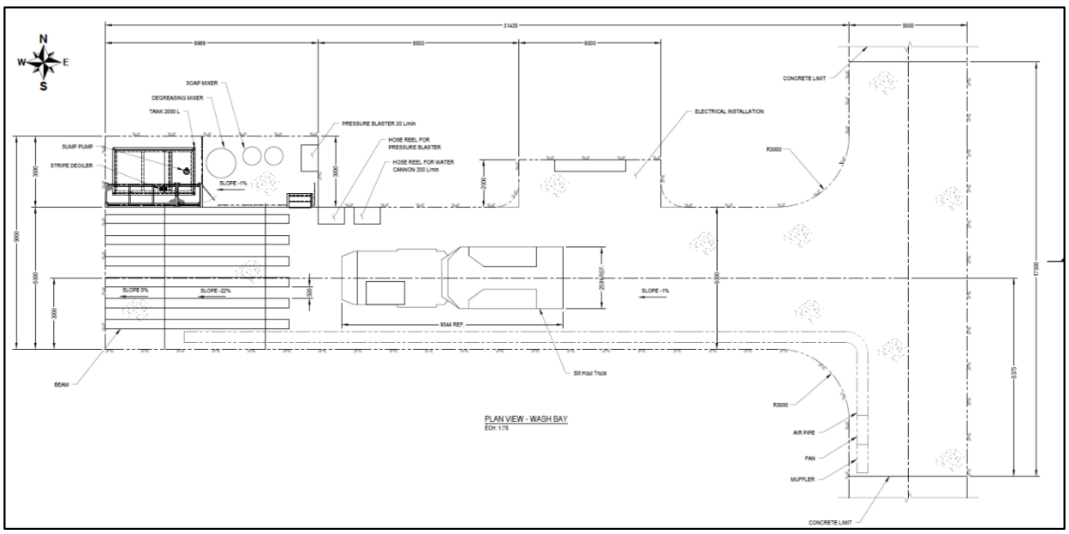
- Distribution panel 600 V panel

- Ventilation starter
- Heating 30 kW
- Electrical service 120/208 V
- Two power take offs, three plugs in some rooms (they will be moved between the rooms as required by mining needs)
- Extension jumbo 200 ft. 1/0 SHD GGC (as needed)
- Cable tray
- Electrical room hardware

In every electrical room, there is a communication and control system as required by the area. A fibre optic distribution backbone and a leaky feeder distribution are planned between the different zones and levels of the mine.

16.8.5 Maintenance Facilities

Underground equipment maintenance facilities are limited to a Wash Bay and Tire Wash Bay as the main facilities will be on surface due to the mine's shallow depth. The surface maintenance facilities are described in Section 18.4.1. The Wash Bay is planned to be on the 400 Level of the R00E Zone. It will be equipped with a pressure washer, a water cannon, and a water collection system. The oil is separated from the water through an oil separator, and the water is recirculated to reduce consumption. Figure 16-33 shows the general arrangement of the planned Wash Bay.



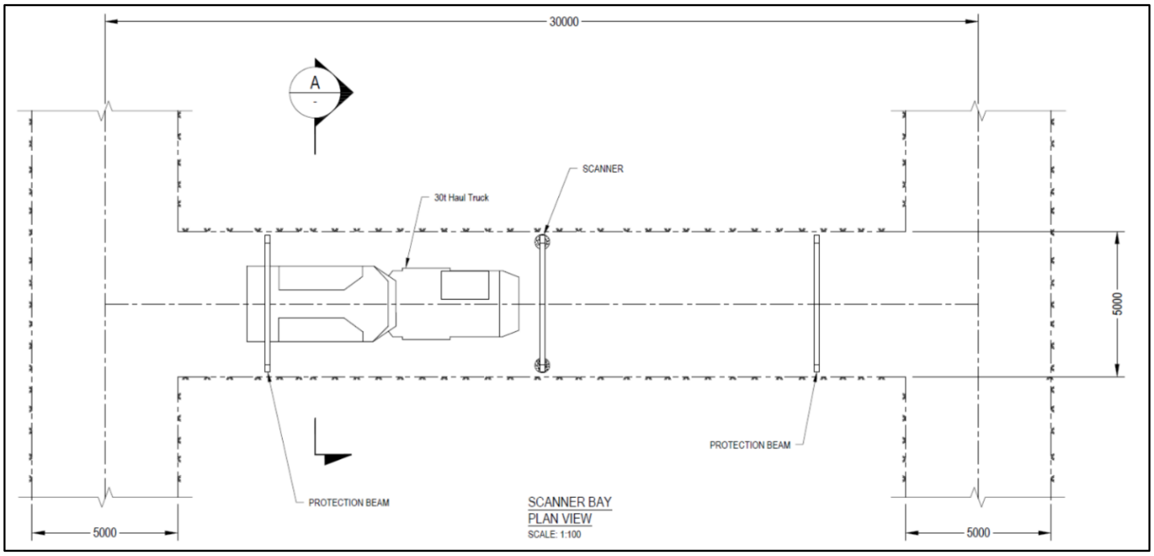
Source: ASDR, 2022

Figure 16-33: Wash Bay Plan View

A Tire Wash Bay is planned near the bottom of the main ramp access and is a pass-through tunnel. It has powerful sprinklers on both sides to spray the tires and a sump to collect and treat the water. This will be a closed loop system to minimize the use of clean water.

16.8.6 Grade Control System

A grade control system is planned near the bottom of the ramp. It is designed as a pass-through gallery that connects two parallel tunnels to allow drive through use from all production areas. On either side of the scanner, a protection levelling beam will be installed. The beams will ensure no material that could damage the scanner is able to pass through. The scanner will provide the truck operator with the approximate grade of the material so it can be sent to the correct mill stockpile. Figure 16-34 shows the general arrangement of the Scanner Bay.



Source: ASDR, 2022

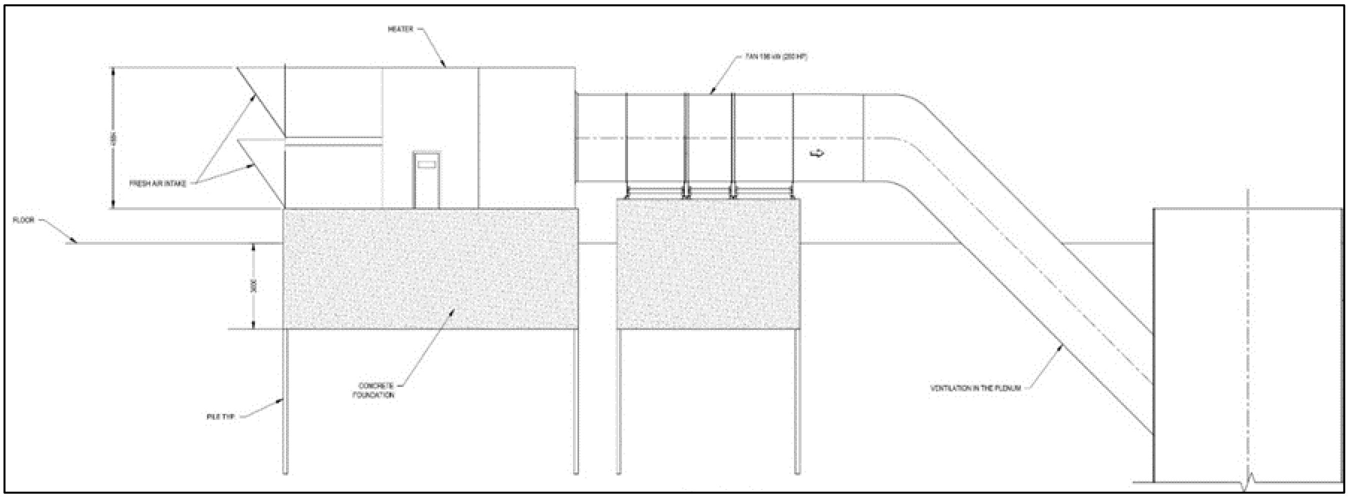
Figure 16-34: Grade Control Scanner Bay Plan View

16.8.7 Primary Ventilation Fans and Equipment

The primary ventilation system consists of intake and exhaust fans located on surface at their respective shafts (FAS and EAS). Both the intake and exhaust systems have dedicated electrical rooms next to the mechanical equipment.

The primary intake fans consist of two 186 kW (250 hp) fans arranged in parallel. They sit horizontally behind the mine air heaters and are attached to the FAS via a sub-collar plenum. The selected fans can produce a flow rate of 350 m³/s at a pressure of 1.8 inches of water gauge. Due to the location of the mine (Northern Saskatchewan, Canada), the ventilation system requires a heating system that can raise the air temperature by a maximum of 48°C (-43°C to +5°C). For this purpose, a 22.5 MW direct fire heater fueled by natural gas has been chosen. It will be integrated with the primary intake fans.

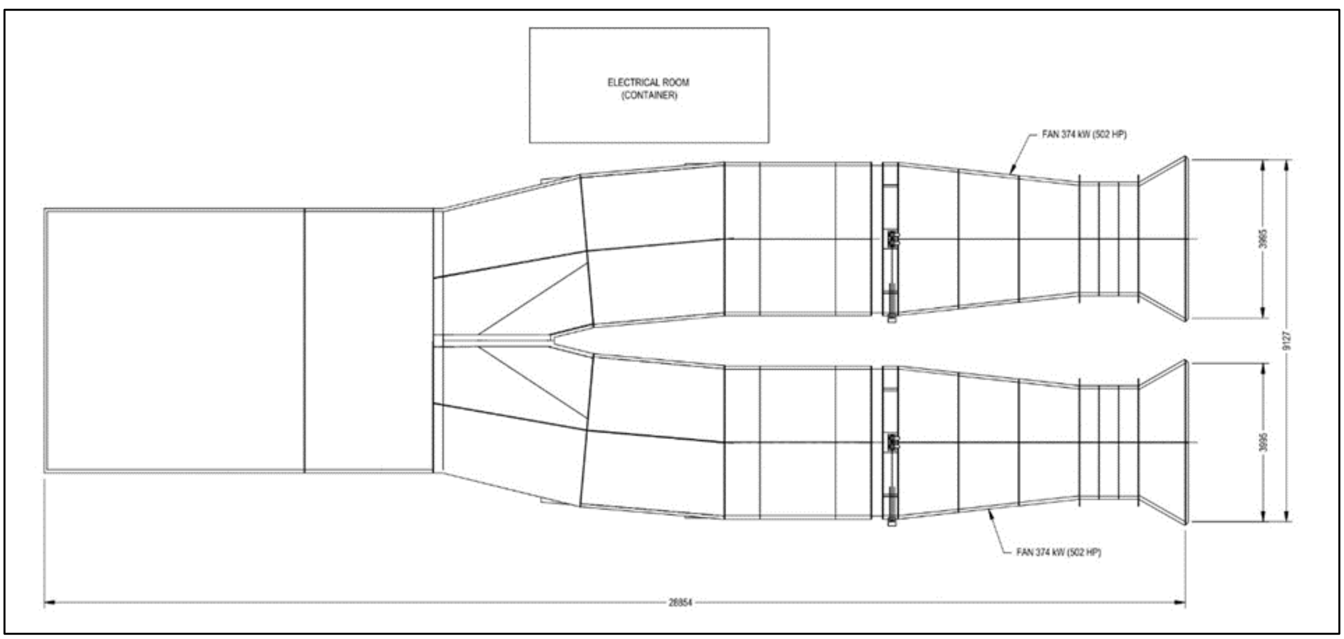
Figure 16-35 shows a section of the primary ventilation intake fans.



Source: ASDR, 2022

Figure 16-35: Primary Intake Fans and LNG Direct Fired Heater Section View

The exhaust fans are the primary air movers in the main ventilation network and generate negative pressure to remove air from the mine. There are two 274 kW (370 hp) fans arranged in parallel capable of producing a flow of 320 m^3/s at negative pressure of 6.9 inches of water gauge. The exhaust air fans will be installed horizontally inline with ductwork and an elbow that is placed over the EAS. Figure 16-36 shows the plan view arrangement of the exhaust fans.



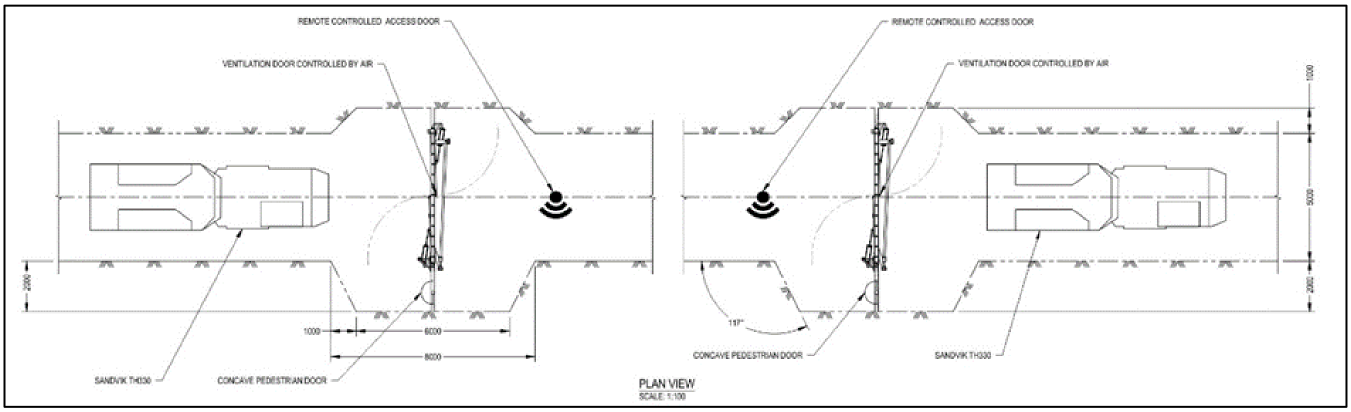
Source: ASDR, 2022

Figure 16-36: Plan View Exhaust Fan Layout

16.8.8 Secondary Ventilation System and Ventilation on Demand

The VOD system is meant to control the air distribution into the mine. It will send all the available airflow and quality data to the mine control room where an operator will be able to manage all the ventilation of the mine. In addition to the instrumentation, they are two main components to this system, the airlock, and the automatic louvers.

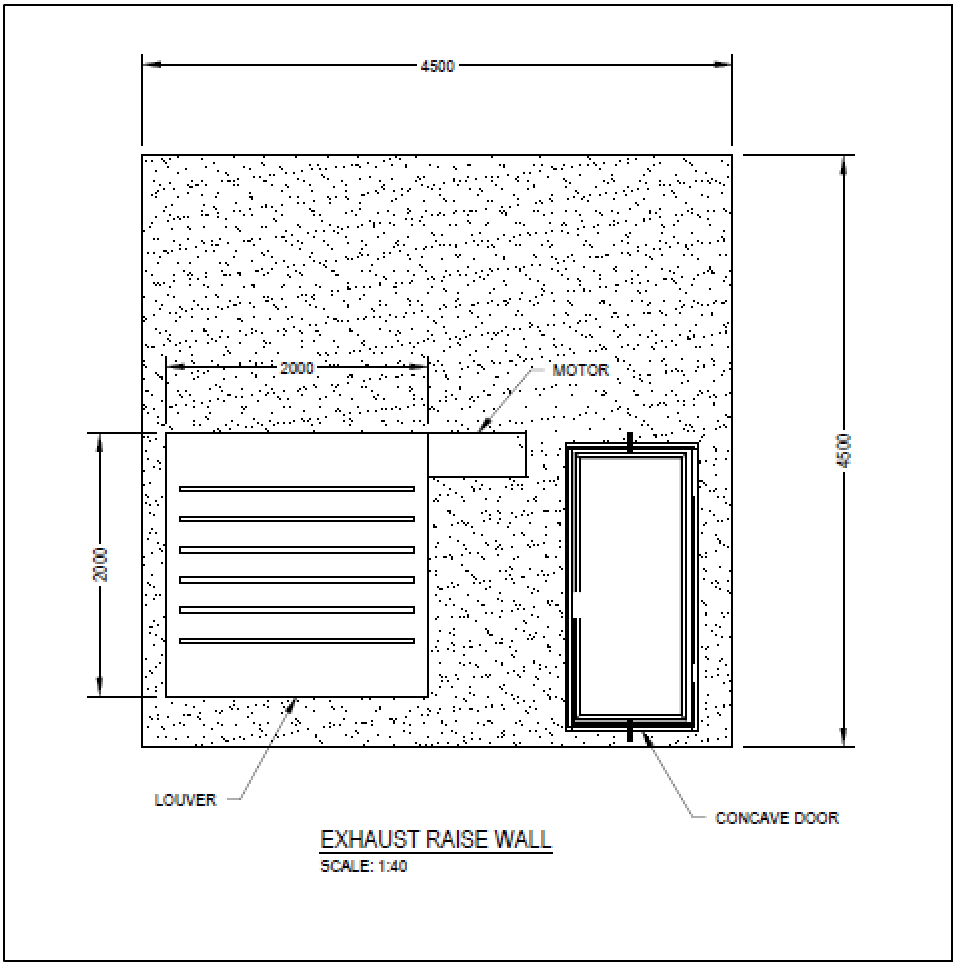
The main objective of the airlock (see Figure 16-37 below) is to stabilize the pressure and prevent a short circuit of the primary ventilation system. It is located on the 400 Level of the R00E Zone. There will be two sets of parallel doors to prevent airflow from going directly from the FAS to the EAS. This way, access can be maintained to the main return air infrastructure for inspection or maintenance as required.



Source: ASDR, 2022

Figure 16-37: Air Lock Typical

The automatic louvers are installed on all production levels at the connection to the exhaust raise/system. They are operated by the surface control room and can be open or closed to adjust air flow as required. Figure 16-38 shows a typical exhaust raise wall with louvers and man door to maintain access for inspection and maintenance.



Source: ASDR, 2022

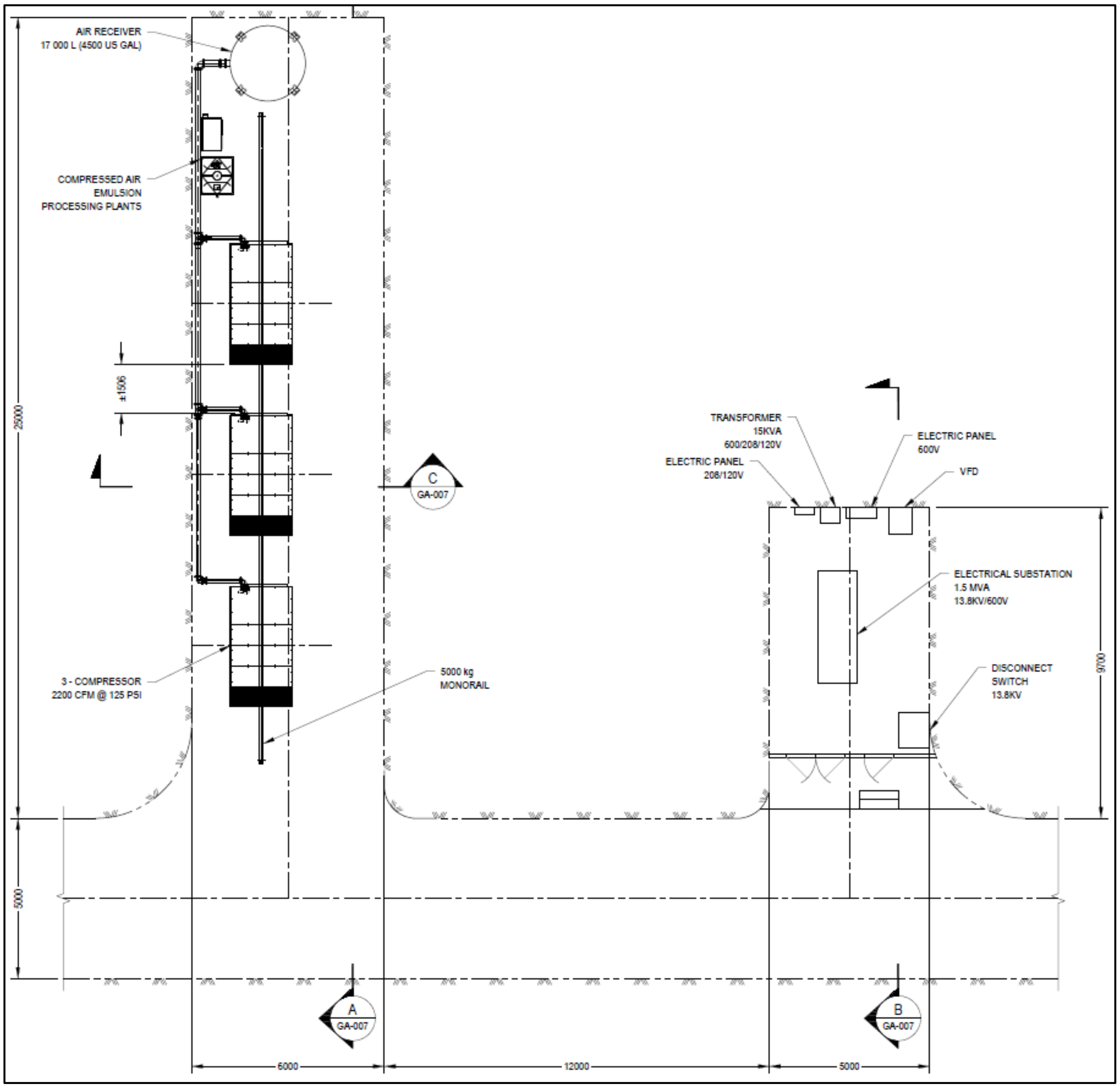
Figure 16-38: Exhaust Raise Bulkhead – Typical

The mineralized zones are kept under negative pressure by an auxiliary ventilation system that can supply 24.7 m^3/s at a negative pressure of 9 inches of water gauge. The air will be channeled through 42 inch (1.07 m) diameter rigid ducts. This will ensure that air is not reused in production headings and that personnel always have fresh air.

All the other zones will use positive pressure auxiliary fan arrangements. This system will supply 25 m^3/s at a pressure of 8 inches of water gauge through flexible ducts of 48 inch (1.2 m) diameter.

16.8.9 Compressed Air Plant

The compressor room is located on the 400 level of the R00E Zone. There are three 1 m^3/s (2,200 cfm @ 125 PSI) screw compressors which supply a 17,000 L tank. There is also an air dryer and an emulsion system to clean the oily water. The heat generated by the compressor will be evacuated out of the room by a ventilation duct (see Figure 16-39). The compressed air will be available for use by the mobile equipment fleet and dewatering system.



Source: ASDR, 2022

Figure 16-39: 400 Level Compressor and Electrical Room

16.8.10 Refrigeration Plant and Brine Distribution

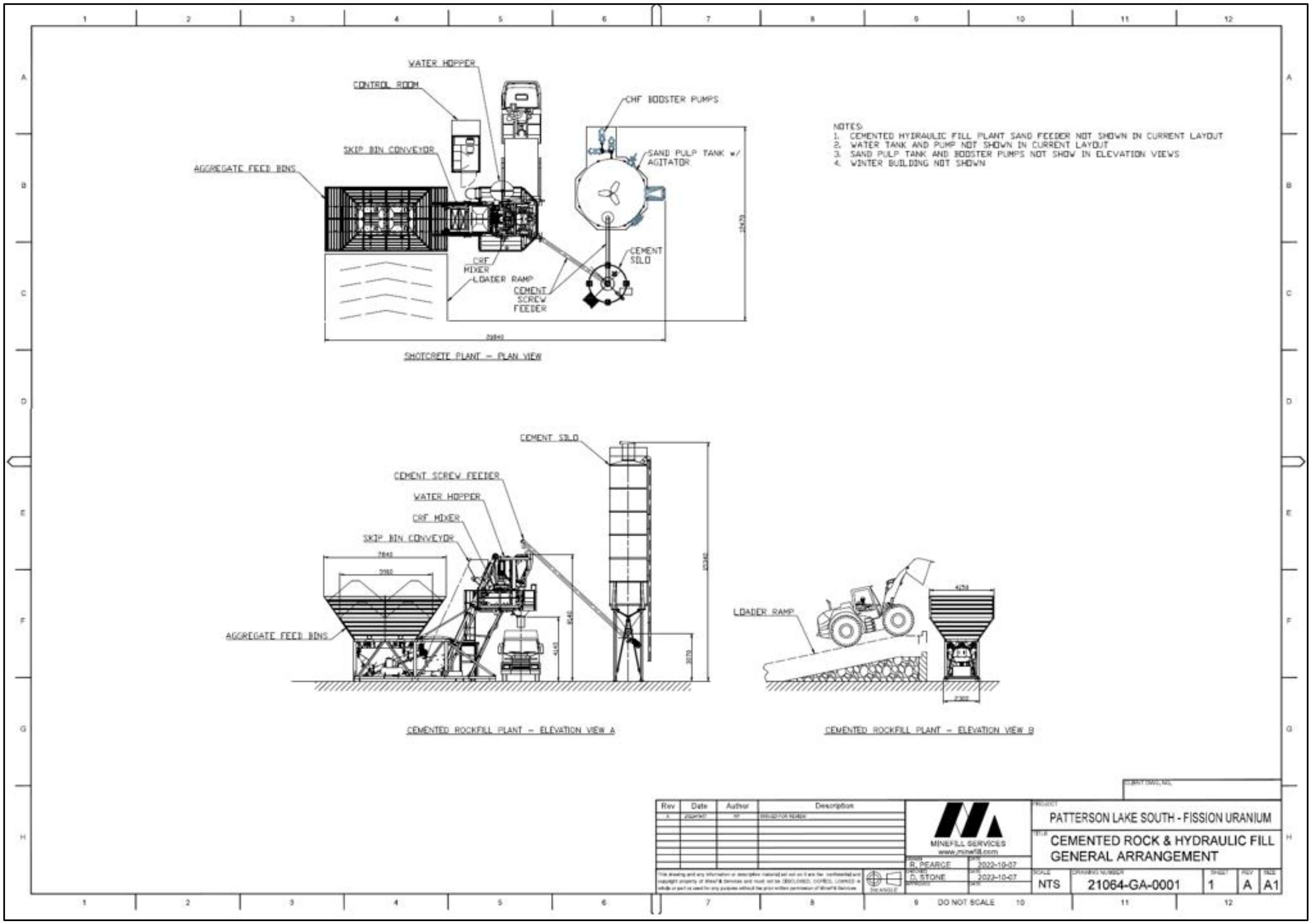
To facilitate mining of the crown pillar of the R780E and R00E, an artificial ground freezing method will be employed to stabilize the ground and limit water infiltration near the overburden-bedrock interface. A refrigeration plant will be located on surface near the FAS, which will pump a chilled 30% calcium chloride (CaCl) to and from surface via dedicated 10 inch diameter insulated pipes installed in the FAS. The brine solution will be fed to dedicated freeze drifts to a series of freeze pipes that have been installed in a fan pattern (four holes per fan) at 4 m fan spacing along the entire strike of the targeted orebodies.

The refrigeration plant will contain five operational skid-based compressors with a total capacity of 1,625 t refrigeration at -40℃ ammonia evaporating temperature (325 t refrigeration per unit). Each unit will consist of a 1,000 hp compressor and a 100 hp brine pump capable of feeding the surface brine mixing tank with 320 gpm of brine at a design temperature of -35°C. From the brine mixing tank, two 100 hp pumps (one operational, one standby) will feed up to 1,600 gpm of brine solution underground. The freeze plant condensers will be air cooled due to the northern climate which eliminates the need for a continuous water source.

16.8.11 Backfill Plant and Cemented Hydraulic Fill Reticulation

The backfill plant will be located adjacent to the mine portal and the surface waste stockpile area. The proposed plant is designed to produce both CRF and CHF with the sharing of facilities such as the feed hopper and cement handling system. The proposed plant can be arranged to supply both CRF and CHF concurrently in batches as needed. The CRF plant has also been designed to allow the batching of concrete for construction use when required.

The CRF will be batched in a pre-engineered packaged CRF plant consisting of three aggregate bins (one for local sand, one for coarse aggregate, and one for fine aggregate), a single cement silo and dosing system, and a twin shaft concrete mixer. A bottom discharge in the mixer will deliver the batched CRF directly into the back of 30 t trucks for haulage underground. The CHF plant is designed to be integral with the CRF plant and shares facilities such as the aggregate (sand) handling and the cement dosing system. After a short residence time in a mixing tank, the CHF slurry passes through a set of three centrifugal pumps in series before delivery down the main ramp in a 90 mm SRCP piping. Figure 16-40 shows the layout of the surface backfill plant.



Source: MineFill Services, 2022

Figure 16-40: PLS Surface Backfill Plant Layout

The backfill plant area will also have a mobile portable crushing and screening plant. The system will consist of a jaw and cone crusher which will be utilized to process mine development waste rock and surface aggregate. -50 cm ROM material will be loaded into a vibrating grizzly feeder with 15 cm openings. The +15 cm material is fed into the jaw crusher to be crushed. The -15 cm material falls through a screen onto the cone feed conveyer. Oversize material (greater than 50 cm) is rejected off the screen to a stockpile. An inclined screen sorts the -15 cm material. +75 mm material from the top deck of the screen will be fed into the cone crusher for further reduction and conveyed into a closed circuit conveyor back to the screen. -75 mm material is then sorted by the screen into two product stockpiles: coarse and fines. The coarse stockpile will be made up of -75 mm to +10 mm material while the fines stockpile will be made up of -10 mm material.

16.8.11.1 CHF Backfill Delivery

The delivery of hydraulic fill to backfill areas will be via 90 mm OD SRCP piping rated at 5 MPa. The primary route will be down the main ramp to the 400 level. From there the flow will be diverted to fill either the R840W or R780E deposits. It is assumed that all of the trunk piping will be down the main ramp. SRCP piping has been selected over mild steel piping because it is far easier and safer to install, it is flexible and can accommodate bends such as ramps without the use of fabricated bends, and it will wear less than unlined steel. The system runs at too high pressure to allow the use of HDPE piping.

From the backfill plant the CHF first passes through a series of three 2 inch x 1.5 inch centrifugal slurry pumps. The pipe will then traverse some 100 m from the backfill plant to the portal where it then follows the main ramp down to 400 level. A diverter valve is located on the 400 level to split the flow from the R840W zone to the R000 zone. Within the main ramp the piping is assumed to be fixed to the back with rigid hangers. Once at the mining sublevel the piping follows the level access into the stopes. The final leg of pipe into the stopes will be PN16 HDPE. The total length of the HDPE runs cannot exceed 500 m due to the pressure limitations.

Calculations show that most of the mine can be reached with a slurry at 68% to 71% solids with estimated friction losses of about 2.4 kPa/m. However, the flow models suggest that the underground routing for access to stopes at the R840W zone will require a fourth stage of centrifugal pumps operating at higher system pressures hence a surface route is suggested for these stopes. A surface line of about 450 m (including 40 m vertical) feeding a roughly 100 m long borehole is required. This line will need to be buried or wrapped in heat trace to prevent freezing during the winter months.

MineFill recommends the following future work:

- Additional flow modelling to optimize the CHF reticulation to crown pillar areas. The R840W may be able to be serviced by the same distribution network instead of a dedicated borehole. This will require a trade-off between additional pumping infrastructure and the cost of a dedicated backfill distribution line.

17.0 RECOVERY METHODS

The Mineral Resource at PLS Property is an example of a basement-hosted vein-type or fracture-filled uranium deposit. Typically, uranium is present as uraninite/pitchblende, which occurs as veins and semi-massive to massive replacement bodies.

The ore will be processed using a conventional grinding, leaching, counter-current decantation, SX and precipitation method producing a uranium concentrate. This section outlines the major design criteria and describes the unit processes of the flowsheet.

17.1 Flowsheet Development

Tetra Tech has completed the design for the process plant and related infrastructure facilities for this FS using tried and proven uranium extraction processes and equipment and has drawn on its knowledge of other Athabasca uranium plants, including Rabbit Lake, Key Lake, and McClean Lake.

The processing plant has been designed to process ore at a nominal throughput of 1,000 t/d to produce market-grade uranium concentrate. The average LOM mill feed grade will be 1.41% U_3O_8, and the anticipated U_3O_8 recovery will be 97.0%. The annual LOM average production will be approximately 4.8 million kg/a (10.6 million lb/a) of concentrate at 95% U_3O_8.

A conventional grinding and leaching circuit will be used for the recovery process. The ore will be trucked from the mine to the ROM pad and ground in an SSAG grinding circuit to 150 µm. The ground ore slurry will be leached using sulphuric acid and hydrogen peroxide at 50°C. The leached slurry will be fed to a counter-current decantation circuit followed by a clarification stage to produce the pregnant leach solution. An SX circuit will purify and concentrate uranium in the solution for yellowcake precipitation. Gypsum will be precipitated to reduce the sulphate content in the strong acid strip solution before precipitating the yellowcake. Hydrogen peroxide and magnesium oxide (MgO) will be used for yellowcake precipitation. The precipitated yellowcake will be calcined at 450°C before packaging in barrels.

The produced tailings will be neutralized and then deposited in the TMF. The effluent and any contact water will also be treated, monitored, and sampled as per the applicable environmental standards and regulations before being discharged into the environment.

A simplified overall process flowsheet is shown in Figure 13-1.

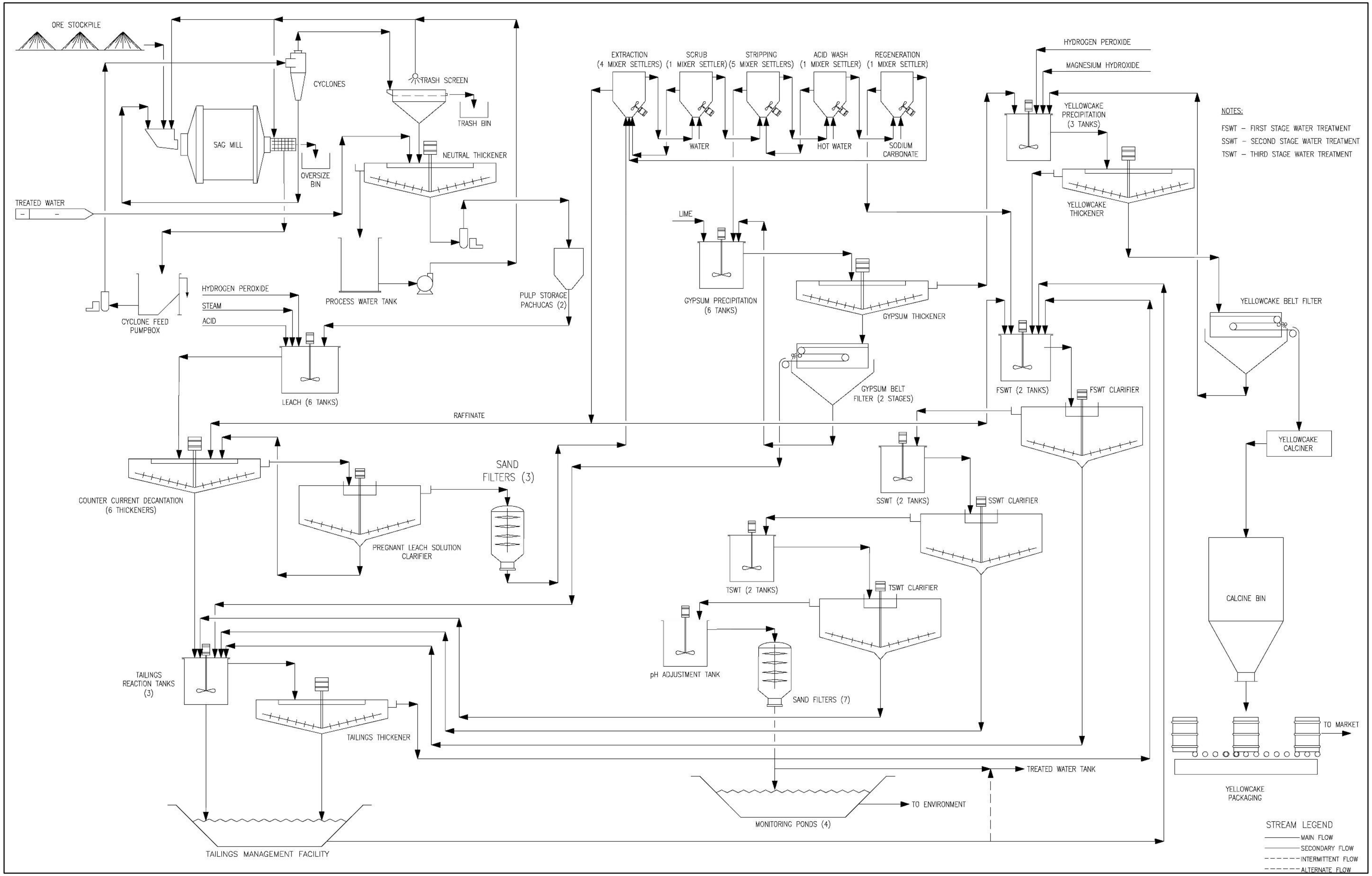


Figure 17-1: Simplified Process Flowsheet

17.2 Process Design Criteria

The processing plant is designed to process mill feed at a nominal throughput of 1,000 t/d for an average annual throughput of 350,000 t. The major criteria used for the processing plant design are listed in Table 17-1.

Table 17-1: Major Plant Design Criteria

Description	Unit	Value	Source
Ore Characteristics			
Specific Gravity	g/cm^3	2.64	4
Moisture Content	%	10	2
SAG Power Index (85^{th} Percentile)	minutes	30.4	4
Crusher Index (85^{th} Percentile)	-	4.78	4
Bond Ball Mill Work Index (85^{th} Percentile)	kWh/t	12.4	4
A_i (85^{th} Percentile)	g	0.129	4
Head Grade, LOM Average	% U_3O_8	1.41	5
Operating Schedule			
Shift/day	--	2	1
Plant hours/shift	h	12	1
Plant h/d	h	24	1
Days/year	days	350	1
Operation Parameters			
Overall Plant Feed	t/a	350,000	1
Overall Plant Feed	t/d	1,000	1
Plant Availability	%	90	2
Nominal Grinding Rate	t/h	46.3	3
Leaching Feed Size (80% Passing)	µm	150	4
Production @ LOM Average			
Recovery	%	97.0	3
Annual Concentrate Production	Million kg/a	4.8	3
Annual Concentrate Production	Million lb/a	10.6	3

Notes:
1. Client
2. Industry/Experience
3. Calculation
4. Test work
5. Mine Plan

The sizing and selection of the grinding mill is based on the comminution test work results. Leaching and subsequent circuits were sized based on optimum residence time determined by laboratory test work. The thickeners were sized based on settling test results with appropriate design factors.

17.3 Process Description

17.3.1 Ore Receiving and Storage

ROM ore will be trucked from the mine to the ROM stockpile adjacent to the processing plant. There will be a low grade stockpile near the ROM stockpile. The size of the low-grade stockpile will be limited because the ore mining rate is matched to the ore processing rate (1,000 t/d). The stockpile pads will be lined with impervious underlining. Run-off water will be collected, contained and sent to the water treatment pond.

Haul trucks will pause under a radiometric scanner which will determine the approximate uranium grade of the loads. Ore will be unloaded onto one of three designated grade sections on the pad. The three grade envelopes are listed in Table 17-2. The graded stockpiles will allow ore blending while reclaiming it to feed the processing plant.

Table 17-2: Designated Grade of ROM Pad Storage Sections

Description	Unit	Value
HG	% U_3O_8	>4.00
MG	% U_3O_8	0.25 to 4.00
LG	% U_3O_8	0.15 to 0.25

The loader will feed a predetermined ratio of the graded stocks to the mill feed bin so that a smoothed average grade is processed downstream, enabling predictive efficient uranium production. A mobile rock breaker will be available on the ROM pad to break rocks larger than 300 mm and loosen frozen ore. The loader will feed ore to a static grizzly screen covering a 60-t capacity ore feed bin. The mobile rock breaker will clear chokes and assist with flow through the screen when necessary. Ore will be drawn from the bottom of the bin by a variable speed belt feeder, which will feed directly to the SAG mill feed hopper. The belt feeder will be equipped with a weightometer and radiometric scanner for process and grade control.

17.3.2 Grinding and Classification

The ROM ore will be ground in a single-stage SAG mill. A 4.2 m diameter x 4.2 m equivalent grinding length SAG mill with 950 kW installed power will be used for grinding. The mill motor will be equipped with a variable frequency speed drive. The grate discharge tapered apertures will discharge pulp with particle top size 25 mm. Mill discharge pulp will be screened at 10 mm using SAG mill trommel. Oversize will be collected in oversize bins and recycled to the mill or discarded to the waste rock storage facility, depending on the uranium grade. Mill discharge will then be pumped to a 500-mm classifying cyclone (one operating and one standby). Cyclone underflow will gravitate to the mill feed hopper and enter the mill along with the fresh ore feed. Pulp density inside the mill will be controlled at 72% (by wt.) by adding dilution water at the mill feed hopper. Cyclone overflow at P_{80} of 150 µm will gravitate via a trash screen to the pre-leach thickener, referred to as the neutral thickener.

17.3.3 Pulp Screening, Thickening, and Storage

Cyclone overflow with 80% particles passing 150 µm will gravitate to a trash removal screen with 1 mm x 10 mm slotted apertures. Spray water will wash the retained debris (wood and plastic fibres) before it discharges off the deck into a bin for disposal. Screen underflow will gravitate via a launder to the neutral thickener feed well.

The neutral thickener (20 m diameter) will thicken the milled pulp to ±52% w/w solids. Flocculant will be added as required. Overflow water will gravitate to the process water tank for recycling to the SAG mill. SAG mill circuit make-up water sourced from the water treatment pond or tank will be added as required. Thickener underflow will be pumped to the pre-leach pulp storage air-agitated pachucas. The thickener and the pachucas will enable additional smoothing of the uranium head grade reporting to the leaching circuit and decoupling the grinding circuit from the leaching circuit.

The pre-leach storage tanks will consist of two air-agitated conical bottom pachucas (6 m diameter x 14 m height). The tanks will provide up to 12-hour surge capacity. Pulp will be pumped to the leaching circuit at a nominal rate of 46 t/h at 52% (w/w) solids density.

17.3.4 Leaching

The front end of the plant will deliver steady-state feed to the leaching circuit, enabling leach efficiency gains through optimal leach residence time, temperature profile, and reagent control practices.

Atmospheric leaching will be carried out in a train of six mechanically agitated rubber-lined carbon steel tanks (5 m diameter x 8 m height, 150 m^3 each) equipped with dual axial flow impellers. Pulp will flow by gravity cascading from the no. 1 leach tank sequentially through to the no. 6 leach tank. It will be possible to bypass any tank for maintenance and select residence by operating five or six tanks. The tanks will be closed top vented via a scrubber to the atmosphere to minimize worker exposure to acidic steam and radon emissions.

Steam will be sparged into the first and second tanks to maintain an operating temperature of 50°C. Sulphuric acid (98%) will be dosed primarily in the first tank and, to a lesser extent, in the second tank to achieve a free acid content in an exit solution of approximately 15 g/L. Hydrogen peroxide (50% H_2O_2) will be dosed to the second tank to maintain an oxidation-reduction potential (ORP) of roughly 480 mV.

Under these conditions, hexavalent uranium is dissolved by sulphuric acid to form uranyl sulphate and uranyl sulphate anions (Equations 1 to 4). Tetravalent uranium is insoluble and must be oxidized by ferric ion (Fe^{3+}) to the soluble hexavalent form (Equations 5 and 6). Hydrogen peroxide oxidizes ferrous (Fe^{2+}) to ferric (Fe^{3+}) ions. Sufficient natural iron mineralization is present in the ore to produce ferrous ions in the first leach tank, which is then available to react with hydrogen peroxide in the second leach tank. Reagent addition will be controlled to result in discharge pulp with 98% uranium dissolution at pH 1.5 and free acid content of 15 g/L H_2SO_4.

Leach reactions are described in the following chemical equations:

$$UO_3 + 2H^+ \leftrightarrow UO_2^{++} + H_2O \qquad \text{Eq. 1}$$

$$UO_2^{++} + SO_4^{2-} \leftrightarrow UO_2SO_4 \qquad \text{Eq. 2}$$

$$UO_2SO_4 + SO_4^{2-} \leftrightarrow [UO_2(SO_4)_2]^{2-} \qquad \text{Eq. 3}$$

$$[UO_2(SO_4)_3]^{2-} + SO_4^{2-} \leftrightarrow [UO_2(SO_4)_3]^{4-} \qquad \text{Eq. 4}$$

$$2Fe^{2+} + H_2O_2 + 2H^+ \leftrightarrow 2Fe^{3+} + 2H_2O \qquad \text{Eq. 5}$$

$$UO_2 + 2Fe^{3+} \leftrightarrow UO_2^{2+} + 2Fe^{2+} \qquad \text{Eq. 6}$$

Discharge slurry density will be approximately 45% solids. Solids weight loss will be approximately 6% to 7%. The final leach tank will overflow to a pump box and then be pumped to the no. 1 CCD feed mix tank.

17.3.5 Counter Current Decantation Thickeners

There will be six CCD thickeners (24 m diameter), each with its own feed mix tank. Thickened pulp will be pumped from the cone of each thickener to the mix tank feeding the next downstream thickener. Thickener overflow will be pumped to the mix tank feeding the upstream preceding thickener. The mixed diluted slurry will flow by gravity from the mix tank overflow to the centre well of the thickener. Flocculant will be added into the flow from the mix tank overflow launder. Internal auto-dilution will enable density control of the feed to the centre well at 3% to 4% solids for effective flocculation.

Each thickener will operate at equal underflow density ≥ 48% solids (w/w). Wash water comprised of raffinate and recycled water will be added to the no. 6 mix tank to achieve a counter current wash ratio of 3 m^3 of wash solution per t of leach feed.

The pH in the CCD no. 6 thickener feed well will be controlled to pH 1.5 by adding acid when necessary to prevent possible uranium precipitation at higher pH. The CCD circuit dissolved uranium recovery will be 99.5% to 99.7%.

The mix tanks (12 m^3 each) will be elevated and positioned so that the overflow will gravitate on a slight gradient with minimal turbulence (no air entrainment) to the thickener centre well and discharge tangentially just below the surface (as defined by vendor specifications).

17.3.6 Pregnant Leach Solution Clarification

Cloudy pregnant leach solution containing approximately 100 ppm suspended solids will overflow from the no. 1 thickener and be pumped to a reactor clarifier (16 m diameter). Coagulant and flocculant will be added at dosages suitable for the degree of turbidity. Settled solids will be pumped from the cone of the reactor clarifier to the no. 1 CCD mix tank.

Clarifier overflow pregnant leach solution containing approximately 50 ppm suspended solids will be pumped to three downflow sand filters (3 m diameter) for polishing. A single pass through a sand filter will reduce the pregnant leach solution-suspended solids to <10 ppm. The sand filters will be backwashed and scoured periodically using clarified pregnant leach solution and compressed air to clean the sand particles. Dirty backwash solution will be collected in a backwash tank and pumped to the clarifier.

Polished pregnant leach solution will be pumped to a 570 m^3 surge tank prior to forwarding to the SX circuit. The pregnant leach solution will be pumped from the surge tank through a steam heat exchanger to raise the temperature to 40°C to the SX section at a nominal flow rate of 139 m^3/h.

17.3.7 Solvent Extraction

The purpose of the SX circuit is to purify and concentrate uranium in the solution for yellowcake precipitation. The SX process is continuous, with a runtime typically of >92% per year. The SX has been designed to process up to 140 m^3/h of pregnant leach solution at a uranium grade ranging from 1.6 g/L to 6.4 g/L U_3O_8. SX uranium recovery will be 99.6%. A small amount of uranium is lost to the raffinate bleed stream and spent regen solution.

The following equations describe the SX stage chemistry (note: R_3N = tertiary amine):

Extraction $$2(R_3NH)_2SO_{4\ org} + UO_2(SO_4)_3^{4-}{}_{aq} \leftrightarrow (R_3NH)_4UO_2(SO_4)_{3\ org} + 2SO_4^{2-}{}_{aq}$$

Stripping $$(R_3NH)_4UO_2(SO_4)_{3\ org} + 4H^+ + 4HSO_4^+{}_{aq} \leftrightarrow 4(R_3NH)(HSO_4)_{org} + 4H^+ + UO_2(SO_4)_3^{4-}{}_{aq}$$

Water wash $$(R_3NH)_2(HSO_4^-)_{2\ org} \leftrightarrow (R_3NH)_2SO_{4\ org} + 2H^+ SO_4^{2-}{}_{aq}$$

Regeneration $(R_3NH)_2MoO_{4\ org} + Na_2CO_{3\ aq} \leftrightarrow 2R_3N_{\ org} + 2Na^+ + MoO_4^{2-}{}_{aq} + H_2O + CO_2\uparrow$

Extraction Stage

As per the outcome of laboratory tests and extraction isotherms, four MS will be used to extract uranium from the clarified pregnant leach solution. The organic solution is comprised of 5% to 7% (v/v) tertiary amine, an extractant highly selective for uranium; 5% to 7% (v/v) isodecanol phase separation accelerator (third phase inhibitor), and the balance 86% to 90% (v/v) kerosene as diluent and carrier.

Mixing of organic and aqueous solutions will occur in the mechanically stirred mixer box compartment of each MS. The mixed emulsion will overflow into the settler box for phase separation. Phase separation efficiency will be optimized by controlling pregnant leach solution temperature at 37 to 40°C using a steam heat exchanger on the incoming flow, maximizing solution density differential (typically aqueous 1.0 to 1.1 t/m^3, organic 0.8 t/m^3), maintaining consistency of clarity of pregnant leach solution with <10 ppm suspended solids including colloidal silica to minimize interphase crud formation and designing solution velocity and residence time in the settler compartments as determined by test work.

Barren organic enters no. 4 MS and contacts aqueous from no. 3 MS. Barren organic will consist of 80% recycled organic, which is protonated. The balance of organic will be regenerated deprotonated organic, which will be re-protonated by contact with acidic aqueous in the first MS. Forward organic flow will be 80% of the pregnant leach solution flow. Organic will be recycled in each MS for continuous mixing in the mixing box at a ratio of 1.5 organic to 1 aqueous. Organic exiting no. 4 MS will report to no. 3 MS. The aqueous exiting no. 4 MS is depleted of uranium and is now referred to as raffinate.

The raffinate will contain entrained organic droplets, which will be recovered in a coalescer after-settler. The raffinate will also contain most of the free acid, dissolved iron, and other dissolved deleterious elements leached from the ore. Approximately 15% of the raffinate will be recycled to supplement the CCD wash solution, utilizing the contained acid. The remainder will be pumped to the effluent treatment section to ensure that there is no build-up of deleterious elements in the SX circuit.

Scrubbing Stage

One scrub mixer settler will be used to wash the loaded organic exiting the extraction section with water. The purpose of the scrub is to remove entrained aqueous droplets from the loaded organic, thereby minimizing the possibility for deleterious elements to pass through to yellowcake precipitation. Spent scrub aqueous will report to extraction no. 1 MS. The ore does not contain significant arsenic; therefore, extensive scrubbing to remove arsenic from loaded organic is not required.

Stripping Stage

The stripping MS will be a series of five units. The strip liquor will be strong sulphuric acid at 430 g/L H_2SO_4 entering no. 5 MS and advancing counter current to the organic flow until exiting no. 1 MS as a pregnant strip solution. The acid strength will reduce as the aqueous advances and will be countered by the addition of concentrated sulphuric acid in no. 1 and no. 2 MS (as needed) to drive uranium transfer to the aqueous phase. The advance flow rate for organic will be 111 m^3/h and approximately 5.6 m^3/h for aqueous. The O/A ratio in the mixer is maintained between 3.0 and 3.5 by aqueous recycle. The pregnant strip solution will report to an organic recovery after-settler prior to gypsum precipitation.

Barren Organic Wash and Regeneration

Barren organic will be water-washed in a single-stage mixer settler for acid recovery. The wash water will be heated in a steam heat exchanger to 40°C before entering the mixer box to raise the temperature of the barren organic before it is recycled to the extraction section, which operates at 40°C. The acid aqueous exiting the wash settler will be pumped to the strip solution make-up tank. The washed barren organic will be pumped to the barren organic tank and recycled to the extraction section. Approximately 50% of the organic flow will be regenerated by contact with sodium carbonate in a regeneration MS to prevent the build-up of contaminants such as molybdenum. Spent regen aqueous will be pumped to effluent treatment.

As a fire control measure, all equipment in the SX plant area will be equipped with organic drains that dump organic into an external underground emergency organic dump tank.

17.3.8 Gypsum Precipitation and Dewatering

The purpose of gypsum precipitation is to reduce sulphate content in the strong acid strip solution before precipitating yellowcake. Milk of lime will be dosed to the pregnant strip solution to react with the sulphate and form gypsum precipitate as the chemical equation:

$$H_2SO_4 + Ca(OH)_2 \rightarrow CaSO_4 \bullet 2H_2O$$

Gypsum precipitation will be carried out in a series of six mechanically stirred tanks (35 m^3 each), providing six hours of residence time. pH will be gradually raised from 1 to a terminal pH of 3.4 as the solution flows by gravity overflow down the series of reactors. A recycle stream of gypsum thickener underflow will be directed to precipitation tank no. 1 to provide an abundance of gypsum seed crystals for nucleation. The flow from the sixth reactor will be pumped to a gypsum thickener (13 m diameter). Thickened gypsum slurry will be pumped from the thickener underflow to a vacuum belt filter (40 m^2). Wash water will be added to the filter cake in the wash zone to displace the uranium solution. The filtered cake will be repulped with acid water and pumped to a two-stage two-hour uranium leach to dissolve uranium that may be associated with the gypsum and washing of uranium solute off the gypsum particles. The re-leached slurry will be pumped to a second belt filter (40 m^2) to recover dissolved uranium in the filtrate. Wash water will be added to the wash zone on the filter. Filter cake will be repulped and pumped to the tailings neutralization circuit. The filtrate containing recovered uranium from the second belt filter will be returned to the first belt filter feed tank, whereas the filtrate from the first belt filter will be pumped to no. 1 gypsum precipitation reactor as a heat diluent or to no. 1 CCD, depending on operational requirements.

17.3.9 Yellowcake Precipitation and Dewatering

The overflow solution from the gypsum thickener will gravitate into a surge tank (30 m^3) which will also act as a settling tank for the retention of any fine gypsum particulates that overflow the thickener. Uranium solution will be pumped from the surge tank through a heat exchanger to maintain a temperature of 25°C to the first of two in series mechanically stirred uranium peroxide precipitation reactors (35 m^3 each). A recycle stream of yellowcake thickener underflow will be directed to precipitation tank no. 1 to provide an abundance of uranium peroxide seed crystals for nucleation.

Hydrogen peroxide will be dosed at a rate of 0.25 kg H_2O_2/kg U_3O_8. The pH will be controlled at 3 by adding MgO. The residence time for uranium peroxide precipitation will be four hours. The uranium precipitation reactions are as follows:

$$UO_2SO_4 + H_2O_2 \rightarrow UO_4.xH_2O + H_2SO_4$$

$$H_2SO_4 + MgO \rightarrow H_2O + MgSO_4$$

Yellowcake pulp will gravitate from the overflow of the no. 2 reactor to a wash tank and then be pumped in the centre well of the yellowcake thickener (13 m diameter). Thickened yellowcake pulp will be pumped from the thickener underflow to a surge tank. Yellowcake will be filtered and washed on a belt filter (8 m^2). The filtrate will be returned to the yellowcake thickener. Washed yellowcake will be discharged from the filter into a calciner feed hopper.

Yellowcake thickener overflow will be removed from the circuit as the barren solution. The barren solution will be pumped to two downflow sand filters (1.3 m diameter) to recover any residual yellowcake solids. The sand filters will be backwashed and scoured periodically, and the backwash solution will report to the yellowcake wash tank. The sand filter filtrate will be stored in a barren strip tank and pumped to the SX strip solution make-up tank, or the YC calciner off-gas scrubber, or the effluent treatment plant as needed.

17.3.10 Yellowcake Calcining and Packaging

Yellowcake will be extracted from the hopper by a ribbon screw feeder and discharged into the rotary calciner (1 m diameter x 10.3 m length) at a moisture content range of 15% to 30%. The rotary calciner will be indirectly heated using liquified natural gas. The nominal peak operating temperature will be 475°C, achieving a chemically stable product enriched to 95% U_3O_8 and having a packing density of 1.85 kg/L.

The calciner will produce two gas streams. The uncontaminated combustion gas will discharge through a stack, and the contaminated process gas will be drawn through a wet scrubber before releasing through a stack. Scrubber effluent will be pumped to the yellowcake thickener to recycle the calciner dust.

Calcined yellowcake will discharge into a bucket elevator and discharge into a storage bin with a three- to four-day production capacity. The bin feed and product transfer points are kept under negative pressure by an induced draft fan that will draw air and dust through a dust capture baghouse. Captured dust will be discharged to the calcine bin.

Calcined yellowcake will be metered from the bottom of the storage bin into 210 L steel drums conveyed on a semi-automated packaging system. The system will be sealed to prevent any dust from contaminating the area. The drums will be sampled before lids and seal rings are fitted. The drums will then be washed and dried. After being weighed, an ID label will be attached that includes the drum tare and total weight as well as the net weight of the contained product. The normal net weight of a drum will be about 350 kg. There will be 40 to 60 drums packaged per plant operating day.

A drum storage facility will provide sufficient room to store nested empty drums and 200 full drums. Full drum consignments will be trucked to the refinery regularly.

17.3.11 Tailings Neutralization

The CCD circuit underflow, filtered gypsum, and first-stage water treatment (FSWT) clarifier underflow will be pumped to the no. 1 tailings reaction tank (150 m^3), where lime, ferric sulphate, and barium chloride will be added to neutralize (increase pH to 4.5) the pulp and precipitate dissolved contaminants. The overflow from the no. 1 tailings reaction tank and the second-stage water treatment (SSWT) clarifier underflow will report to the no. 2 tailings reaction tank (150 m^3), where lime will be added to increase pH to 7. The overflow from no. 2 tailings reaction tank and the third-stage water treatment (TSWT) clarifier underflow will report to tailings reaction tank no. 3 (150 m^3), where final lime will be added to increase pH to 10. Each tank will provide one hour of residence time.

Neutralized tailings slurry will be pumped from the third reaction tank into the feed well of the tailings thickener (20 m diameter). The tailings thickener underflow will be pumped to the TMF, and the thickener overflow will be sent to the first stage of effluent treatment.

17.3.12 Effluent Treatment

Site effluents are treated to remove elements of concern to produce water suitable for release to the environment or recycling for site use. The treated effluent quality must comply with all applicable standards and regulations.

The effluent treatment plant will process the raffinate, spent regen, barren solution, tailings thickener overflow, underground mine water discharge, and the site run-off from potentially contaminated areas. The feed settling pond and run-off ponds will provide surge capacity that allows the effluent treatment plant to be fed at a constant flow rate. The feed water will be pumped to the FSWT tanks (250 m^3 each). There will be two tanks in a series to provide a reaction time of one hour. Milk of lime will be dosed to the acidic effluent to begin the neutralization and precipitation of metal hydroxides. The final pH from the FSWT will be 4. Barium chloride will be added to precipitate radium, and ferric sulphate will be added to precipitate molybdenum and traces of arsenic and selenium in the effluent. The second tank overflow will be pumped to the FSWT clarifier (16 m diameter), where precipitates will settle out with the aid of a flocculant and report to the tailings reaction tank no.1.

The FSWT clarifier overflow will be pumped to the SSWT tanks consisting of two mechanically agitated tanks (250 m^3 each) in series, providing one hour of reaction time. Again lime, ferric sulphate, and barium chloride will be added to cause further precipitation of contaminants at pH 7 controlled by trim addition of sulphuric acid if required. The second tank overflow will be pumped to the SSWT clarifier (16 m diameter), where precipitates will settle out with the aid of a flocculant and report to the tailings reaction tank no.2.

The SSWT clarifier overflow will be pumped to the TSWT reactor tanks (250 m^3 each). For the third time, lime, ferric sulphate, and barium chloride will be added to cause the final precipitation of contaminants at pH 10, controlled by trim addition of sulphuric acid if required. The second tank overflow will be pumped to the TSWT clarifier (16 m diameter), where precipitates will settle out with the aid of a flocculant and report to the tailings reaction tank no.3.

Water from the TSWT clarifier overflow tank will be polished to remove fine suspended solids by pumping through the effluent sand filters. The filtered water will be pumped to the pH adjustment tank (500 m^3), where dilute sulphuric acid will be added to bring the pH down to 7 (neutral). The treated water will be recycled as make-up water as needed to reduce the amount of fresh water usage. Excess water will be pumped into one of the three monitoring ponds for quality control before discharging into the environment.

17.3.13 Feed and Effluent Monitoring Ponds

Mine water from underground sumps will be discharged to a feed settling pond on surface. The pond can contain four to five days of normal mine water discharge. As water is retained in the pond, suspended solids settle. Surface run-off water from potentially contaminated site areas such as the ore storage pads, and potentially contaminating uses such as truck and maintenance shops, will also discharge to the feed settling pond. Water may be pumped from the feed settling pond into the process/treated water tank. Excess water is pumped from the feed settling pond to FSWT. The flow of water to FSWT is maintained at a prescribed flow rate.

Effluent monitoring ponds will store excess treated effluent until water parameters are assayed and confirmed to meet discharge criteria. The flow is sampled when a pond receives water from the pH adjustment tank. Once a monitoring pond is full, a composite water sample representing a full pond is taken and assayed for all parameters of concern. Once the assays are confirmed to be within the required ranges, the pond will be discharged into the environment. If assays are not within the required ranges, the water will be pumped back to the feed settling pond for reprocessing. A monitoring pond will hold about 24 hours of treated effluent at the nominal flow rate.

17.4 Reagents and Consumables

The type of reagents and major consumable consumption rate (at an average LOM grade of 1.41% U_3O_8) for the process plant are summarized in Table 17-3.

Table 17-3: Reagents and Major Consumables Consumption

Reagent/Consumable	Consumption (t/d)
Sulphur	50.95
Hydrogen peroxide (50% H_2O_2)	19.20
Sodium carbonate	1.39
MgO	3.03
Barium chloride	0.58
Ferric sulphate (60% solution)	9.54
Lime	66.24
Flocculant (Magnafloc 333 or eq.)	0.35
Flocculant (Magnafloc 10 or eq.)	0.02
Alamine	0.08
Isodecanol	0.07
Kerosene	0.41
Grinding Media	0.54

17.4.1 Reagent Handling and Storage

To ensure containment in the event of an accidental spill, all reagents will be prepared and stored in a separate, self-contained area within the designated building and will be designed to accommodate 110% of the content of the largest tank. The reagents will be delivered to the required addition points by individual metering or centrifugal pumps. All the reagents will be prepared using treated and filtered water. The reagent system will include unloading and storage facilities, mixing tanks, transfer pumps, and feeding equipment. The storage tanks will be equipped with level indicators and instrumentation to ensure that spills do not occur during normal preparation operations. Appropriate ventilation, fire, and safety protection will be provided at the facility. Material safety data sheets (MSDS) will be provided to the operating staff as a training and reference source. Each tank, reagent line, and addition point will be labelled following the Workplace Hazardous Materials Information Systems (WHMIS) standards. All operational personnel will receive WHMIS training and additional training for the safe handling and use of the reagents.

Sulphur

Sulphur (for acid preparation) will be received in solid form in bulk tankers and stored in a dedicated area. It will be melted in the sulphur melting unit and stored in a vendor-supplied tank after impurity removal. It will then be used in the sulphuric acid plant to produce the acid needed. The concentrated sulphuric acid produced will be stored in carbon steel tanks and delivered by metering pumps. The containment area will be equipped with sumps and pumps to recycle any spillage.

Hydrogen Peroxide (50% H_2O_2)

Hydrogen peroxide will be received in liquid form in bulk tankers and stored in stainless steel tanks. The hydrogen peroxide will be added without dilution by metering pumps to the leaching and yellowcake precipitation circuit.

Sodium Carbonate

Sodium carbonate will be delivered in bulk bags. It will be unloaded in a hopper equipped with a bag breaker and dust collector. The sodium carbonate will be mixed with treated water in a mixing tank at 5.2% solids density and then stored in a holding tank and distributed via metering pumps to the SX circuit.

Magnesium Oxide

MgO will be delivered in bulk bags. It will be unloaded in a hopper equipped with a bag breaker and dust collector. The MgO will be mixed with treated water in a mixing tank at 10% solids density, then stored in an agitated holding tank and distributed via metering pumps to the yellowcake precipitation circuit.

Barium Chloride

Barium chloride will be delivered in bulk bags. It will be unloaded in a hopper equipped with a bag breaker and dust collector. The barium chloride will be mixed with treated water in a mixing tank at 10% solids density, then stored in a holding tank and distributed via metering pumps to the water treatment circuit.

Ferric Sulphate (60% Solution)

Ferric sulphate will be received in liquid form in bulk tankers and stored in an FRP tank. Ferric sulphate will be added without dilution by metering pumps to the water treatment circuit.

Lime

Quick lime (>90% CaO) will be delivered by bulk tankers and stored in a dedicated silo. It will be retrieved from the silo by a screw conveyor and slaked. The slaked lime slurry at 20% solids will be stored in an agitated tank and distributed via a pressurized lime loop throughout the processing plant.

Flocculant (Magnafloc 333 or eq. and Magnafloc 10 or eq.)

Flocculants will be received in bags on site. A flocculant screw feeder will feed the flocculant eductor using freshwater addition. The mixed solution will be transferred and stored in a holding tank. The packaged flocculant mixing system will run automatically based on the solution level of the holding tank. The flocculant will be made up to 0.5% solution strength and added via metering pumps to the thickeners and clarifiers.

Alamine

Alamine will be received in liquid form in an intermediate bulk container and stored in an FRP tank. Alamine will be added without dilution by metering pumps to the barren organic tank.

Isodecanol

Isodecanol will be received in liquid form in an intermediate bulk container and stored in an FRP tank. Isodecanol will be added without dilution by metering pumps to the barren organic tank.

Kerosene

Kerosene will be received in liquid form in bulk tankers and stored in an FRP tank. Kerosene will be added without dilution by metering pumps to the barren organic tank.

Other Reagents

Antiscalants, as required, will be added to minimize scale build-up in water lines. This reagent will be delivered in liquid form and metered directly into the intake of the water pumps.

New reagents will occasionally be tested to determine their effect on metal recovery and concentrate grading. These reagents will be handled in accordance with MSDS requirements. A facility for mixing and dosing these test reagents will be provided.

17.4.2 Consumables

The major consumable items for the comminution circuit will be grinding media and mill liners. 125 mm grinding media will be used in the SAG mill. Liners are an essential component of the SAG mill. Other consumables include screen decks, filter cloths, concentrate drums, and laboratory supplies. Maintenance spares for the processing plant and assay laboratory will also be provided.

17.5 On-Line Sample Analysis

The plant will rely on automatic sampling and analysis of various streams. The system will provide the necessary information for process control and sufficient sample quantities for checking, standardization, and possible metallurgical test work. The process streams that will be sampled periodically are:

- Cyclone overflow
- Leaching feed
- Leaching discharge
- CCD thickener residue
- SX circuit feed
- Raffinate solution
- Pregnant strip solution
- Washed gypsum residue
- Barren strip solution
- Calciner feed
- Calcined yellowcake

- Tailings thickener underflow
- Treated water

The information obtained from these samplers will provide metal recoveries and grades of all process streams, thus enabling an overall process metal and performance balance. The analyzed and excess sample slurries will be collected in the sample return tank and returned to the slurry feed stream to the leaching circuit.

17.6 Assay and Metallurgical Laboratory

An assay laboratory will provide all the routine assays for the mine, the processing plant, and the environmental and geological departments. The main instruments will include:

- AAS
- X-ray fluorescence analyzer
- Gamma-ray spectrophotometer
- ICP-MS
- Pycnometers

The metallurgical laboratory will undertake all necessary tests to monitor metallurgical performance and, more importantly, to improve process flowsheet and efficiency. The laboratory will be equipped with:

- Laboratory jaw and cone crusher
- Dust collection system
- Laboratory ball mill
- Ring and puck pulverizer
- Ro-Tap® sieve shaker and test sieves
- Oven-style moisture determination equipment
- Sedimentation devices and laser particle sizer
- SX shakeout glassware
- pH meters
- Bench precipitation reactors
- Convection oven
- Weighing devices
- Filtering units (pressure/vacuum filters)
- Fume hoods with extraction fans
- Bulk sample preparation equipment, including drying ovens, laboratory glassware, and reagents

Appropriate samplers will be available for routine bulk sampling and plant surveys for process control and metallurgical accounting.

17.7 Water Supply and Compressed Air

17.7.1 Fresh and Potable Water Supply System

A freshwater supply system will be installed to provide fresh and potable water to the mine and process plant. Freshwater will be pumped from wells near the process plant, supplied to a freshwater storage tank, and distributed by pumping. All the freshwater pipelines outside heated buildings will be buried below the freezing level. Freshwater will be used primarily for the following:

- Firewater for emergency use
- Potable water supply

The freshwater tank will always be full and capable of providing at least two hours of firewater in an emergency. The potable water will be treated via chlorination and ultraviolet lamps and stored in a tank before delivery to various service points.

17.7.2 Treated Water

Treated water will consist of the treated effluent from the water treatment plant (WTP). The water will be directed to a treated water storage tank and pumped to the distribution points in the processing plant. As with fresh water, treated water supply and distribution pipelines outside the heated buildings will be buried below the freezing level. Treated water will be used as reagent make-up water and gland seal water.

17.7.3 Air Supply

Separate air service systems will supply air to the following areas:

- Storage pachucas: Dedicated air compressors will provide the high-pressure air required for air agitation in the storage pachucas.
- Plant services: High-pressure air will be supplied by two separate air compressors.
- Instrumentation: Instrument air will be generated at plant sites separately from two dedicated oil-free air compressors, which will be dried and stored in a dedicated air receiver.

17.8 Process Control and Instrumentation

The plant control system will consist of a distributed control system (DCS) with personal computer (PC)-based operator interface stations (OISs) located in control rooms. In conjunction with the OIS, the DCS will perform all equipment and process interlocking, control, alarming, trending, event logging, and report generation. DCS input/output (I/O) cabinets will be located in electrical rooms and interconnected via a plant-wide fibre-optic network.

A separate control system/control room will be required in the power generation plant with components provided by the power generation system vendor.

Field instrumentation will consist of microprocessor-based "smart" type devices. Instruments will be grouped into process areas and wired to local field instrument junction boxes within those areas. Signal trunk cables will connect the field instrument junction boxes to DCS I/O cabinets.

Intelligent-type motor control centres (MCCs) will be located in the electrical rooms throughout the plant. The MCC remote operation and monitoring will be via DeviceNet (or other approved industrial communications protocol) interface to the DCS.

Programmable logic controllers (PLCs) or other third-party control systems supplied as a part of mechanical packages shall be interfaced with the plant control system via ethernet network interfaces.

A supervisory expert control system is proposed to control product particle size and optimize the fresh mill feed tonnage in the grinding circuit. Expert supervisory control will be developed to optimize the set points for controllers at the regulatory level. Mill solid concentration variable-ratio control, dilution water flow rate control, and level control will be carried out at the regulatory level to reach the control targets. The set-point modification by expert control for the dilution water controller will provide optimal dynamic performance. The set-point adjustment for the feed rate controller will ensure long-term stability in the particle size even if the mill feed hardness should change.

The plant control room will be staffed by trained operations personnel 24 hours daily. A central control room in the process plant will be provided with three OISs. Control and monitoring of all processes in the process barges will be conducted from this location. Control and monitoring functions will include, but are not limited to, the following:

- Grinding mill belt feeder (zero-speed switches, side-travel switches, emergency pull cords, belt scales, metal detectors, and plugged chute detection)
- Grinding mill (mill speed, bearing temperatures, lubrication systems, motors, and feed rates)
- Grinding particle size monitoring and control by particle size analyzers for the primary and secondary grinding circuits
- Pump box, tank, and bin levels
- Variable speed pumps
- Cyclone feed density controls
- Leach and precipitation tanks (level controls, pH, reagent addition, and others)
- X-ray analyzers and samplers
- Thickeners (drives, slurry interface levels, underflow density, and flocculants addition)
- Belt filters
- Reagent handling and distribution systems
- Tailings disposal system
- Water storage, reclamation, and distribution, including tank-level automatic control (via ethernet remote I/O)
- Air compressors
- Instrumentation packages

17.8.1 Remote Monitoring

Closed-circuit television (CCTV) cameras will be installed throughout the process plant, with monitors in the control room. The CCTV monitoring locations will include ROM Pad, SAG mill, calciner, yellowcake packaging area, and concentrate loadout area.

17.9 Annual Production Estimate

The annual production estimate will vary according to the ramp-up, mining production plan, and metallurgical performance outlined in Section 13. The annual production estimates based on the mine production plan and metallurgical performance outlined in Section 13 are presented in Table 17-4.

Table 17-4: Projected Concentrate Production

Year	Ore Processed (t)	Head Grade (% U_3O_8)	Overall Recovery (%)	Yellowcake Grade (% U_3O_8)	Recovered U_3O_8 (million lb)
Year 1	217,303	1.38	97.0	95.0	6.40
Year 2	340,686	1.32	97.0	95.0	9.59
Year 3	351,186	1.68	97.0	95.0	12.59
Year 4	358,996	1.15	96.9	95.0	8.81
Year 5	357,670	1.29	97.0	95.0	9.84
Year 6	348,675	1.87	97.0	95.0	13.98
Year 7	311,574	1.83	97.0	95.0	12.23
Year 8	361,360	1.11	96.9	95.0	8.56
Year 9	215,252	1.15	96.9	95.0	5.31
Year 10	144,786	1.15	96.9	95.0	3.56
Total	**3,007,487**	**1.41**	**97.0**	**95.0**	**90.86**

Note: Numbers may not add due to rounding

17.10 Staffing

Personnel requirements are developed based on the operational requirements, shift, equipment attendance, safety, training, and maintenance requirements. Average annual process plant staffing requirements are summarized in Table 17-5. The staffing is based on two 12-hour shifts per day on a two-week fly-in/fly-out basis.

Table 17-5: Plant Staffing Requirements

Area	Personnel Required
Management	10
Operations	44
Metallurgical and Assay Laboratory	11
Process Plant Maintenance	32
Total	**97**

18.0 PROJECT INFRASTRUCTURE

18.1 Overview

The PLS project site is located in northern Saskatchewan, approximately 550 km north-northwest of the city of Prince Albert and 150 km north of the community of La Loche, as illustrated in Figure 4-1. The project site is adjacent to and readily accessible year-round via Saskatchewan provincial Highway 955. Year-round pioneering access roads connected to Highway 955 exist on site for supporting exploration activities, however, either new or upgraded all-weather access roads will be required for external and internal access within project site and to Highway 955 during project construction and operation.

There is no permanent power supply available at or near the project site. A permanent power generation plant on site will be required to fulfill the project power demand during construction and operation. A power generation option trade-off study was undertaken during the FS to determine the optimal method of providing power to the Project. Options included the construction of a high-voltage transmission line from various take-off points, and an on-site powerplant. A subsequent review of diesel power plants and LNG power plants showed that an LNG power plant is the preferred option for power generation as LNG power generation will likely incur less costs and emissions than diesel power generation. The power plant will include a waste heat recovery system to recover waste heat from the power generators for heating process plant and ancillary buildings nearby.

Fresh water will mostly be supplied by drawing water from groundwater wells or Patterson Lake. Other potential sources of water include mine dewatering system, site drainage collection, local water wells, and water reclaimed from TMF and WTP.

There is no reliable fibre optic network or cellular service available at or near the project site. An external communication system will be required during project construction and operation. A communication tower will be erected on site to provide local cellular and two-way radio services within and near the project site.

The PLS project site will comprise the mine site, waste rock/stockpile management facility (WRMF), process plant area, TMF, permanent camp, ancillary buildings and support infrastructure, and a network of access roads within the project site, as presented in Figure 18-1.

Major infrastructure of the mine site will consist of an underground mine with fresh and exhaust air ventilation shafts, a decline for underground access and ore transport from underground to the surface, a freeze plant, dewatering wells, a backfill plant, and an ISP. The WRMF will store non-acid generating (NAG), and potential acid generating (PAG) materials and overburden in separate stockpiles. The WRMF will be lined and equipped with a seepage collection system where required.

The process plant area will include the ore stockpile, main process plant, SX plant, acid plant, effluent treatment facility, surface run-off and monitoring ponds, assay laboratory, administration building, fuel storage and refuel station, vehicle maintenance complex including machine shop and warehouse, power generation plant, power distribution system, LNG storage, and laydown areas. There will be other site support infrastructure beyond the process plant area, including surface explosive magazine, site communication tower, security gatehouses with access controls, freshwater intake, and water distribution system. The process plant pad will be terraced to take advantage of the natural topography and facilitate gravity flows between ponds.

The TMF related systems will consist of infrastructure that is required to safely manage the tailings including tailings transport system, deposition system, a reclaim water system, a sedimentation pond, and a polishing pond.

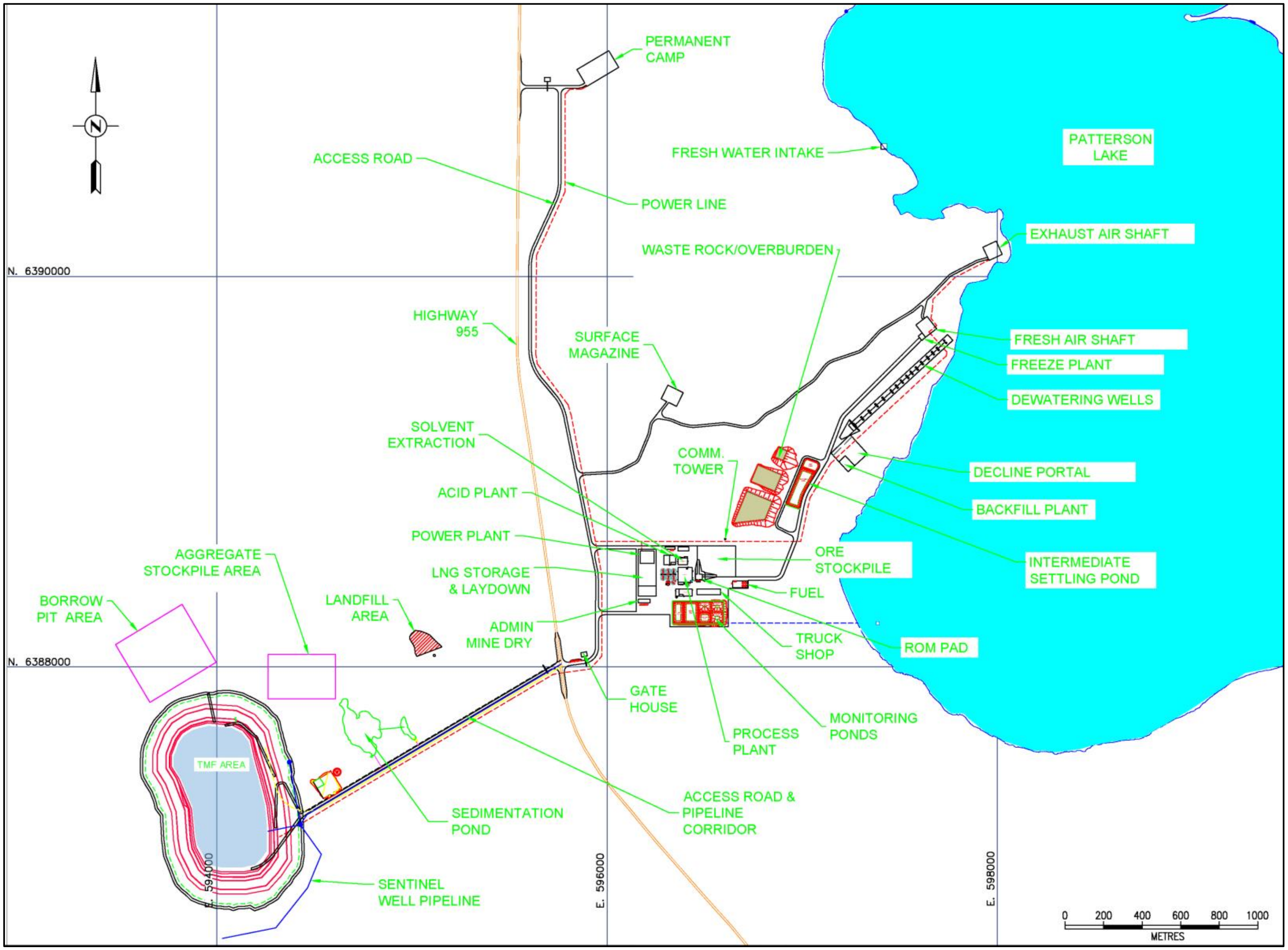


Figure 18-1: Overall Site Layout

18.2 Access Roads

The PLS project site can be accessed by Highway 955 from the community of La Loche. Highway 955 traverses the Property in a north-south direction and comes under the jurisdiction of the Saskatchewan Ministry of Highways and Infrastructure (MHI). Permanent site access roads will facilitate vehicle, material, and supply movements throughout the site and infrastructure facilities. The access roads at the site will be constructed to provide:

External access roads:

- South access from Highway 955 to the process plant area, via a security check point
- South access from Highway 955 to the TMF, via a security gate. This access and #1 above will form an access route between the process plant area and TMF
- North access from Highway 955 to the permanent camp, via a security check point.

Internal access roads:

- Between the process plant area and the underground mine decline, dewatering wells, paste plant, and air ventilation shafts
- Between the process plant area and the permanent camp
- Between the surface explosives magazine and the process plant area/the underground mine decline. Access to surface explosives magazine will be controlled and restricted to the designated service contractors and security only.

The site access roads are classified into primary and service roads. Each category uses an optimized typical section depending on the intended use. The general design criteria for each road type are presented in Table 18-1. A total of 5.7 km of primary access roads and 3.4 km of the service roads are needed for the project site based on the FS site layout. All site access roads and access controls will be monitored and radio-controlled by either site security or operational personnel depending on the location. All external access roads on Highway 955 will be gated to prevent accidental entry by unauthorized vehicles.

Table 18-1: Roadway Design Criteria

Parameters	Primary Road	Service Road
Classification	Double Lane	Single Lane
Design speed	Maximum posted speed limit + 20 km/h	
Maximum speed for mobile equipment	30 km/h	20 km/h
Speed Limit	60 km/h – Process plant area to camp	30 km/h
Driving Lanes	2.0 m x 3.0 m	1.0 m x 3.0 m
Shoulders	1.0 m	1.0 m
Total Surface (Width)	8.0 m	5.0 m
Vertical gradient	8% (Max)	10% (Max)

table continues…

Parameters	Primary Road		Service Road	
Granular base	250 mm		200 mm	
Crushed aggregate	150 mm		100 mm	
Roadway cross slope	2% (Min) – 3% (Max)			
Superelevation	4%			
Minimum corner radius	15 m for semi-tractor trailers and 10 m for light-duty vehicles			
Horizontal alignment criteria	Speed (km/h)	Min curve radius inside crown (m)	Min curve radius outside crown (m)	Min radius with 4% superelevation (m)
	40	65	80	60
	50	115	135	100
	80	400	450	280
Vertical alignment criteria	Speed (km/h)	Minimum SSD (m)	Minimum K Values Vertical Crest Curves	Minimum K Values Vertical Sag Curves
	40	45	4	9
	50	65	7	13
	80	140	26	30

18.3 Mineral Processing Facilities

Please refer to Section 17 for detail descriptions of the mineral processing facilities.

18.4 Site Ancillary Infrastructure

Ancillary buildings and infrastructure for supporting the mine operation and ore processing at the site will consist of a truck shop complex including machine shop and warehouse, permanent accommodation camp, assay laboratory, administration building, fuel storage and refuel station, power generation plant, power distribution system, LNG storage, surface storage magazine, communication tower, security buildings, freshwater intake, water distribution system, and laydown areas.

18.4.1 Truck Shop Complex

The truck shop complex will consist of an 80 m long x 24 m wide pre-engineered sprung building designed to accommodate facilities for repair, maintenance, and rebuilding of underground mining equipment, haul trucks, light vehicles, and mobile equipment. The facility will also provide storage space for spare parts and consumables, offices, lunch room, washrooms and dry for truck shop personnel, first aid, emergency response station, and necessary equipment storage.

The truck shop complex will be located near the process plant. The total usable ground floor area of the building will be approximately 1,900 m^2, including three vehicle service bays, one wash bay, one tire change and lube bay, first aid and emergency response station (where the fire truck and ambulance are stationed), a welding area, a machine shop, and a warehouse. The second floor will provide an additional 400 m^2 of change facilities complete with lockers, showers, and washrooms, a lunchroom equipped with a kitchen and appliances, and office space.

The vehicle service and wash bays will have vehicle access doors on both north and south sides to facilitate vehicle drive-through to eliminate the needs for backing up vehicles or making U-turns, which will enhance the traffic safety for other road users in the area.

The first aid and emergency response station will include a first aid room and separate ambulance and fire truck bays. It will also have storage for emergency response equipment.

The machine shop will be outfitted with machine tools and cutting equipment. Ventilation fans and flash shields will be provided in the welding area for personal protection. Air compressors and receiver tanks inside the truck shop complex will provide compressed air for pneumatic tools.

A modularized lubricant storage enclosure will house tanks for storing lubricants, coolants, and waste oil for the mine and plant mobile equipment fleets. The lubricant storage enclosure will also contain air-operated transfer pumps for supplying lubricants to the truck shop dispensing reels in the service bays. A pipe rack will connect the truck shop to the lubricant storage building. A separate modular exterior storage unit will be provided for waste oil and spent coolants. Waste lubricant recovery systems will pump used oil and coolant to holding tanks located at the lubrication storage facility for recycling or disposal. A bermed spill containment area, sorbents, and spill kits will be provided where the new and used fluids are stored. Fire-proof containers will be provided for storing used oily rags prior to disposal.

The warehouse integrated into the truck shop complex will house materials, service parts, and supplies for mine and plant mobile equipment maintenance. The warehouse will be serviced by electric forklifts. A ready line outside the truck shop complex will provide parking for mine mobile equipment units awaiting service or repairs.

18.4.2 Permanent Accommodation Camp

A permanent accommodation camp for housing the personnel on site will be constructed next to the north gatehouse, as shown in Figure 18-1.

The permanent camp will consist of modular building sections for reception, office space, storage, utilities, kitchen, dining hall, four 50-person dormitories (200 single-occupancy rooms), and recreational space and will be connected with prefabricated fire-rated egress corridors. The site will be fence-secured and have dedicated fresh and fire water tanks. The other support facilities will also be located around the camp building, including parking, an e-house, a waste incinerator, potable water supply and treatment, a modular rotatable biological contactor sewage treatment unit, and a septic pond. The permanent camp layout is shown in Figure 18-3.

The permanent accommodation camp will be constructed as per applicable building and fire code requirements. Appropriate fire detection, suppression, and communication equipment will be installed. The camp will be built for occupancy, with all electrical, communication, lighting, mechanical, sprinklers, plumbing equipment and fixtures, finishes, furniture, and related items, as well as inspected, tested, pre-wired, pre-piped, and pre-assembled as much as practically possible before shipment to site. All dorm rooms will be single occupancy only. Each dorm room will be equipped with a bathroom, plumbing fixtures, light fixtures, windows, bed, desk, chair, closet, baseboard heater, 110 V outlets, internet/WiFi, and cable TV. The permanent accommodation camp will be constructed and ready for operations during the last year of construction (year -1). During the project construction phase, the existing Big Bear accommodation camp, located 20 km north of the PLS project site, can be utilized. Air services for air transportation to an off-site location will be provided by a flight service contractor. Personnel will be transported on ground from the off-site airstrip to site.

18.4.3 Administration Building and Mine Dry

The administration and mine dry building will be a 28 m x 12 m two-storey modular building with a total floor area of 672 m^2. The ground floor will house 125 lockers for men and 30 lockers for women, along with restrooms, change space, janitorial, electrical, and mechanical rooms. The second floor will consist of a lunchroom, offices, workstations, and conference rooms for mine personnel. The lunchroom will be equipped with fridge, stove, microwave, coffee maker, dishwasher, and cupboards. The office spaces shall be equipped with furniture such as desks, chairs, computers, and telephones.

18.4.4 Assay and Metallurgical Laboratory

Refer to Section 17.6

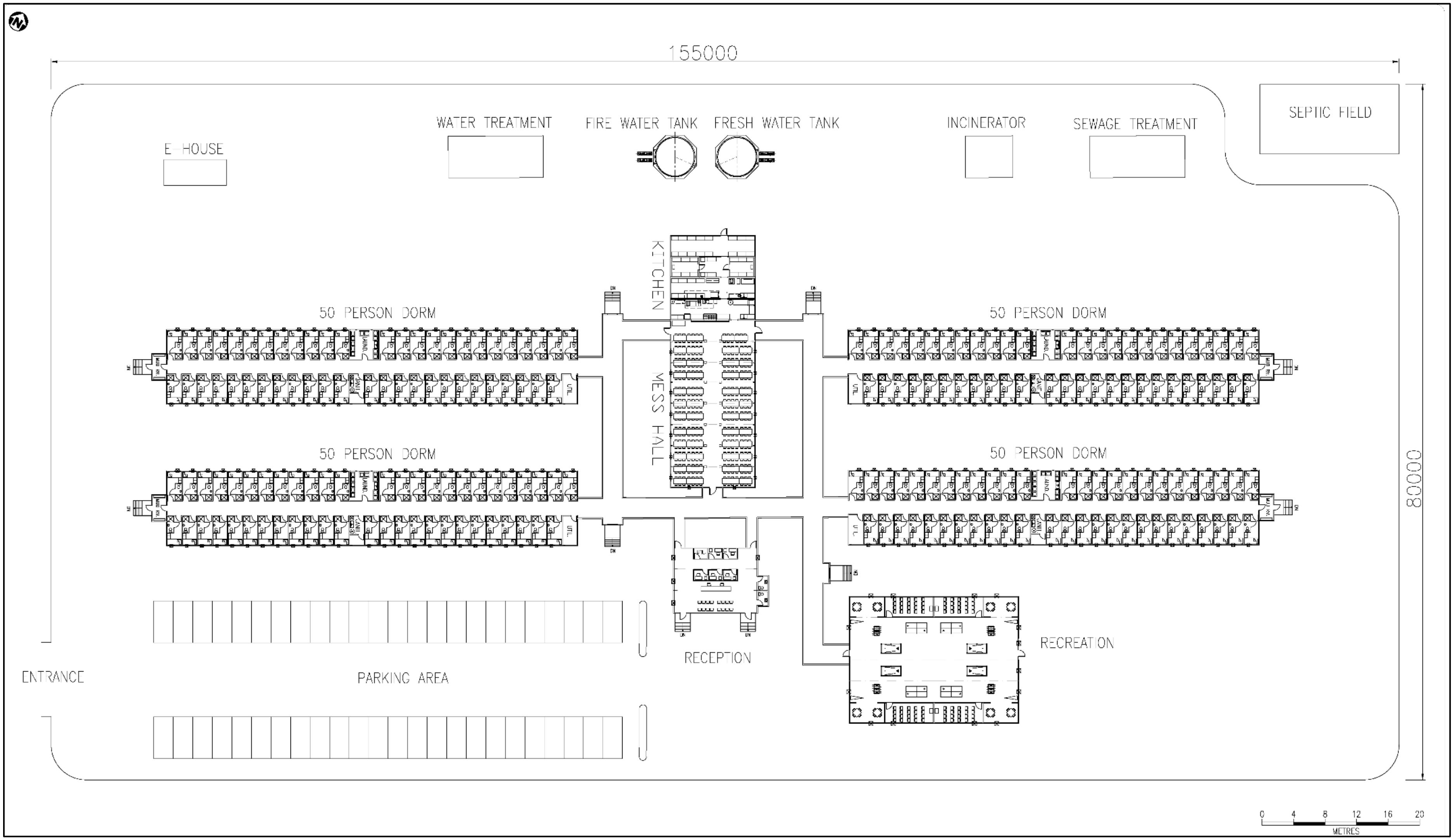


Figure 18-2: 200-Room Permanent Accommodation Camp Layout

18.4.5 Warehouse

The site will also house a warehouse building to store supplies and equipment, such as pumps, motors, steel balls, etc., required for plant operation and maintenance. It will be a 92 m x 52 m insulated, heated, pre-engineered building with a 5-t overhead crane and truck receiving platform. The facility will be equipped with interior and exterior lighting and an electric forklift for offloading and stacking pallets.

18.4.6 Cold Storage Building

A cold storage building is required for the short- and long-term storage of consumables requiring protection from environmental elements such as rain and snow. The building will be a 21 m x 55 m insulated, unheated, single-storey sprung structure. The building will be supplied with light vehicle truck access doors at each end and accompanying main access doors adjacent to the vehicle doors. The building will be provided with interior and exterior lighting.

18.4.7 Gatehouse

There will be two site access control gatehouses. Each gatehouse will be a 6.1 m x 3.6 m single storey modular building with a waiting area for visitors, a reception counter, and a washroom. The south gatehouse near the process plant area control of incoming equipment and supply deliveries. The north gatehouse near the permanent accommodation camp will control access of the incoming traffic to the camp. All personnel coming to the site will report directly to this gatehouse.

18.4.8 LNG Storage and Laydown Area

The LNG storage and laydown will be a 100 m x 92 m graded surface, which will be used to receive, store, and supply LNG to the power plant. The area consists of an LNG offloading pump station to transfer LNG from shipping tankers to insulated storage tanks, a vapourizer for turning LNG to gaseous form, a waste heat vapourizer, water/glycol pumps for building heating, a control room, and pipeline infrastructure for transporting LNG within site. The storage area will house six 100 m^3 LNG storage tanks, providing about five days of power generation capacity.

The laydown area will also be constructed with drainage channels and a sump to collect the rain/snow melt. Periodically, it will be pumped to the site water treatment facility for effluent treatment.

18.4.9 Power Supply and Distribution

A permanent power transmission facility with sufficient capacity exists and is located approximately 230 km from PLS project site. During the FS, Tetra Tech completed a power supply trade-off study and compared several power supply methods. At the time the trade-off study was completed, LNG power generation on site was deemed the most preferred power supply method based on economic and technical merits.

An onsite LNG-fired power plant will be procured, installed, and operated to supply the required power. The planned power plant is based on eight LNG-fired generator sets, seven operating and one stand-by (n + 1 criteria), each rated at 2.5 MW continuous generating at 13.8 kV, with a total installed generating capacity of 20 MW. The total continuous running power requirement for the site is estimated to be around 16.9 MW. Critical spare parts will be stored on site and engines will be rebuilt periodically to minimize unplanned downtime. Uninterrupted Power Supplies (UPS) and battery banks on site will provide the power supply to critical electronics and safety equipment in case of interruptions in power generation or distribution.

Each generator assembly will be housed in a prefabricated, insulated, and sound-attenuated enclosure with skid-mounted heat recovery system. These enclosures will also house the ancillary equipment and associated infrastructure, including switchgear, MCCs, low voltage distribution and grounding systems, and control equipment and panels. All electrical modules will be procured as a walk-in package to simplify the installation process.

The modularized power generator sets will be factory fabricated and commissioned by the manufacturer to enhance reliability and reduce labour costs. All enclosures will have a fire detection system and alarm panels with aerosol-based fire suppression systems. The generators will also be fitted with a heat recovery system to recover the waste thermal energy produced during combustion for plant and infrastructure heating.

Power distribution to the mine, TMF, permanent camp and other areas within site will be provided by 13.8 kV overhead distribution lines. Depending on the load requirements, it may be supplied as is at 13.8 kV or stepped-down to 4,160/600/120 V as required. The large, fixed-speed motors (>200 kW) and variable-speed motors (>400 hp) are serviced with a 4,160 V system. In general, motors and other loads below 200 kW will be fed from one of several 600 V systems. General power and lighting will be supplied with a 600 V system. Resistance grounding will be used at all distribution voltage levels. All buildings will be equipped with proper grounding and lightning protection.

18.4.10 Acid Plant

Based on sulphur-burning technology, the sulphuric acid requirement for leaching, CCD, SX, and effluent treatment will be obtained from an on-site acid generation plant. The plant can produce a maximum of 250 t/d of monohydrate sulphuric acid (98.5% w/w) using the double absorption-conversion process and consists of four catalyst beds. In addition to acid generation, the acid plant will produce superheated steam that will be utilized in the leaching and other circuits within the process plant.

The plant components will be modularized and shipped for final assembly and installation at the site with technical support from the manufacturer. The acid plant will be housed in a 60 m x 48 m insulated, single-storey, pre-engineered building containing three areas: sulphur handling and storage area; sulphur melting, filtration and storage area; and acid generation area along with laydown, mechanical, electrical rooms, and storage tanks for sulphuric acid. The building will be supplied with truck access for sulphur receiving and provided with interior and exterior lighting.

18.4.11 Surface Explosives Magazine Facility

A 60 m by 60 m pad surrounded by earth berms will be prepared during site preparation for use by the blasting service supplier for storing the explosives magazine. The pad is 250 m from the nearest road and 350 m from the nearest building. Access to the pad will be controlled and restricted to the blasting service supplier and production and development blast crews.

18.4.12 Fuel Storage and Dispensing

Diesel will be used for general site equipment, surface and underground mobile equipment, backup generators, and other ancillary services.

Modular diesel fuel tanks (known as “ISO tanks”) will be used for fuel storage and transported to the site. The ISO tanks are double-walled. Each tank is protected by a boxy roll cage consisting of welded steel beams that protect the tank from external impact. The flow controls of the tank are valved. The valves will stay closed to prevent fuel leakage or escapement of fuel vapour. The tanks will be routinely inspected and tested for preventive maintenance.

Each tank will be equipped with leak detection instruments that measure the vacuum between the tank walls. Tanks with a lower-than-normal vacuum reading will be removed from operation and sent to the maintenance shop for diagnosis and repair.

The ISO tanks will be placed in a designated fuel farm area adjacent to the truck shop complex and they will provide approximately one week of fuel storage capacity. A modular fuel dispensing station will provide a means for fuelling mobile equipment. The ISO tanks will be transported to and removed from site by the fuel supply vendor for refuelling. The ISO tank storage area will be protected by spill containment berms and fuel resistant liners. The designed containment volume of the spill containment area will be at least 110% of the capacity of the largest fuel storage tank in the containment area. Any fugitive fuel contained within the containment area will be removed as soon as practically possible. The containment areas will be instrumented with monitoring devices, which will alert the control centre of any detected leakage or spillage. Spill response vehicles and equipment will be stationed at the truck shop complex, on standby 24/7, and dispatched immediately when required.

18.4.13 Communications

A voice and data communications system will be established at the mine site via a microwave radio link and will consist of a microwave antenna mounted on a tower near the process plant. All telecommunication systems will be supplied as a design-build package.

A communications network will be established among occupied buildings utilizing fibre-optic technology and wireless communication for voice, Internet, and intranet traffic. The design will include:

- Voice over Internet Protocol technology using wide area network (WAN) links for voice communications and video conferencing systems.
- VHF radio system will be installed with provision for handheld units, mobile units, and base stations.
- A telephone PBX system and cellular phone service will be provided for telephone communications.
- Satellite television for the camps is also planned.
- A base and client station will be provided for wireless connection to the network system. The system will include a smart card access system to enable secure login to the network for desktop and laptop users.
- The local area network (LAN) system will utilize switches to connect to users' computers, and the WAN system will use routers with multi-protocol label switching capabilities to support voice and high bandwidth capabilities.

A backup satellite system rated to handle the full communications bandwidth will also be installed. Backup power to communication and critical control systems will facilitate the orderly shutdown and start up of equipment, control systems, and backup computers.

A pre-manufactured trailer consisting of the main communication contractor and all sub-systems will act as a main telecommunication central office, which will serve the construction phase and later expand for the operation phase of the project.

18.4.14 Heating and Ventilation

Heat recovery from the power plant will provide heating for buildings and facilities near the power plant. Waste heat from the power plant will be transferred by transfer pumps through a primary glycol circulating system loop throughout the site. A secondary circulating loop will be the distribution system for individual complexes and facilities. A backup heat source using a propane-fueled boiler will be provided when required.

The facilities, such as office space, board rooms, dining space, electrical, and communication and control rooms, will be air-conditioned. Localized controls will provide climatic control for the unit heaters.

Continuous ventilation will be provided for all personnel-occupied and selected unoccupied spaces. Ventilation rates will vary depending on the level of occupancy and the intended use of the area. Ventilation systems will include make-up air units for a continuous supply of tempered air, general exhaust fans for contaminant removal, and, where appropriate, localized exhaust fans to remove contaminants directly. For buildings near the power plant, glycol supply to the make-up air units will be the primary heat source. Only non-toxic and biodegradable glycol will be used.

18.4.15 Fresh and Fire Water

Freshwater requirements for the site will be supplied from the ground water wells. Freshwater will be used primarily for potable water supply and firewater for emergencies. Water from the effluent treatment plant will be utilized to the maximum extent possible for process-related activities to minimize the freshwater requirement. All the freshwater pipelines outside heated buildings will be buried below the freezing level. Potable water for the site will be pumped to the water treatment units (chlorination and ultra-violet disinfection), stored in a potable water tank, and distributed to the various facilities on the site.

Potable water for the accommodation camp will be supplied by groundwater wells located near the camp site with dedicated water treatment units and storage tanks to avoid long pipelines. Water supply from Patterson Lake will be used as a backup source of water

Fresh/firewater for the plant and other facilities on the site will be stored in a fresh/fire water tank located at the process plant with a fire water reserve. The reserve water for firefighting will meet a two-hour demand at 2,000 US gal/min at 110 psi boost. A firewater loop will be installed throughout the site with a backup diesel pump and jockey pump to supply water to the fire hydrants. Fire extinguishers and automatic sprinkler systems will also be installed throughout the facilities. Emergency showers and eye wash stations will be established at predetermined locations.

The SX plant will be provided with a specially designed fire suppression system. The SX plant will utilize a carbon dioxide (CO_2)-based fire suppression system. Spare CO_2 cylinders will be stored in a connected backup configuration. If a fire is detected in one mixer settler unit, all units will be simultaneously injected with CO_2 as a precaution. The complete fire suppression system will be integrated with the DCS for auto-operation. In an emergency, the fire system takes control of the process for an orderly shutdown.

Dedicated fire mains complete with hydrants will be provided at the accommodation camp.

Fire alarm systems at the site facilities will report to the emergency response/first aid unit located at the truck shop complex, which will be monitored 24 h/d.

18.4.16 Sewage Treatment

Sewage from the accommodation camp and other site facilities will be collected and treated using rotating biological contactors (RBC)-based sewage treatment module. The treatment module will consist of a preliminary sedimentation tank, RBC, a secondary sedimentation tank, and finally UV disinfection. The treated water will be recycled or pumped into the TMF. Residual solids/sludge will be shipped off site twice a year by a specialized licensed contractor for final disposal. Washroom facilities in remote buildings will be serviced by a vacuum truck.

18.4.17 Hazardous Wastes

All the hazardous waste will be segregated and placed into designated containers at the point of generation. All the collected waste will be temporarily stored in a lined laydown area near the site fuel storage facility and shipped offsite to a recycling or disposal facility.

Some of the typically generated hazardous waste handling protocols are listed below:

- Waste oil and organic waste liquids such as antifreeze, solvents, and grease will be shipped to an offsite recycling facility.
- Old tires will be used to construct vehicle protection barriers on the haul roads, where necessary, any excess tires will be shipped to an offsite recycling facility.
- End-of-life electronics, light bulbs, and batteries will be collected and shipped to an off-site recycling facility.
- Soil contaminated with hydrocarbon will be collected and treated in a bioremediation land farm located near the on-site landfill. In the land farm, the contaminated soil will be spread 30 cm to 45 cm thick layer and tilted and treated with fertilizer to promote bacterial growth and activity. Soils will be monitored for pH, water content, nitrogen compound concentrations, microbial population density, and others to ensure optimal conditions for biodegradation. The land farm will be lined, and any run-off water will be collected and treated before being sent to the tailings facility. Bioremediated soil will be stockpiled in the waste rock facility or used in progressive reclamation projects.

18.4.18 Non-Hazardous Wastes

Non-hazardous waste will be separated into putrescible and non-putrescible waste. The putrescible waste, such as food, will be segregated and incinerated at an onsite facility. Non-putrescible, recyclable waste will be collected and stored in an onsite landfill or shipped to an offsite recycling facility. The landfill will be periodically covered under a layer of NAG waste rock to reduce windborne pollution from the waste. The run-off water from the landfill will be collected and treated at the onsite water treatment facility.

18.4.19 Medical/First Aid

First aid posts will be provided at the accommodation camp and the truck shop complex. A full-time nurse will be in attendance at the first aid station at the camp. A fully equipped ambulance for advanced life support will be located at the truck shop complex. Emergency Med-Evac air ambulance services will be contracted out to registered services in the nearby communities.

18.4.20 Security

Access to the project site will be monitored and controlled by the security on site. There will be two security gatehouses, at each of the two site access roads from Highway 955. Each gate house will be stationed by the site security 24/7. The site access to the TMF will be gated and only accessible to site and contractor crews. The TMF access gate is also monitored and remotely controlled by the site security 24/7.

18.5 Waste Rock Management Facility

In accordance with REGDOC-2.11.1, this section explains the management of waste rock material generated during the PLS project life to ensure the protection of the environment and the health and safety of the people. The management of tailings and TMF is discussed in Section 18.6. The low-grade ore stockpile is discussed in Section 17.3.1.

18.5.1 Mine Overburden Stockpile

The mine plan has shown that during underground mine development activities such as box cut and decline construction, approximately 55,000 m^3 of excavated material would be generated. It is expected that most of the material in the overburden would be stored and managed at the WRMF and later used as mine backfill material.

The mine backfill material stockpile will be HDPE lined to collect the run-off water into a settling pond, which will be treated at an on-site WTP before being released into the environment.

18.5.2 Waste Rock Stockpile

The project is expected to generate approximately 500,000 m^3 of waste rock material during the LOM. Out of total waste rock material, non-PAG accounts for 74% (~370,000 m^3) and PAG is estimated to be the remaining 26% (~130,000 m^3) of the material. All of this material will be trucked back underground as backfill after processing through the backfill plant.

The general arrangement of waste rock and mine backfill material stockpiles are shown in Figure 18-1 and Figure 18-3.

All the run-off water from overburden and waste stockpile will be directed to the settling pond, which is designed to withstand a 1:100 years 24-hour rainfall event.

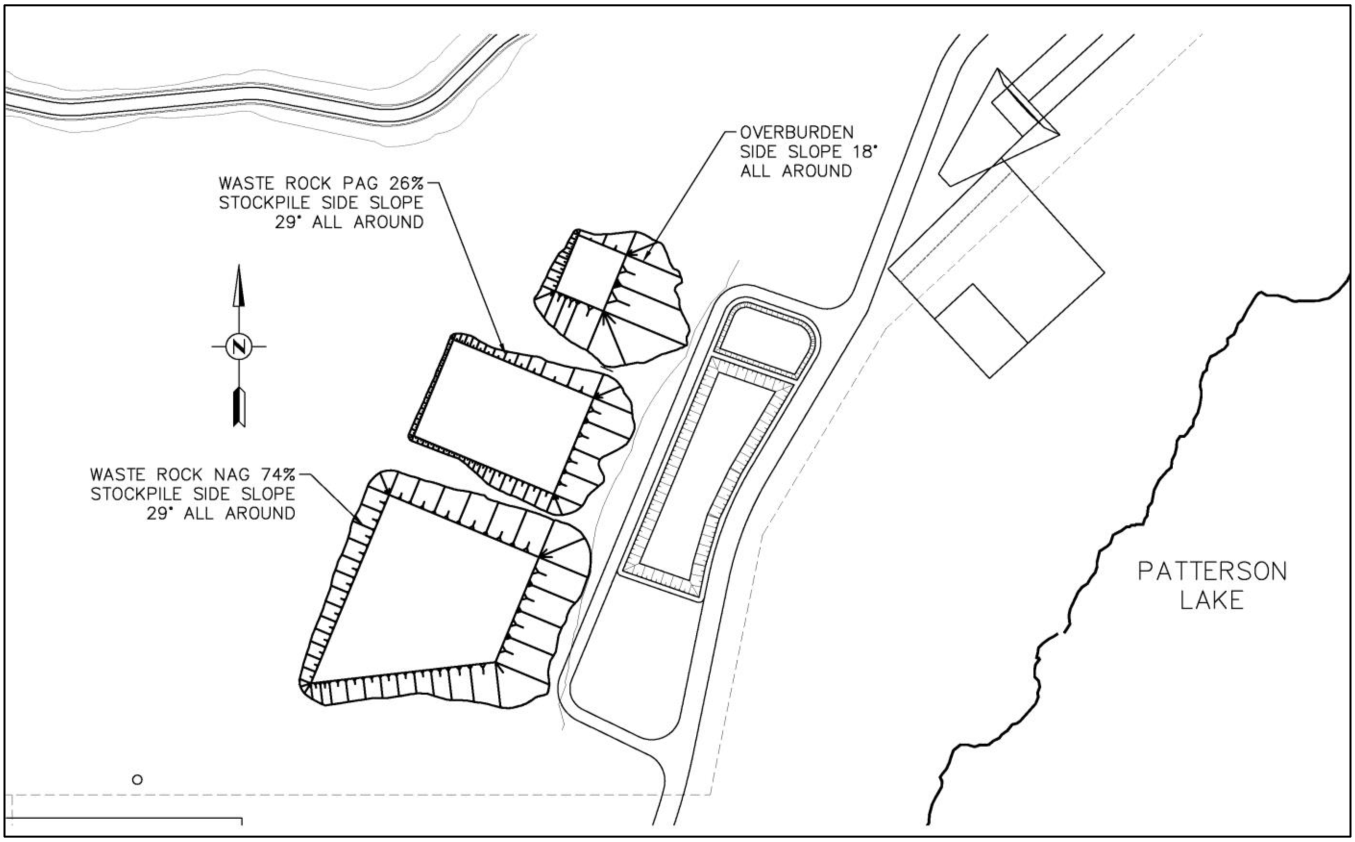


Figure 18-3: General Arrangement of Waste Rock and Mine Backfill Material Storage Stockpiles

18.6 Tailings Management Facility

This section outlines the development plan for the TMF in sufficient detail to support the FS of the project.

18.6.1.1 Background

The site geology and groundwater regime are core considerations in TMF development since tailings containment and the prevention of potential release of contaminants to groundwater are primary design considerations. A TMF location adjacent to Patterson Lake was initially proposed in the PEA. That location was subsequently rejected due to proximity to Patterson Lake and a siting study undertaken during the PFS to identify suitable alternative locations.

The PFS siting study proposed two alternative TMF locations (primary and secondary) that were screened during a 2019 field investigation (Clifton 2019). A preferred location was selected approximately 5 km southwest of the mine with a secondary location approximately 4.3 km directly west of the mine. While this investigation concluded that the preferred TMF location was more favorable, nothing observed in the investigation precluded use of the secondary location.

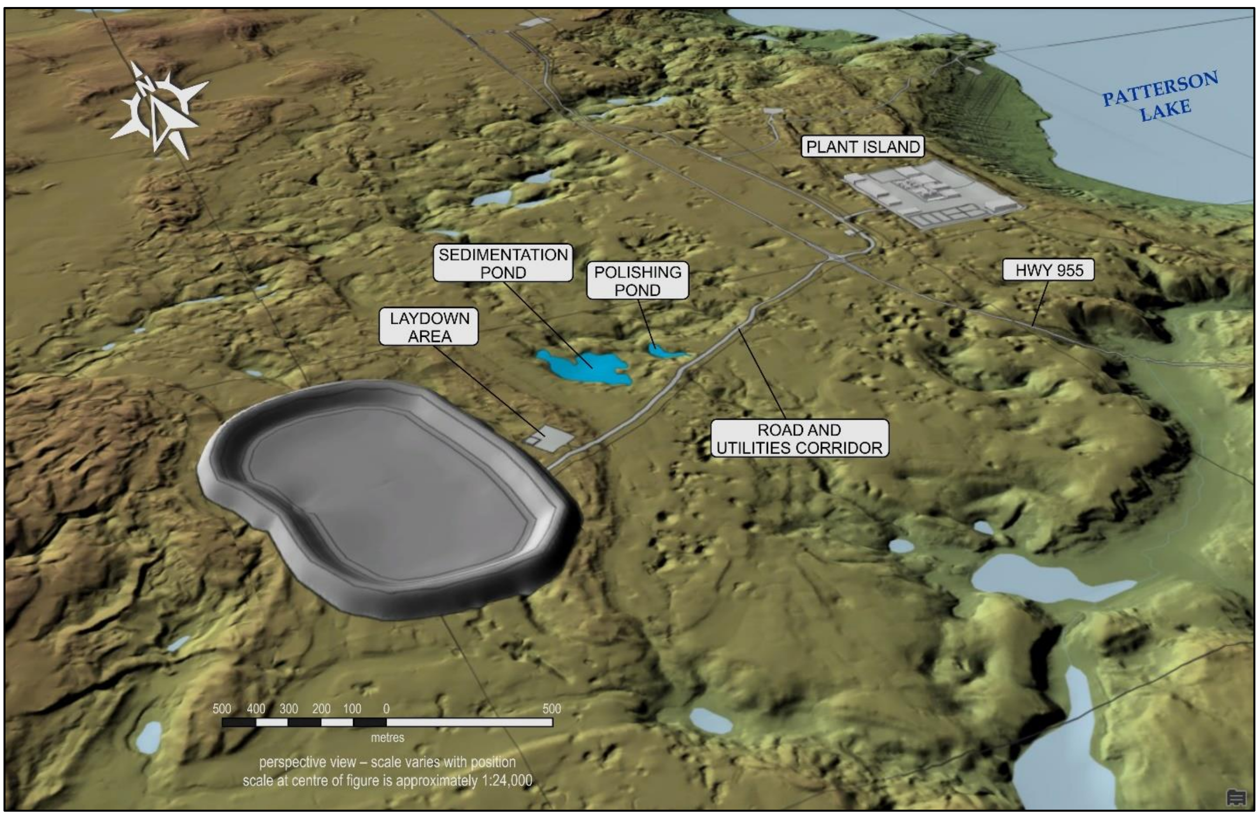
One of the decisive factors in selecting the primary area was the ability to site the TMF fully in the Patterson Lake watershed, while runoff from the secondary site could potentially drain into Alberta.

In the spring of 2021, the layout of the TMF was optimized in the preferred location to make best use of local topography, reduce earthwork quantities, and provide better integration with the local topography following closure.

18.6.1.2 Site Description

The landscape in the vicinity of the proposed TMF was created by multiple glaciations which deposited thick glacial (Quaternary) sediments overlying the Precambrian basement. The last glacial retreat left a series of recessional moraine ridges as well as a series of meltwater channels positioned roughly perpendicular to the moraine deposits. The TMF site is bounded by a high morainal ridge on the west and smaller ridge on the east as illustrated in Figure 18-4. Pitted glaciofluvial (outwash) deposits, often coarse-grained, form the surface landform at the TMF and in the channel to the north. The surrounding terrain is hummocky with numerous pothole lakes and heavily eroded drainage channels.

Field investigations at the current TMF location, (Clifton 2019; 2021) confirmed thick (up 25 m) stratified sand with minor silt facies overlying till, with the basement occurring at depth as illustrated in Figure 18-5, a typical stratigraphic column for the TMF site. The upper 25 m of sediments are poorly-graded, medium to fine grained sand with traces of gravel, silt, and clay. These materials, ubiquitous within the proposed TMF footprint, are permeable sediments with a mean hydraulic conductivity estimated at 3 x 10^{-5} m/s. The water table, controlled by local infiltration and the regional flow system, ranges from approximately 7 to 15 mbgs in the TMF area.



Source: LiDAR data provided by FCU.

Figure 18-4: Hillshade Projection of LiDAR Data Showing Site Topography

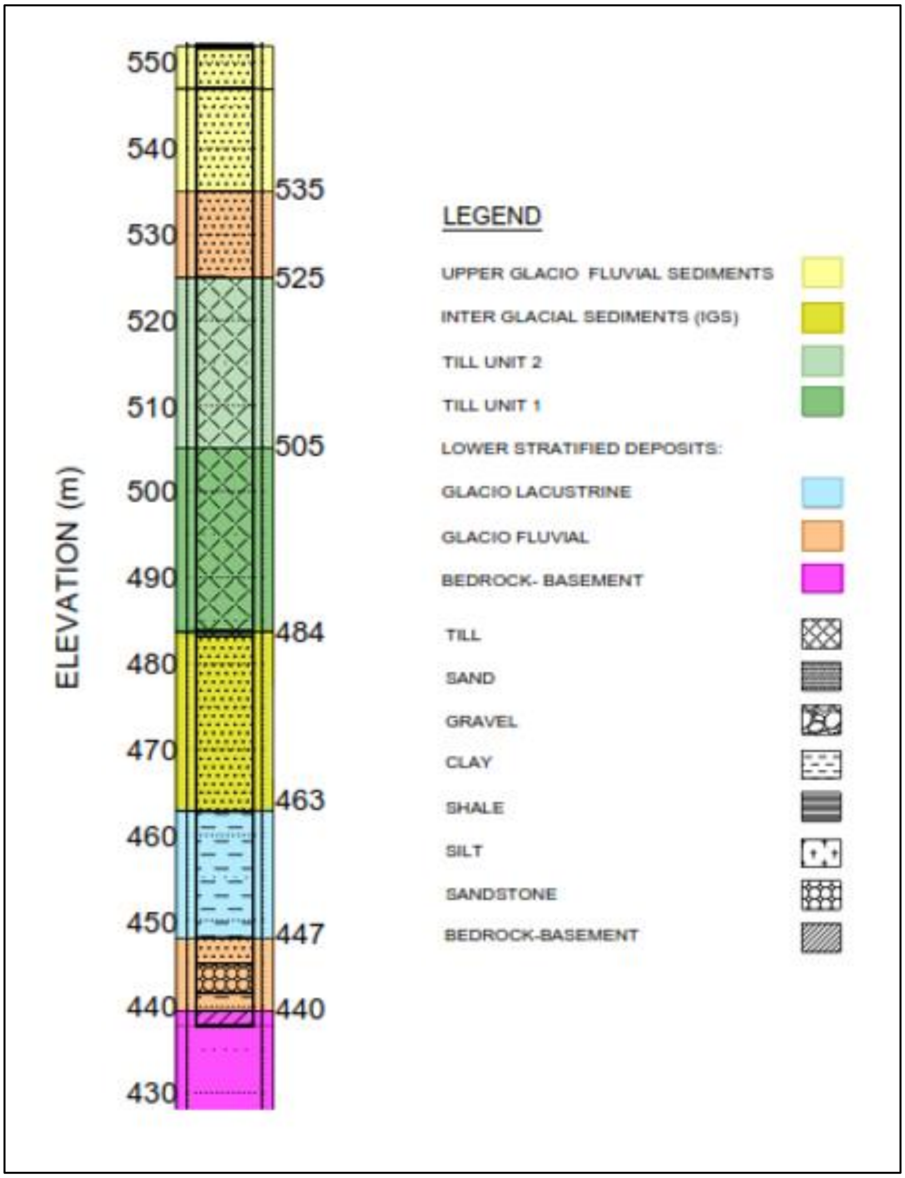


Figure 18-5: Typical Stratigraphic Column at the TMF site (PLS-TMF-06) – Ground Surface Approximately 550 masl

Drainage at the site is controlled by the local topography. A regional drainage divide exists approximately 4 km west of Patterson Lake and is identified as a watershed boundary (NHN Watershed 07CD, Gov. Canada 2016) where water to the east of the boundary drains toward Patterson Lake and water to the west drains, eventually, into Alberta.

18.6.1.3 TMF Philosophy

The TMF must provide secure storage of the uranium process plant tailings and process fluids during operations in a fashion that supports secure disposal in the long-term (approximately 10,000 years). It must also mitigate acid rock drainage (ARD), radon release, and dusting. The chosen TMF design technology is based on constructing a purpose-built pit to support sub-aqueous injection of tailings during operations, then decommissioning the TMF in a fashion that incorporates it into the surrounding terrain and sustains saturated tailings in the long-term.

The TMF incorporates a modified version of the pervious surround technology successfully used at other uranium tailings management sites in the Athabasca Basin of northern Saskatchewan. Constructing the TMF in permeable drift sediments will require special design considerations to maintain containment during operations as local naturally occurring low conductivity materials, such as clay, are in short supply.

An engineered double barrier system is proposed to prevent release of contaminated fluids to the country soils.

The TMF is designed to provide containment during operations and decommissioning phases through use of perimeter and basal drains (pervious surround) that will promote consolidation with the entire facility contained with an engineered double barrier. Pore water that is expelled through the drains will be collected in an underdrain where it will be returned to the process plant for treatment. When consolidated, the tailings will form a low-permeability mass inside the engineered double barrier, where preferential groundwater flow will be through the country soils around the tailings. Diffusion is the principal long-term solute transport mechanism for the tailings mass.

Upon mine decommissioning the tailings water cover will be drained, the tailings mass loaded with soil to promote further consolidation, and then capped with an engineered soil cover. The engineered cover system will provide a barrier to the inflow of fresh water from precipitation and runoff while limiting ingress of oxygen to mitigate both acid drainage and leachate generation.

18.6.2 Basis of Design

18.6.2.1 Guiding Principles

Canadian regulators require that proponents provide sufficient information to demonstrate that the uranium process plant TMF can be constructed, operated, and decommissioned effectively. The long-term objective of the TMF is to leave the site in a state that is physically safe and provides secure, long-term storage of the tailings with acceptable levels of predicted future impacts. All designs must demonstrate that they are technically feasible, environmentally acceptable, and can be acceptably decommissioned. Regulators require application of best available technologies for TMF design in a fashion that produces a high level of environmental protection that is as low as reasonably achievable while also considering social and economic factors.

The guiding principles used for the design of the TMF included the CDA guidelines (CDA 2007) and the 2014 technical bulletin on the application of the CDA guidelines to mining dams, with the understanding that certain items will need input from other sources.

The design work for the TMF has considered construction, operation, closure, and landform stages. The design has adopted the philosophy of "designing and operating for closure" (MAC 2021). This philosophy considers closure of the TMF at any stage of design or operation to limit the additional engineering and effort required to return the TMF site to a natural landform, reducing the potential for long-term adverse environmental impacts and attendant corporate liability.

18.6.2.2 TMF Operations

The design scenario for the TMF is subaqueous deposition of thickened slurry tailings into a lined pervious surround pit. Tailings will be transported to the TMF in a pipeline as a thickened slurry. Spill prevention and control measures for the slurry pipeline will be incorporated to provide protection against leaks and spills along the tailings pipeline corridor.

The tailings will be sub-aqueously deposited using a relocatable barge in a manner that facilitates even distribution of tailings and prevents particle segregation, to produce a uniform, low permeability consolidated tailings mass. A water cover, consisting of clarified tailings solution will be maintained to support barge deposition of tailings while preventing freezing of the tailings and providing a barrier to low energy radiation, dust, and radon release. Excess water will be returned to the process plant for treatment and release to the environment.

The TMF can also act as an emergency storage for site water if there is a large storm event that overwhelms the site storage (e.g., a PMP event). Excess water would be returned to the process plant for treatment over time and released to the environment.

18.6.2.3 Tailings Production

The FS estimated a required design capacity to support a 10-year mine life with 1,000 t of ore processed daily. After processing and the generation of precipitates, a total of 1,120 t of tailings solids will be produced daily. Provision has also been made for additional capacity by assuming that a 25% increase in daily tailings production will occur over the scheduled 10-year mine life.

The tailings slurry as deposited in the pit will have a bulk density of approximately 40% solids by mass and will rapidly settle to a bulk density of approximately 50%. Sizing of the TMF was based on this rate of settlement plus ongoing consolidation of the tailings, inclusion of a 3-m thick water cover, and provision for 2 m of freeboard in the final year of operation. The total storage available in the TMF is approximately 8,200,000 m^3.

18.6.2.4 Geotechnical Design and Water Management

Geotechnical design of the TMF will be in accordance with the CDA guidelines and the technical bulletin on the application of the guidelines to mining dams (CDA 2014). The geotechnical design of the TMF has resulted in the general arrangement (Figure 18-4) and typical details illustrated in Figure 18-7 and Figure 18-8.

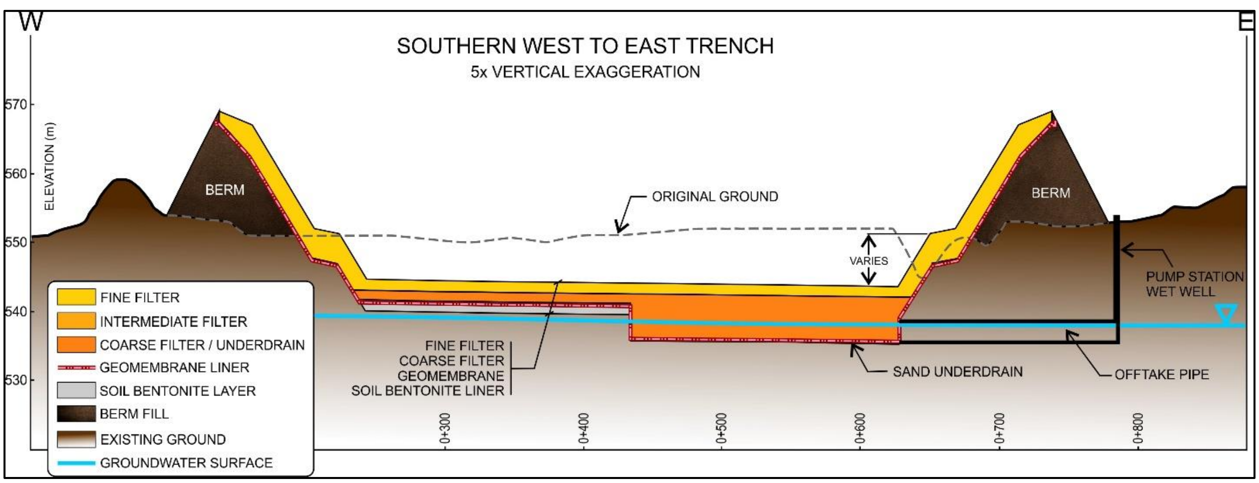


Figure 18-6: TMF Pit East to West Cross Section

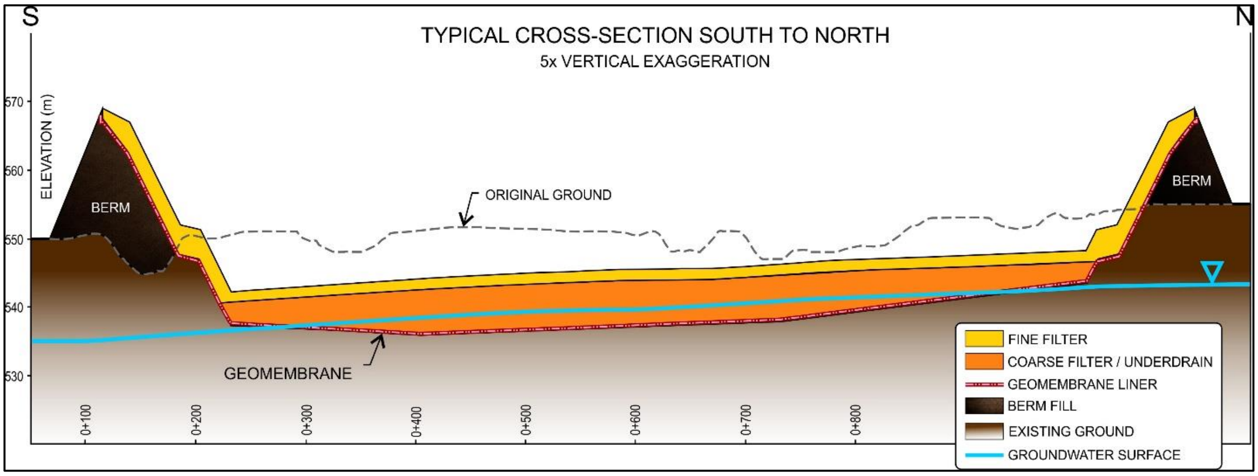


Figure 18-7: TMF Pit North to South Cross Section

The engineered double barrier system is essential for successful operation of the TMF. The engineered double barrier on the TMF floor will consist of a thick SBL overlain by a geomembrane. The SBL has been designed with a low hydraulic conductivity to provide a second barrier to seepage loss from the TMF. In addition, the ion exchange capacity of the soil-bentonite barrier will further attenuate releases of metals and radionuclides that may pass through the geomembrane liner.

The barrier system for the berm slopes will consist of a double geomembrane liner without a SBL underlay. The second membrane will maintain secure containment on the slopes where the applied head will be small due to the overlying free-draining filter that will conduct the tailings solution to the underdrain.

An operational water budget was completed to estimate the process flows in various segments of the TMF (Figure 18-8). The main water input to the TMF is process water incorporated in the tailings slurry. In the first few years of TMF operation, a large volume of water will be required to develop and maintain the 3 m water cover; consequently, makeup water will be required from the sentinel wells and no water will be returned to the process plant. The volume of water cover volume is anticipated to stabilize in the third year of operation when equilibrium will be reached with water recirculation to the process plant.

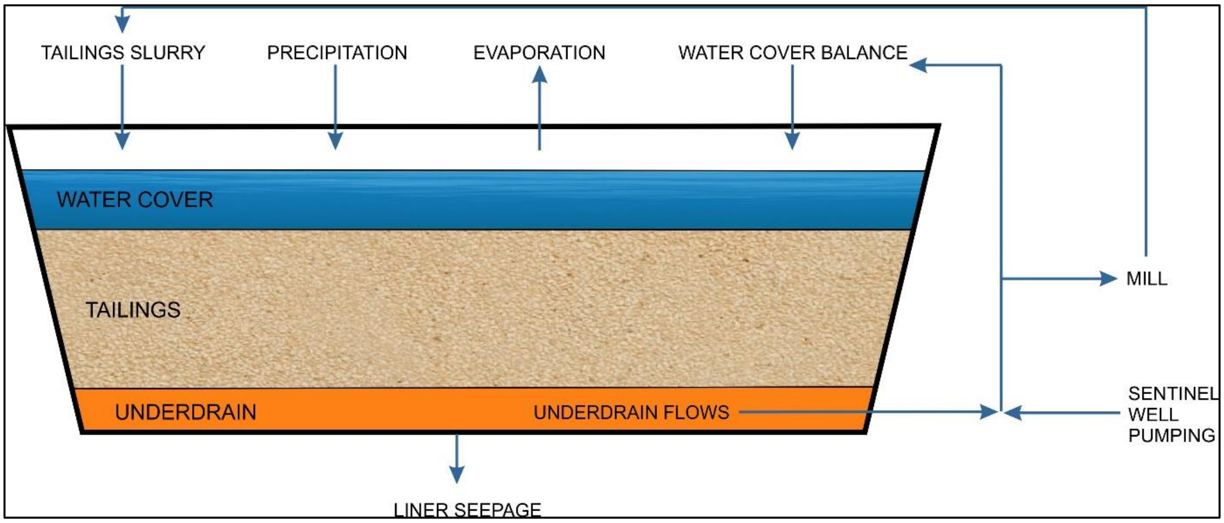


Figure 18-8: Inputs and Outputs for TMF Water Balance

18.6.3 Environmental Impacts

18.6.3.1 Conceptual Site Model

Information collected in the field investigations was used to develop a conceptual site model (CSM) to assess potential impacts to the groundwater system. A CSM (ASTM E1689-95) is a description of the geologic, groundwater, and solute characteristics of a site. The CSM is a representation of an environmental system that facilitates evaluation of the transport of solutes from a source via a pathway to a receptor.

The primary source for solutes is the tailings solution in the TMF. A secondary, but relatively minor, source of potential solutes is the support infrastructure for the TMF (e.g., roads and ramps) that are in contact with mining activities. Only primary sources were considered in this assessment.

Solute release from the TMF is the most significant environmental concern. Other sources will be mitigated in the following manner:

- Surface water interaction with TMF infrastructure – Fresh runoff that is generated upstream of the TMF will be routed around the facility; precipitation intercepted by TMF infrastructure (roads, laydowns, gatehouse, etc.) would be stored for short periods of time, tested, and released when water quality allows;
- Release of radon – The tailings water cover will prevent radon release from the tailings; and
- Dust from tailings – Saturated tailings and a water cover will prevent tailings dusting.

The primary pathway is groundwater for solute transport. Groundwater flows from northwest to southeast (Figure 18-9), and solute releases to groundwater and surface water runoff ultimately report to Patterson Lake but potential local receptors such as local pothole lakes and ravines must also be considered.

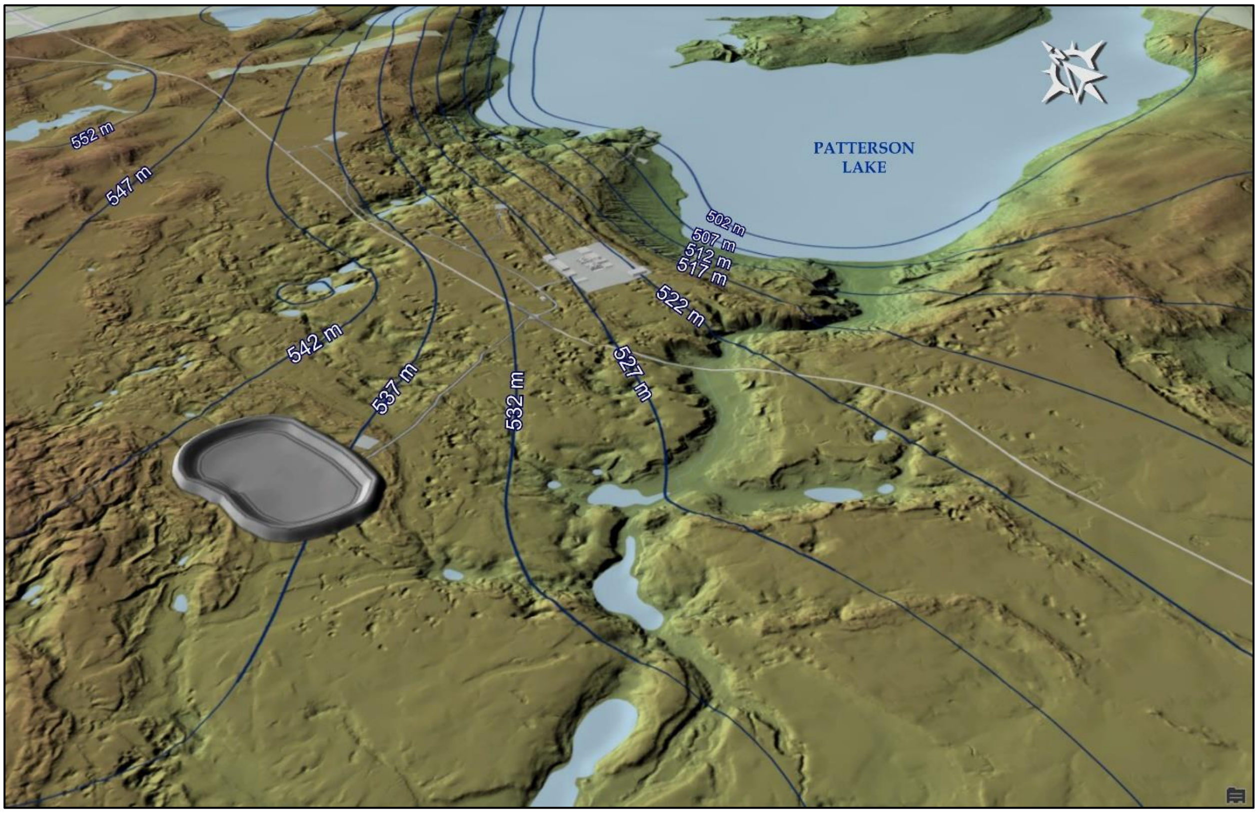


Figure 18-9: Estimated Piezometric Surface (masl) from Calibrated Groundwater Model

18.6.3.2 Aqueous Solutes

Analysis of the tailings pore fluid was completed by SGS (2020), the analytes measured are compiled in Table 18-2.

Table 18-2: Tailings Pore Water Analytes

TSS, TDS, pH, Alkalinity, Conductivity, F, Cl, SO_4, and NH3+NH4	Lead-210, Radium-226, Radium-228, Thorium-228, Thorium-230, Thorium-232, Potassium-40, and Uranium-238	Hg, Ag, Al, As, Ca, Cu, Fe, K, Mg, Mn, Mo, Na, Ni, Pb, S, Se, Si, U, and Zn

Species in the list of analytes with dissolved concentrations exceeding Saskatchewan Environmental Quality Guidelines (SEQG) and/or MDMER are Ra-226, As, Cd, Cu, Se, and U. Unique exceedances of total concentrations of some analytes that do not show a dissolved exceedance include Ag, Al, Fe, Ni, and Pb. The total exceedances were typically found in the short-term tests, between 0 and 28 days. Further analyses only considered dissolved species that exceeded SEQG or MDMER guidelines.

18.6.3.3 Release Mechanisms and Mitigation Strategies

Three potential release mechanisms have been identified for the TMF, with two identified as critical and one as typical. The critical release mechanisms included:

- Global stability failure of the TMF pit berms; and

- Failure of the engineered barrier system.

A typical release mechanism is slow migration of solutes through the engineered barrier (liner) system, as no liner system can be considered "impervious" for the design life of the TMF.

The two critical release mechanisms could potentially result in significant adverse environmental impacts. Global instability of the TMF confining berms could release tailings to the southeast to a ravine and, potentially, to Patterson Lake. Failure of the engineered barrier system may release retained pore water to the groundwater system, eventually discharging to Patterson Lake. TMF design processes have focused on mitigating the risks of these critical failure modes.

A potential failure mode for the berms is a global stability failure under either static or seismic conditions. Stability of both excavated and embankment slopes were designed to meet the requirements of the CDA guidelines under the most adverse design condition, including the designated seismic acceleration for the Patterson Lake area. Similarly, static liquefaction has been assessed using site-specific data and was found to satisfy design criteria. The design geometry and technical specifications for berm construction will ensure that the constructed works will be stable for all identified construction, operating, and post-decommissioning conditions.

The potential failure mechanism for the tailings management system is piping of tailings or filters and plugging of the drainage system. A graded filter/liner system has been designed to mitigate against piping and internal erosion of the tailings and filters. A water management system coupled with large drainage layers has been implemented to reduce the potential for plugging, by maintaining saturated drains and by providing excess drain capacity.

The principal release mechanism of concern is migration of solutes through the engineered double barrier system. The engineered double barrier system has been designed to minimize solute release from the TMF. The primary release mechanism during operations would be dominated by seepage through liner defects. Once the water cover has been drained and the tailing have been consolidated, the release mechanism will be dominated by diffusion, a much slower process.

Any release through the liner system will eventually report to the groundwater system and be transported down-gradient in the regional flow system. Numerical modeling has identified probable flow paths for any such releases; a monitoring system and locations of potential solute recovery wells to mitigate such releases have been informed by those models.

18.6.3.4 Potential Groundwater Receptors and Estimated Environmental Impacts

The elevation of the water table, and ground water flow directions in the area potentially impacted by the TMF was illustrated in Figure 18-9. This analysis indicated that water flowing from the TMF may potentially discharge at three locations: a ravine approximately 500 m to the south of the TMF; a small pothole pond approximately 1.0 km to the southeast of the TMF; and Patterson Lake, approximately 3.2 kms to the east of the TMF. These are the potential receptors for any solutes released from the TMF (Figure 18-10).

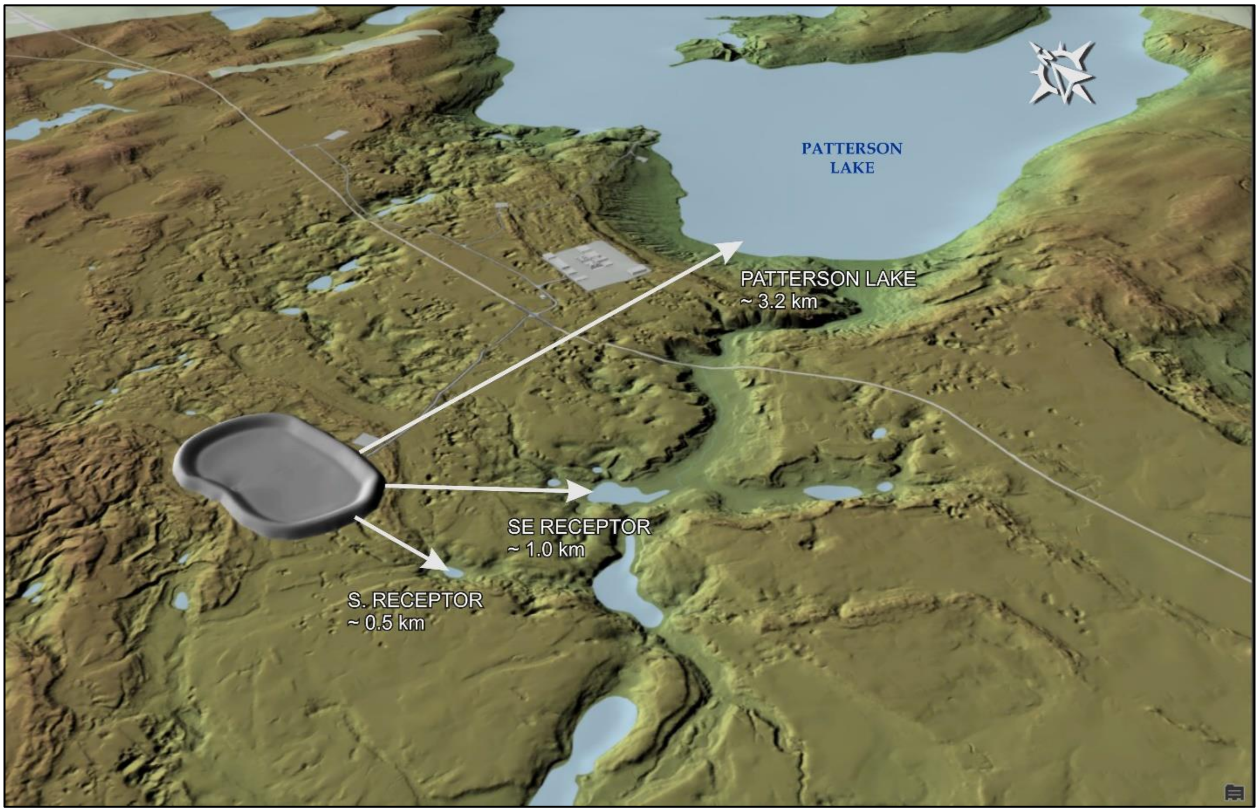


Figure 18-10: Anticipated TMF Leachate Receptors

Further numerical modeling was carried out to estimate relative environmental impacts that might report to each receptor over the design life of the facility. Different chemical species will migrate at differing rates depending on factors such as the degree of attenuation by the engineered double barrier, country soil and groundwater system and other factors such as radioactive decay with time. Common ions such as sulphate (SO_4) and chloride (Cl) are generally assumed to be unattenuated in the soil while species such as arsenic (As) may be attenuated and radium (Ra-226) may be both attenuated and undergo decay with time.

Preliminary analyses indicate that the engineered double barrier system will be effective in attenuating migration of metals (As) and radionuclides (Ra-226); modeling indicates that these species are not likely to break through to the water table during the 10,000-year design life of the TMF.

The potential migration of chloride was modelled to confirm potential receptors, inform potential locations for groundwater monitoring piezometers and solute recovery wells, and estimate impacts to the environment. Figure 18-11 illustrates the flow pathway of chloride in the groundwater system and demonstrates that if release from the TMF did occur, the three identified receptors could be impacted to varying degrees. The estimate of chloride migration in relation to receptors is presented in Figure 18-12.

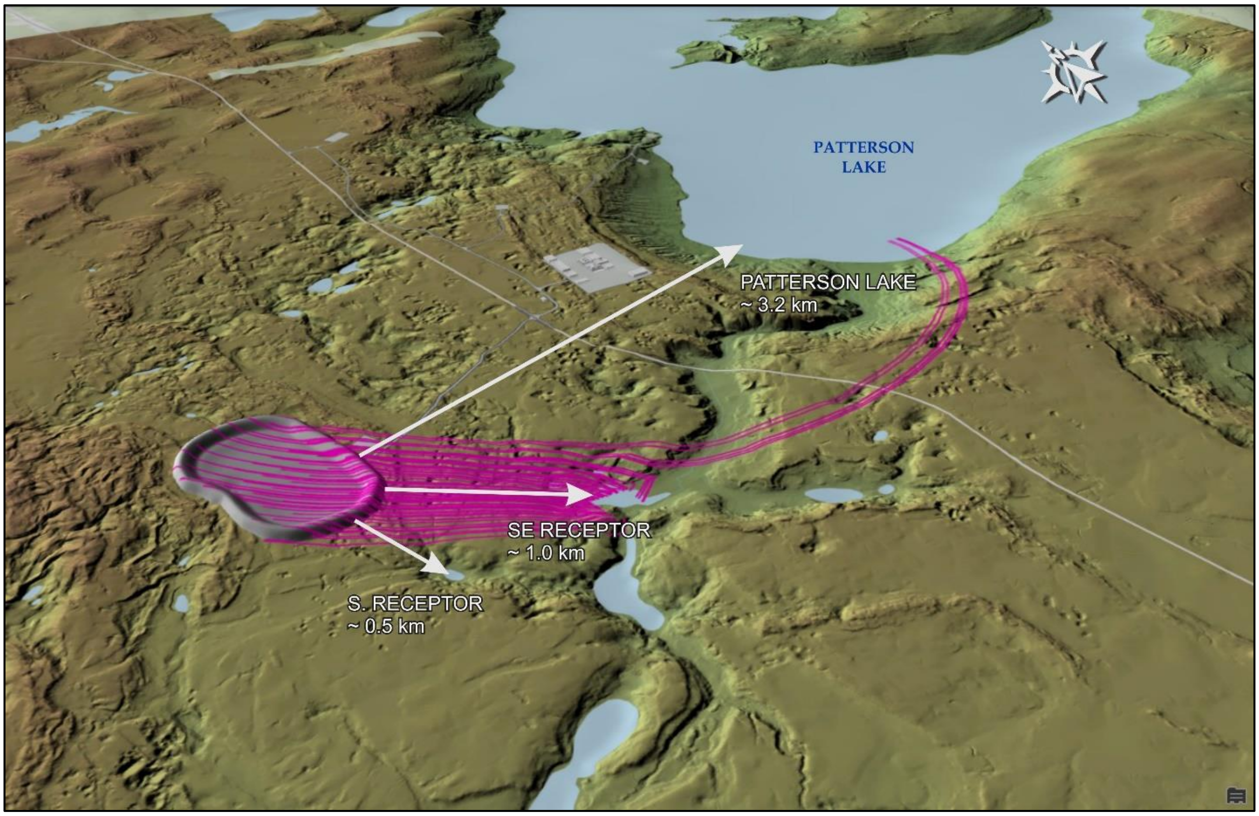


Figure 18-11: Particle Tracking Analysis Showing Solute Migration Pathways to Potential Receptors

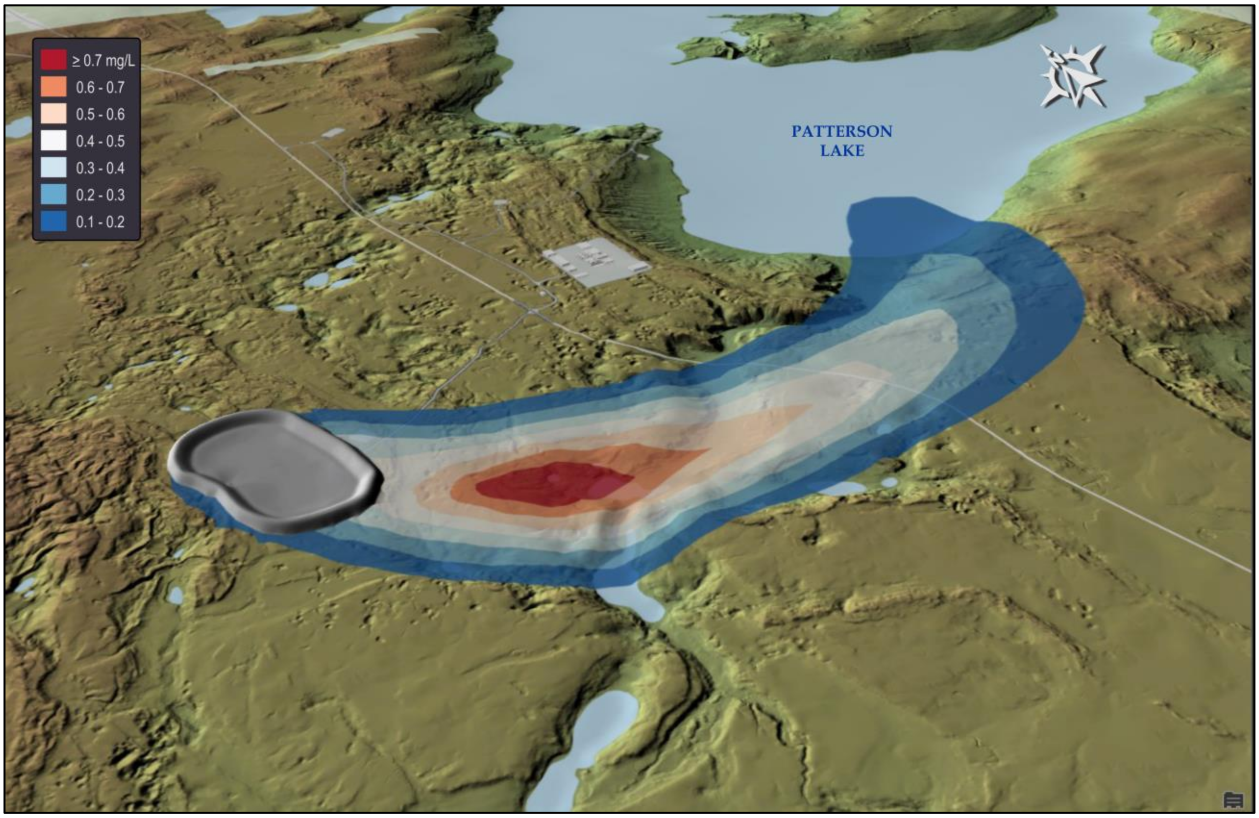


Figure 18-12: Estimate of the area impacted by chloride transport after approximately 10,000 years. Contours are concentration in mg/L. Source concentration of the TMF pore water is approximately 60 mg/L.

18.6.4 TMF Construction

Construction of the TMF will follow a phased approach starting with site preparation (Phase 1), followed by two stages of TMF construction and operation (Phase 2), and finally, construction for closure including transition/active care (Phase 3).

Phase 1 site preparation will include preparing the site for construction, by clearing and constructing the required infrastructure such as access roads, laydown areas, dewatering retention ponds, and installation of wells. Aggregate is required for the perimeter and basal drains and is available in necessary quantities in and adjacent to the TMF. Part of the site preparation will include excavating, crushing/screening, and stockpiling aggregate that is available within the TMF footprint and directly north of the TMF area.

Phase 2 TMF construction and operation has been designed in two stages. Stage 1 is construction of the berms using materials excavated below the site original ground surface, after the aggregate has been removed. This stage is termed the "earth balance" stage. After the berms have been constructed to the balanced design, the pump station, underdrain, floor liners, floor drains, and wall liners and drains will be constructed to support process plant operation. The TMF constructed to the balanced design can be operated for approximately four years before the remainder of the berm height will need to be constructed.

The Stage 2 construction will require additional borrow material to construct the berms to the final elevation. This stage is termed the "Ultimate Stage" and can be constructed either progressively over the remainder of the TMF operating life or in a single event allowing for optimization of operational expenditures.

During the Stage 1 construction and operation, robust environmental and geotechnical instrumentation/monitoring networks will be installed. The environmental monitoring network, consisting of monitoring wells and air quality monitoring stations, will be installed to facilitate collection of baseline conditions. The geotechnical instrumentation will be installed in stages. Instrumentation is required during construction to support the observational method, while post construction instrumentation is required to monitor performance and to assess long term performance.

Phase 3 of the TMF construction is transition and active care and will be the start of the closure process. The water cover will be removed from the tailings and all additional waste placed in the TMF. The tailings will then be covered with approximately 5 m of soil to promote consolidation and decrease the time active care will be required (active care is the process of monitoring consolidation and treating pore water that is expelled during consolidation). The final part to TMF closure will be constructing the engineered soil cover, decommissioning the water treatment facilities, disposal of sludges and other contaminated materials, contouring the TMF cover and integrating it into the local area as a landform.

Figure 18-13 provides a preliminary construction, operation, and decommissioning schedule.

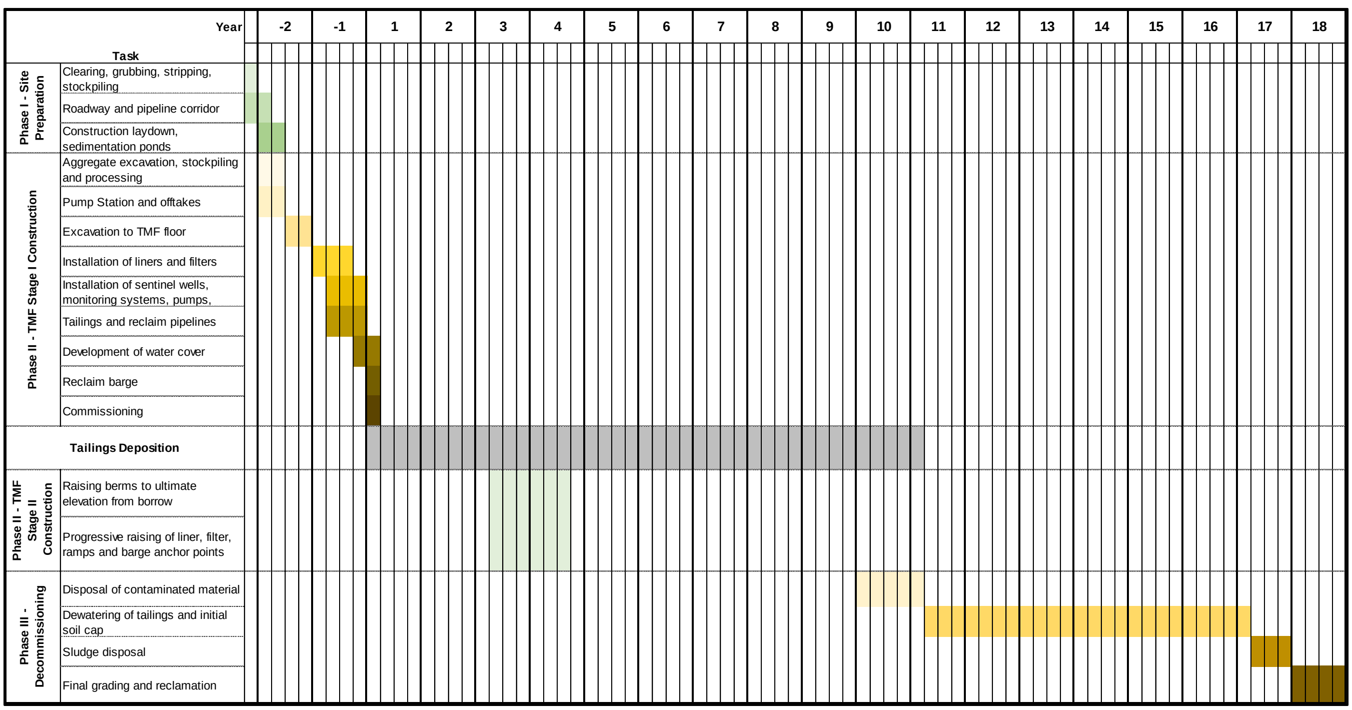


Figure 18-13: Preliminary TMF life schedule.

19.0 MARKET STUDIES AND CONTRACTS

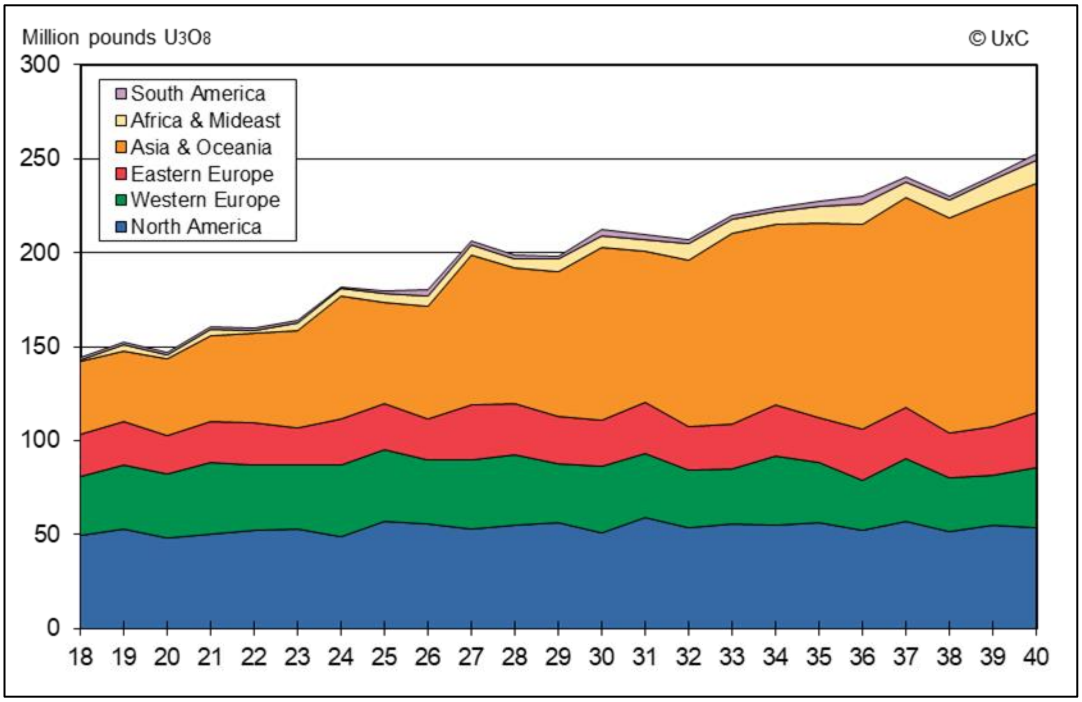
19.1 Market Overview

The principal commodity of the Project is U_3O_8, a uranium concentrate commonly known as yellowcake. The primary end-use for yellowcake is in the manufacturing of fuel bundles which are used in nuclear power plants that produce electricity. Yellowcake is sold between producers and end-users or intermediaries, and is sold both under long term contracts and in the spot market.

19.2 Market Demand

The demand for yellowcake is directly correlated with the global demand for nuclear energy. The demand for nuclear energy is in turn driven by an overall increased demand for electricity, as well as a shift away from carbon-based fuel sources. It is estimated that global consumption for electricity had grown from 10,000 TWh in 1990 to 26,760 TWh in 2020. According to the International Energy Agency (IEA) 2021 World Energy Outlook, expected consumption is expected to grow by 75% to 46,700 TWh by 2050. According to the International Atomic Energy Agency (IAEA), demand for nuclear fuel is expected to increase, with an estimated 56 new reactors currently under construction.

Market demand is shown graphically in Figure 19-1.



Source: UxC, LLC, Q3 2022

Figure 19-1: World Uranium Requirements, Base Case, 2018-2040

19.3 Market Supply

The supply of yellowcake can come from two sources: primary (uranium mines) and secondary (recycled uranium, re-enrichment of uranium, and conversion of military uses). The primary uranium market is relatively concentrated, with very few companies, countries, and projects accounting for the majority of primary uranium supply. In 2021, 10 companies accounted for approximately 84% of global primary uranium production, as shown in Table 19-1.

Table 19-1: Primary Uranium Market Supply by Company

Position	Company	2021 Production (Mlb U_3O_8)	Percent of Total	Cumulative
1	JSC Natl Atomic Co Kazatomprom	33.1	26.3%	26.3%
2	Uranium One Inc.	11.6	9.2%	35.6%
3	Orano SA	11.6	9.2%	44.8%
4	Cameco Corp.	9.7	7.7%	52.5%
5	Navoi Mining & Metallurgical	9.1	7.2%	59.7%
6	JSC Atomredmetzoloto	8.5	6.7%	66.5%
7	Taurus Mineral Ltd.	7.7	6.1%	72.6%
8	BHP Group	5.0	4.0%	76.6%
9	Heathgate Resources Pty Ltd	4.9	3.9%	80.4%
10	China National Uranium Corp.	4.4	3.5%	83.9%
--	Remaining	20.3	16.1%	100.0%
--	**Total**	**125.9**	**100.0%**	**--**

Note:
Data compiled from S&P Global CapitalIQ.

The top 10 uranium mines, by production, are shown in Table 19-2.

Table 19-2: Primary Uranium Market Supply by Project

Position	Project	Location	Mining Method	2021 Production (Mlb U_3O_8)	Percent of Total	Cumulative
1	Cigar Lake	Canada	UG	12.2	9.7%	9.7%
2	Central Mining District	Uzbekistan	ISR	9.1	7.2%	16.9%
3	Inkai	Kazakhstan	ISR	9.0	7.1%	24.1%
4	Husab	Namibia	OP	8.6	6.8%	30.9%
5	Katco	Kazakhstan	ISR	7.4	5.9%	36.8%

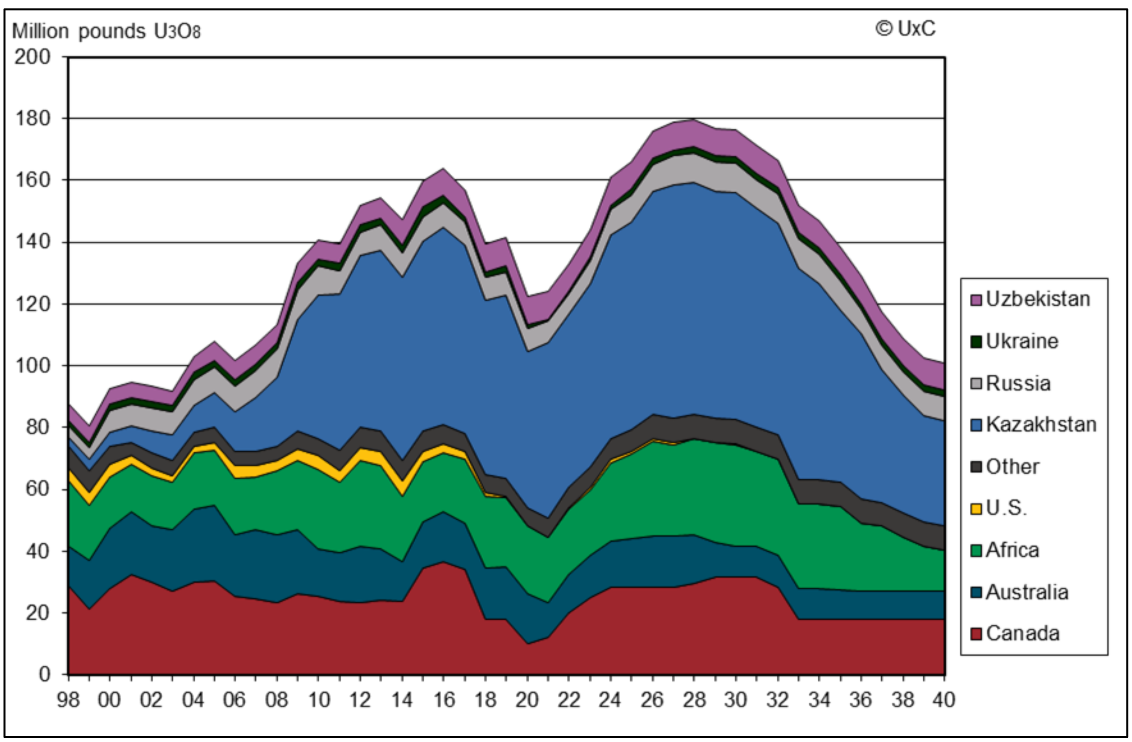
table continues…

Position	Project	Location	Mining Method	2021 Production (Mlb U_3O_8)	Percent of Total	Cumulative
6	Karatau	Kazakhstan	ISR	6.7	5.3%	42.1%
7	Rossing	Namibia	OP	6.4	5.0%	47.1%
8	Somair	Niger	OP	5.2	4.1%	51.2%
9	Olympic Dam	Australia	UG	5.0	4.0%	55.2%
10	Beverley	Australia	ISR	4.9	3.9%	59.0%
--	Remaining	--	--	51.6	41.0%	100.0%
--	**Total**	**--**	**--**	**125.9**	**--**	**--**

Notes:
1. Data compiled from S&P Global CapitalIQ.
2. ISR stands for In-Situ Recovery, UG stands for Underground, and OP stands for Open Pit.
3. Uranium from Olympic Dam is recovered as a by-product.

It is noted that Cameco Corp.'s (Cameco) McArthur River mine, which had been in a state of care and maintenance since 2018, has been restarted. In November 2022, McArthur produced its first batch of yellowcake. Cameco intends to ramp up production at McArthur River to 15 million lb per year by 2024. This would place it amongst the top two primary uranium producers globally.

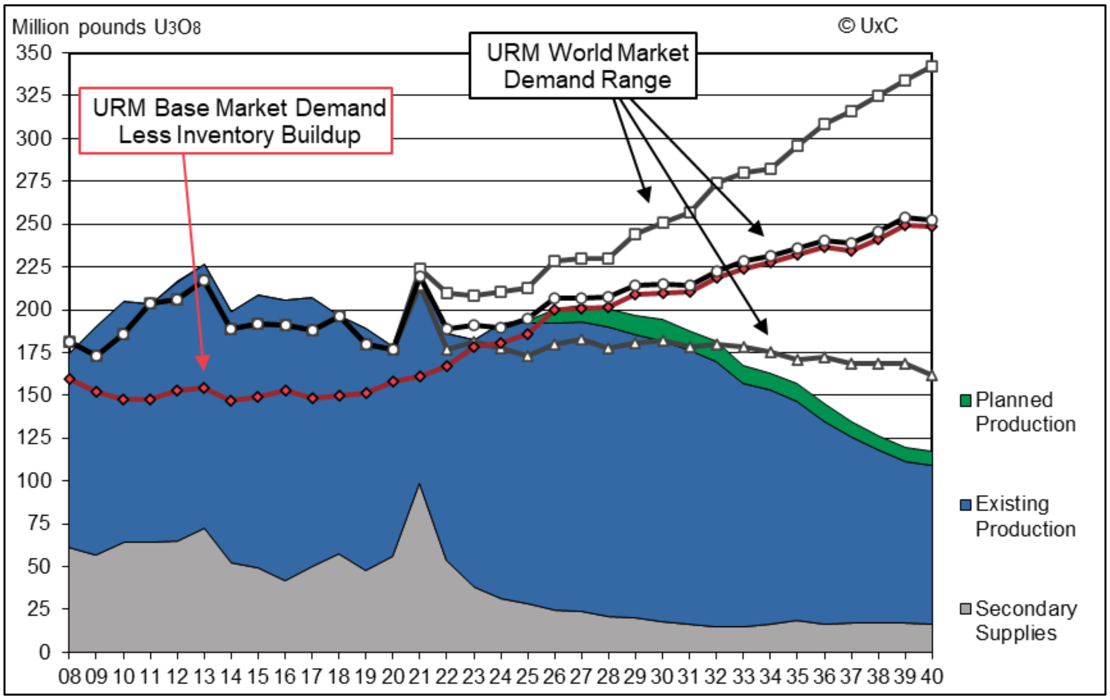
Historical and forecasted uranium production, which excludes the McArthur River restart, is shown in Figure 19-2.



Source: UxC LLC, Q3 2022

Figure 19-2: Existing and New World Uranium Production, 1998-2040

The expected market demand, coupled with a decline in supplies, could result in a supply gap beginning in the second half of the 2020s, as shown in Figure 19-3.



Source: UxC LLC, Q3 2022

Figure 19-3: Market Demand vs. Mid-Case Production Sources, 2008-2040

It is noted that this figure does not account for other primary supply sources that could be brought into production in the event of rising prices.

19.4 Market Prices

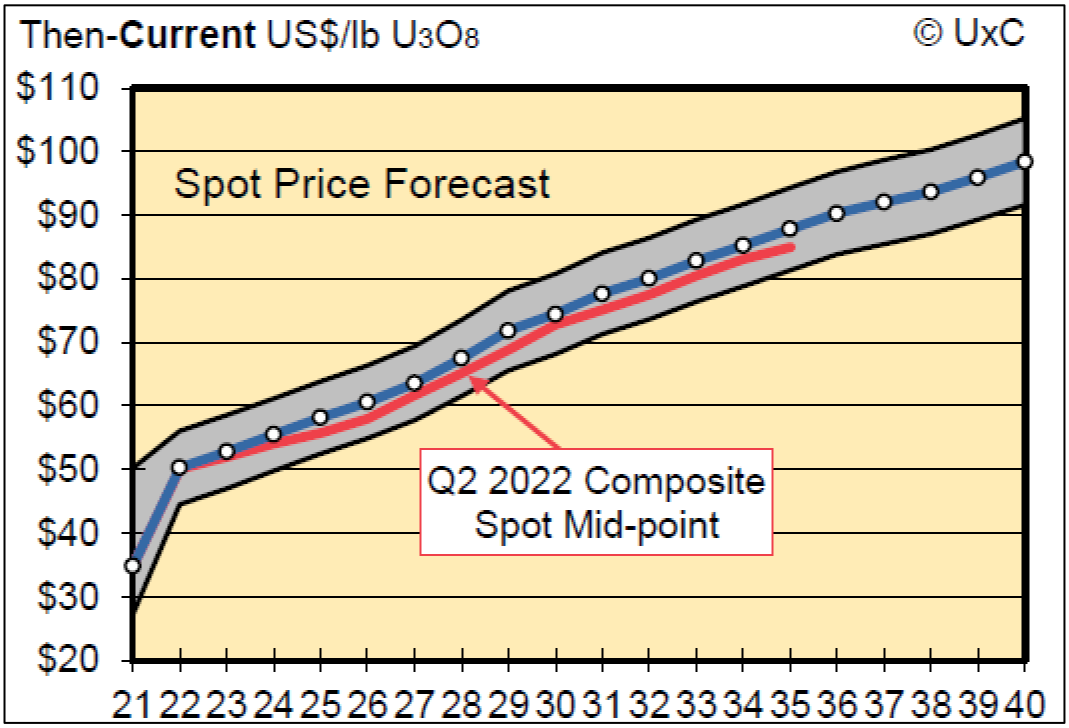
Since 2005, spot uranium prices have varied considerably between US$21/lb U_3O_8 and US$120/lb U_3O_8. Several events have impacted the current spot price over the past decade, starting with the Fukushima-Daiichi nuclear accident in March 2011. A large-scale earthquake and tsunami disabled the power supply and cooling of three reactors, causing radioactive material to be released into the environment. In September 2013, Japan shut down its entire fleet of nuclear reactors pending a safety review. The first reactor was restarted in August 2015, and by the summer of 2023, it is forecasted that 17 reactors will be operating again, representing approximately half of the pre-Fukushima total. The historical spot price is shown in Figure 19-4.

Source: S&P Capital IQ

Figure 19-4: Historic Spot Uranium Price

The extended closure of Japan's nuclear power plants had caused a supply glut, as utility companies were no longer consuming uranium and selling what they had already purchased back into the spot market, causing the price of U_3O_8 to decline. This supply glut has corrected itself over the last number of years, with major uranium producers curtailing production in order to rebalance the market. A number of geopolitical events have caused the spot price of U_3O_8 to more than double over the past five years from US$23/lb U_3O_8 to US$54/lb U_3O_8. Several nuclear reactors that were slated for early closure have had those decisions cancelled. Other nuclear reactors that were approaching their useful life have been refurbished, prolonging their operations by an average of 20 years. Additionally, the invasion of Ukraine by Russia has disrupted the global energy market, causing many countries to re-evaluate where and how they will obtain a reliable source of fuel for their nuclear power plants. Small modular reactors have also gained prominence as a means to power remote, isolated areas, with a number of countries exploring their applicability.

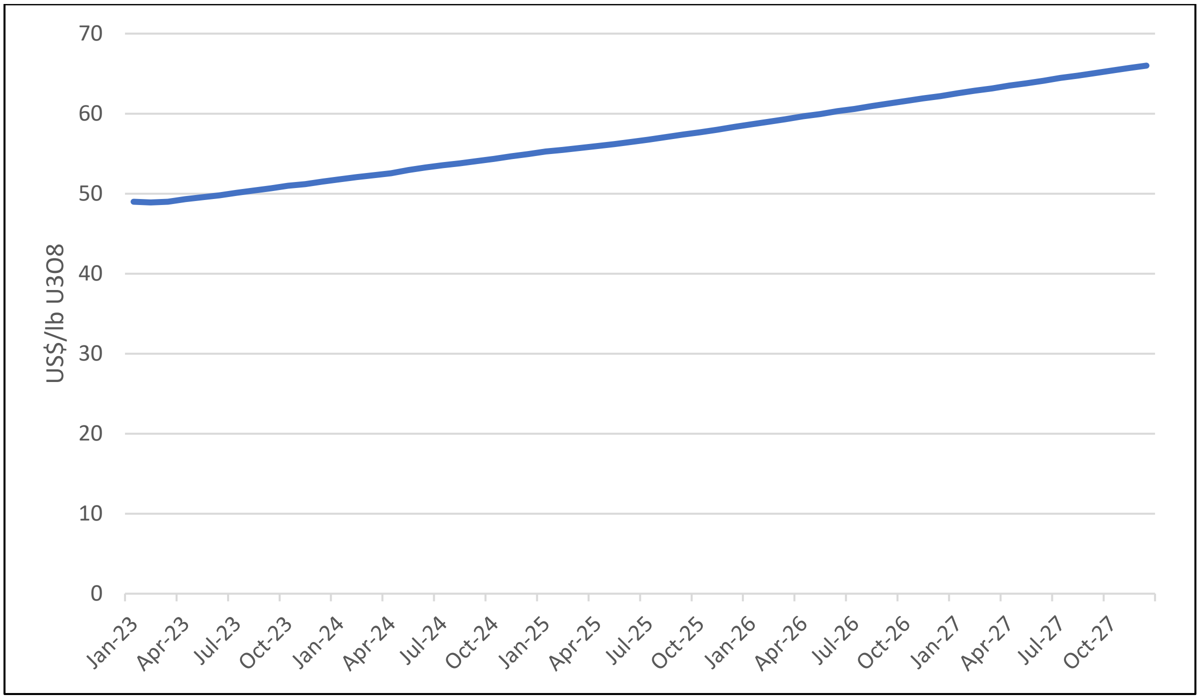
Based on renewed interest in nuclear energy, as well as some uncertainty around replacing primary supplies of U_3O_8, the price of U_3O_8 is forecasted to rise, as shown in Figure 19-5.



Source: Uranium Market Outlook, Q3 2022, UxC LLC

Figure 19-5: Historic Spot Uranium Price

Furthermore, the CME Group (formerly NYMEX) publishes a U_3O_8 futures price. It is noted that these futures contracts are a financial instrument and do not represent physical delivery of U_3O_8. They are used occasionally by producers and consumers of U_3O_8 to hedge contracts. The futures price is shown in Figure 19-6.



Source: CME Group, January 2023

Figure 19-6: UxC Uranium U_3O_8 Futures

19.5 Conclusions

Based on the forecasted U_3O_8 price as well as the futures market, and considering the Project is still years away from production, it is reasonable to assume a flat price forecast of US$65/lb U_3O_8 for the duration of the Project.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACTS

20.1 Introduction

The Patterson Lake area represents a potential new mining region in the western Athabasca Basin with several discoveries in the area with the potential to be developed. Two of these discoveries, including the Triple R deposit of FCU's PLS Project (the Project), have entered the provincial environmental assessment (EA) process, which will include considerations of cumulative impacts with other local projects. The only previous producing westside mine was Orano's now decommissioned Cluff Lake mine that was served by Highway 955, which runs past the PLS site. The potential impacts from a uranium project in northern Saskatchewan are well known, and well regulated, with oversight from both the federal and provincial governments and local communities. The environmental performance of modern uranium mines has been very good and is well documented by the CNSC. The protection of the environment and human health and safety will continue be key areas of focus for project success.

This section is based on examination of available literature and reports available either online or supplied by FCU (either directly or through its consultants), discussions with FCU management and personnel, discussions with consultants and regulators, and several past site visits. While documentation was reviewed, it was not an audit nor an exhaustive assessment of compliance. The focus was on items that might be material to the FS and, or with the potential to impact the progress of the Project as it moves towards production.

The PERA that was conducted for the PFS has been reviewed and updated for the FS. The PERA was conducted for the PLS Property and designed to incorporate a level of detail consistent with the feasibility stage of the Project. It examines what is proposed regarding site facilities, areas of physical disturbance, effluent releases, emissions to the environment, and makes an estimate of the potential impacts after mitigation. A PERA is not an exhaustive examination of potential environmental interactions but is designed to focus on those with the most potential for significant impacts.

20.1.1 Site Setting

The following describes the main environmental components of the Project as they may be affected by the proposed project design. Combined with the current preferred development options, the following are some of the key factors considered in the PERA discussion.

20.1.1.1 Geology

- The ore deposit is covered by a 50 m to 80 m thick sequence of overburden that allows a pathway for migrating fluids.
- Location of the R780E zone, the largest ore body, is directly under Patterson Lake.
- Handling and disposal of waste rock and the determination of its acid generating or leaching potential.

20.1.1.2 Hydrology

- Patterson Lake is the first lake downstream of Broach Lake, which is the headwater lake of the Clearwater River system. Consideration of the following is necessary to understand the need to protect water:
  - Cumulative effects of another mining operation discharging treated mining and sewage effluents into the lake upstream of the Project.

  - Potential impacts to traditional users.
  - Preston Lake Wildlife Refuge (permanent, legislated), approximately 40 km downstream. Preston Lake Wildlife Refuge consists of an island located in Preston Lake at 57° 24’ N, 109° 11’ W from *The Wildlife Management Zones and Special Areas Boundaries Regulations*, 1990.
  - Clearwater River’s Heritage River designation.
  - Clearwater River Provincial Park (48 km SSE, longer by river), per the *Parks Act*.
  - Clearwater River drains into Alberta and joins the Athabasca River at Fort McMurray approximately 300 km downstream.
  - Protection of water quality will be paramount.
- Relatively flat terrain to west of Patterson Lake with drainage to Alberta a potential concern:
  - Marguerite River Wildland immediately along the AB/SK border in AB (38 km due west). This is where water would drain to if discharged into the Alberta watershed.
  - Maintain project within one Saskatchewan watershed (Clearwater River).

20.1.1.3 Hydrogeology

- Understand groundwater movement from the TMF towards Patterson Lake.
- Understand the potential for groundwater movement from the overburden into the decline and shafts, and from the overburden through to bedrock and its impact to underground mining voids.
- Protection of Patterson Lake and the Clearwater River drainage and its biota.
- Quantity and quality of groundwater.

20.1.1.4 Ecological

- Minimizing impacts to the local environment.
- Considerations of rare and endangered species.
- Main terrestrial species of concern is caribou.
- Main aquatic species of concern is lake trout.
- Potential conflicts with rare and endangered plants.

20.1.1.5 Social and Economic

- First Nations and Métis traditional use and resource harvesting in the area, including hunting, fishing, and trapping.
- General use of the area, such as Patterson Lake’s use for fishing, including commercial, subsistence, and recreational use.
- Consultation and engagement outcomes, and feedback on project components.
- Impact communities (e.g., communities on the west side along Highway 955), cost/benefit discussion.

The above list of potential of effects will be updated and modified in the EIA process as it is informed by the Indigenous engagement activities undertaken as part of the Indigenous engagement agreements (described below) and interest-based community discussions.

20.2 Environmental Baseline Program

Modern EIAs require significant environmental and social baseline data to predict potential impacts through an ERA type process supported by pathways modelling and analytics, which in turn informs the design of appropriate mitigations. Since the start of their exploration program, FCU has contracted CanNorth to undertake environmental baseline work, which has included data collection programs starting in 2013. In July of 2020, CanNorth submitted a comprehensive environmental baseline report designed to prepare the Project for future licensing and regulatory requirements. The programs collected a full suite of environmental data including climate and meteorology; noise; air quality; surface water hydrology; water and sediment quality; plankton, benthic invertebrate, and fish communities; fish habitat, chemistry, and spawning; ecosite classification; vegetation communities; wildlife communities; species at risk; and heritage resources (CanNorth 2020). Between 2013 and 2022, the hydrological and meteorological monitoring stations were visited to extend these datasets. Additionally, in 2015, CanNorth submitted a site condition and reclamation report in support of local reclamation initiatives by FCU (CanNorth 2015).

In 2021, FCU commissioned CanNorth to complete an environmental baseline program to provide updated information for the EIA and to investigate additional study areas for Project planning. The program is using current best practices and the most up to date survey protocols. A comprehensive report on this program will be submitted to FCU in early 2023.

20.2.1 Aquatic Resources

The aquatic study areas (ASA) in the 2020 and 2022 studies included numerous lakes and bays of Patterson Lake located in the vicinity of the Project, as well as numerous waterbodies downstream (Beet Lake, Naomi Lake, Clearwater River, and Lloyd Lake). Water quality results illustrated that lakes in the ASA contained adequate dissolved oxygen, near neutral pH, and low levels of nutrients, ions, metals, and radionuclides. The only parameters that exceeded the SEQGs in some waterbodies were iron, aluminum, and copper.

At least 15 species of fish are known to be present in the ASA, including lake whitefish, northern pike, walleye, white sucker, lake trout, and yellow perch. None of these species are of conservation concern. Aquatic habitat mapping indicated a variety of habitat types are present in the ASA, with suitable habitat for fish spawning, rearing, feeding, and overwintering provided by most waterbodies. Metal and radionuclide concentrations in northern pike and lake whitefish flesh and bone samples from the ASA were frequently below laboratory detection limits.

Continuous hydrological monitoring stations were established at the inflow and outflow to Patterson Lake and three other stations were established along the effluent flow path between Patterson Lake and the Clearwater River. At the outlet of Patterson Lake, the 1:100 year high and low flows were predicted to be 5.28 m^3/s and 0.44 m^3/s, respectively. Due to the length of the hydrological monitoring program, both wet (2014 and 2020) and relatively dry years (2017) have been observed throughout the ASA.

20.2.2 Terrestrial Resources

The 2018 and 2022 terrestrial baseline programs included a combination of database searches for rare, at-risk, or sensitive plant and wildlife species, along with intensive field programs to assess the vegetation, wildlife, and wildlife habitat present within the Project area. The 2018 and 2022 program included vegetation and landcover classification and mapping, ecosite classification, rare plant surveys, a winter tracking survey, a breeding water bird and raptor nest aerial survey, a breeding bird survey, an amphibian acoustic survey, an ungulate pellet group/browse survey, a small mammal survey, a semi-aquatic furbearer shoreline survey, and bat surveys. For this program, the terrestrial study areas were centred on the Project footprint and included the Site Study Area that was approximately 25 km^2, the Local Study Area (LSA) that was approximately 225 km^2, and the regional study area (RSA) that was approximately 2,025 km^2 (45 km by 45 km) to adequately document those species with larger home range sizes. The 2022 program included many of these same surveys that were completed in areas that were not assessed in 2018 and 2019 and collected additional wildlife and vegetation data on a smaller scale that was more focused on the Project footprint.

Terrestrial species considered to be Species of Conservation Concern (SOCC) are defined as federally and provincially legislated species that are identified on federal and provincial tracking lists and activity restriction guidelines. This includes:

- Species listed under Schedule 1, Schedule 2, or Schedule 3 of the federal *Species At Risk Act* (SARA) as endangered, threatened, or special concern);
- Species listed by the Committee on the Status of Endangered Wildlife in Canada (COSEWIC) as endangered, threatened, or special concern that are not listed under SARA;
- Species listed as endangered, threatened, or vulnerable in *The Wildlife Act*;
- Species listed as ranking of S1, S2, or S3 or a combination of rankings (including an applicable rank modifier such as “M” for migrant) by the Saskatchewan Conservation Data Centre (SKCDC); and
- Species included in the Saskatchewan Activity Restriction Guidelines for Sensitive Species.

In 2018 and 2022, a total of 19 vegetation SOCC were identified in the Project areas, with conservation rankings ranging from S1 to S3 (vulnerable/rare to uncommon). It should be noted that none of these are federally protected species. The provincial Activity Restriction Guidelines for Sensitive Species apply to vegetation species with conservation rankings between S1 and S3, thus, mitigation for these species may be required depending on the location of project components.

In 2018, 2019, and 2022, a total of 13 wildlife SOCC were observed during wildlife field surveys and incidentally throughout the Project areas (Table 13-1). Nine of these species are listed federally as species at risk. The Project area is considered to provide a moderate to high amount of suitable habitat for the species listed above based on field data and supervised satellite image habitat classification.

Table 13-1 provides a list of rare and endangered wildlife species that have been found by CanNorth in the Project area.

Table 20-1: Rare and Endangered Wildlife Species Within the Project Area: FCU – PLS Property

Scientific Name	Common Name	SKCDC/COSEWIC	SARA Status
Wildlife Species of Conservation Concern			
Contopus cooperi	Olive-sided Flycatcher	S4B/Special Concern	Threatened
Chordeiles minor	Common Nighthawk	S4B/Special Concern	Threatened
Euphagus carolinus	Rusty Blackbird	S3B/Special Concern	Special Concern
Gavia stellata	Red-throated loon	S1B/-	--
Gulo	Wolverine	S2/Special Concern	Special Concern
Hirundo rustica	Barn Swallow	S4BSpecial Concern	Threatened
Lontra canadensis	River otter	S3/-	--
Pandion haliaetus	Osprey+	S3B/-	--
Pinicola enucleator	Pine grosbeak+	S2B/-	--
Podiceps auritus	Horned grebe	S5/Special Concern	Special Concern
Rangifer tarandus caribou	Woodland Caribou	S3/Threatened	Threatened
Myotis lucifugus	Little Brown Myotis	Endangered	Endangered
Myotis septentrionalis	Northern Myotis	Endangered	Endangered

While the baseline studies have found both plant and wildlife SOCC throughout the Project area, there do not appear to be any issues that would require material or unusual mitigations. There is nothing identified that would conflict with currently proposed activities, however, this will be re-examined in depth in the EIA and, if necessary, a mitigation strategy will be proposed.

The ENV has been working on Range Planning for woodland caribou in Saskatchewan since Environment and Climate Change Canada (ECCC) issued their “Recovery Strategy for the Woodland Caribou, Boreal population, in Canada – 2012” (the Recovery Strategy). In its report, in response to a finding that woodland caribou are threatened in Canada, ECCC assessed populations in each province and set a minimum undisturbed habitat limit of 40% to 65% to allow woodland caribou populations to remain stable or grow. Provinces are expected to meet this commitment.

Saskatchewan was divided into two major ranges: SK1, the Boreal Shield, which covers most of northern Saskatchewan including the Athabasca Basin areas where a threshold of 40% undisturbed habitat was identified to ensure a sustaining population; and SK2, the Boreal Plain region that covers, for the most part, the commercial timber region of the province where a threshold of 65% undisturbed habitat in the range has been identified to ensure a sustaining population. The province further divided SK2 region into three sub areas for management: western, central, and eastern. The Project area lies in the very northwestern corner of the SK2 west zone very close to the border with SK1, an area of SK1-like terrain with similar sparse development. Currently, the SK1 range is considered “likely self-sustaining” and the SK2 range is considered “likely not self-sustaining”.

Currently, the province has a draft plan in place for SK2 East, while the final range plans were released for SK2 West in October 2021 and SK2 Central in 2019. The province is currently initiating range planning in SK1. The province is working to comply with ECCC’s recovery strategy. Saskatchewan understands the need to balance economic activity with woodland caribou protection and entered into a Section 11 Agreement under SARA, which allowed the province to enter into a conservation agreement with ECCC to benefit a species at risk or enhance its survival in the wild. The agreement must describe conservation measures consistent with SARA and may include measures to protect critical habitat. It is not clear at this time what affect that would have, if any, on developing a Caribou Mitigation Plan (CMP) for the Project.

FCU has been an active participant in provincial consultations related to the Recovery Strategy especially related to the SK2 West area. For the EIA, a CMP will be required that describes the proposed mechanisms for mitigating potential impacts to woodland caribou. The plan must describe, in descending preference, how the company's activities will avoid or minimize impacts to woodland caribou habitat and, if the impacts cannot be avoided or suitably minimized, propose enhanced reclamation and/or offsetting compensation. Currently, human-caused disturbances are calculated on the actual disturbed area plus a 500-m buffer, and the disturbance will be deemed to last 40 years or more (the time to regrow vegetation in the area to be considered functional woodland caribou habitat). For instance, for a temporary road, the area affected will be the length of the road plus the 500-m buffer, so a 10-km temporary road would have a footprint of at least 10 km^2. The 40-year countdown to becoming woodland caribou habitat would not begin until the road has been decommissioned, reclaimed, and the revegetation is self-sustaining.

Since Saskatchewan has indicated it will not use woodland caribou or the habitat rating to stop development (at least not in the near term), the level of development in an area and the type of woodland caribou habitat will have a large bearing on the level of mitigation and/or offsets required. In the Patterson Lake area, there is a conflict between the provincially assigned level of woodland caribou habitat (Tier 1, or best habitat), the existing level of disturbance (human-caused disturbance and fire disturbance), and the observed ongoing use of the area by woodland caribou despite the disturbance (from Alberta collar data, local observations, and recent surveys). There is also considerable controversy about the inclusion of fire and lakes in the definition of disturbed habitat. If monitoring can demonstrate that there is a stable level of woodland caribou in the area, this would meet the goals of the Recovery Strategy and allow the disturbance necessary to support development. FCU will have to make the case based upon the baseline work and population studies in the EIA.

In earlier heritage resource work on the property, one site was identified for avoidance (HjOi-2), or if avoidance is not possible, a formal archaeological excavation would be required prior to any disturbance. In 2022, the updated project plans were sent to the Heritage Conservation Branch who requested that an additional Heritage Resources Impact Assessment (HRIA) be completed on the new areas. Two CanNorth archaeologists assisted by one northern community member completed the HRIA in September of 2022 utilizing a combination of pre-impact and post-impact methodology. No known or new archaeological sites were found during the program and HjOi-2 was not found to conflict with the Project.

20.2.3 Consultation, Engagement, and Socio-Economics

Consideration of the human environment in EIA is a rapidly growing and evolving regulatory and social requirement. The human environment components selected for consideration in EA vary according to project activities, environmental setting, and key areas of interest or concern identified by regulators and communities. The components identified generally relate to social, cultural, health, and economic settings and effects. To identify those key areas of community interest and concern that require assessment, engagement with potentially affected groups is required. Engagement with potentially affected groups to identify, evaluate, and mitigate potential impacts can help inform regulatory approvals and licensing requirements and set a path toward achieving social license to operate which is expressed as community support.

Engagement in Saskatchewan is described as two separate but intertwined processes: the Duty to Consult and Interest Based Engagement. The Government of Saskatchewan has the Duty to Consult First Nations and Métis groups on any decision with the potential to affect Aboriginal or Treaty Rights. Similarly, the Government of Canada has the Duty to Consult Indigenous groups on any federal decision with the potential to affect Aboriginal or Treaty Rights. As the Project progresses through the regulatory process, several provincial and federal decisions will be made that must be informed by consultation with potentially affected Indigenous groups. Implementation of the Duty to Consult is guided by a combination of provincial and federal regulatory guidance documents and internal review procedures. At the federal level, the CNSC provides guidance on consultation in regulatory documents, including REGDOCs 3.2.1 *Public Information and Disclosure*, 2018 and 3.2.2 *Indigenous Engagement*, 2022*,* and Indigenous Services Canada has published *Aboriginal Consultation and Accommodation – Updated Guidelines for Federal Officials to Fulfill the Duty to Consult,* March 2011. The province of Saskatchewan provides guidance in the *Proponent Handbook – Voluntary Engagement with First Nations and Métis Communities to Inform the Government's Duty to Consult Process* (2013) which reflects the regulatory process outlined in the *First Nation and Métis Consultation Policy Framework, 2010*.

Although the Duty to Consult lies with the federal and provincial governments, the procedural aspects of the Duty to Consult are frequently undertaken by the proponent. FCU received a list of consultation activities from ENV to be undertaken by FCU, which will be used by the provincial government to inform its Duty to Consult. Among other things, FCU will be expected to meet with each potentially affected community to discuss a consultation plan and appropriate budget. The consultation plan should include opportunities to inform communities of the nature of the proposed activities, as well as their potential impacts and proposed mitigation strategies, and to receive feedback or information on current traditional land uses and potential impacts to Treaty and Aboriginal rights. FCU will also be expected to work with the communities to determine reasonable accommodations (e.g., an impact benefit or other agreement) to avoid, minimize, or mitigate adverse impacts.

FCU began engaging with local communities in 2011. Since 2020, FCU has worked with community leadership from Clearwater River Dene Nation (CRDN), Métis Nation – Saskatchewan, Buffalo River Dene Nation (BRDN), Birch Narrows Dene Nation (BNDN), Ya' thi Nene Land and Resource Office (YNLR), and Athabasca Chipewyan First Nation (ACFN) to develop agreed-upon engagement approaches, work plans, and budgets. By the end of 2022, FCU had finalized engagement and capacity agreements with all the Indigenous groups identified as potentially affected by the Project.

The engagement agreements will guide engagement activities between FCU and each community in a way that aligns with community protocols, expectations, and capacity. Several community-based studies were initiated in 2022 and will be completed in early 2023 to inform the EIA. In early 2023, once FS results are available, FCU intends to visit each community to formally introduce the Project to community members. FCU will continue to meet quarterly, at minimum, with each community for the duration of the EA process. A suitable post-EA meeting schedule and frequency will be identified to support ongoing engagement with each community near the end of the EIA process.

As part of the EIA process, FCU will also be required to conduct interest-based engagement with key stakeholders and members of the public. Unlike the Duty to Consult which aims to identify impacts to Aboriginal and Treaty Rights, interest-based engagement focuses on sharing information and collecting feedback on potential effects that may not be tied to Aboriginal and Treaty Rights.

Engagement with municipalities and key stakeholder groups has been ongoing for some time, albeit at a slower pace during COVID-19. Engagement will be conducted using best practices from the International Association of Public Participation, which focuses on aligning the level of engagement and influence with the level of impact each group may experience. Groups with greater potential to experience impacts as a result of Project activities will be engaged with a higher level of influence on the outcome of the EIA.

The results of engagement and consultation activities with Indigenous communities and stakeholder groups will be used to inform EIA scoping exercises including refinement of valued components, identification of potential project-environment interactions, effect pathways, and groups with potential to be disproportionately impacted. Input will also be used to expand or update baseline conditions, evaluate effects, and identify or refine mitigation measures.

To help ensure that FCU's consultation and engagement efforts are aligned with ENV's expectations, meetings are being held monthly to track progress and identify concerns as they arise. Representatives from the CNSC are also invited to participate in these monthly meetings, but it is expected that once FCU initiates the CNSC licensing process the format through which they engage with CSNC will change.

FCU provides employment and business opportunities to local communities. FCU has provided direct employment during exploration programs for local workers and contractors, including technicians, camp staff, carpenters, drillers and driller's helpers, drivers, and lumber suppliers. In addition, FCU conducts business with Big Bear Camps, which provides food, lodging, some construction and maintenance services, and security, all of which employ and engage local people. Although not always feasible, FCU requests its contractors employ as many local people as possible when completing work on the Project. Total Project expenditures on northern vendors in 2021 was $4.04 million and in 2022 was at least $1.57 million.

FCU is working closely with economic development representatives from local communities to help prepare community members to participate in employment and economic development activities associated with the Project as they arise. These efforts include sharing current employment and contract opportunities with community members, sharing lists of anticipated contracts and employment opportunities expected during construction and operation, supporting community-led training initiatives, and maintaining an open dialogue with communities regarding upcoming economic development goals, service providers, and partnerships.

In 2021, FCU developed a Community Investment Guide which establishes a process to formalize how community organizations can submit funding proposals, and how these proposals will be reviewed and awarded. Since FCU started tracking community investments in 2013, they have contributed over $180,000 to local community initiatives, with over $100,000 contributed in 2022 alone. Priority is given to local community organizations to support culture, education, recreation, wellness, and crisis response initiatives. Some examples of initiatives that FCU has contributed to include the Mining Matters Earth Science Camp, minor sports leagues, holiday food hampers or suppers, safety training initiatives, and community celebrations like Treaty Days and Palmbere Métis Days.

While projects need to be described on a stand-alone basis for the approvals process, there is the potential to share services with other developments in the area, should they proceed. Such services could include: a camp and camp catering, an airstrip, shared power generation facilities, shared medical, ambulance and evacuation facilities, road maintenance, and environmental monitoring. Shared services would provide opportunities for local businesses, as well as benefit the environment and provide operational saving for participating operations.

20.3 Preliminary Environmental Risk Assessment

The PERA is presented in the following tables and represents a review of the Project as it is presented in this FS. The operation assessed is the underground only option with a decline and two ventilation shafts, a process plant and ancillary operations, and a purpose-built TMF. The PERA also considers the likelihood of another uranium mining operation immediately upstream in Patterson Lake in consideration of cumulative impacts. The PERA considers the life cycle of the Project through to decommissioning and reclamation.

There are a number of receptors in the area with the potential to be affected. While they will be assessed in more detail in the EIA pathways modelling, receptors include lake users, including traditional users of Patterson Lake (and by extension, the Clearwater River), wildlife and vegetation, site personnel, and road users. The potential for impact comes from site emissions that include dust, noise, runoff, treated effluent, air emissions, contaminants of concern (COCs), radionuclides, and the potential for an uncontrolled spill or release. The following tables (Table 20-2 to Table 20-6) provide the details of the updated PERA for the proposed Project. This is based upon the current Project design presented in this FS.

Table 20-2: Project PERA Summary Table – Terrain and Habitat Disturbance, FCU – PLS Property

Disturbance	Description	Mitigations	Discussion
Ground clearing	Clearing for all facilities including: ▪ Roads ▪ Process plant pad ▪ Waste/ore stockpiles ▪ Camp ▪ TMF area ▪ Aggregate quarries	Minimization of clearing. Reclamation of unused areas. Keeping facilities as compact as possible.	Remains a major impact to the areas cleared but can be remediated at decommissioning. The goal is to minimize the amount of area disturbed. Develop a CMP. Offsets may be required if residual risks. Minimize impacts on natural drainage.
Direct Impacts to Patterson Lake	Expected to be minor with a dock, water intake, and treated effluent discharge diffuser.	Designed to conform to DFO standard practice. Maintain a riparian buffer between the lake and the Project.	While expected to be of limited impact, offsets may be required for habitat impacts.
TMF	TMF will be required. Will require excavation of a purpose-built pit.	Preferred method is a pervious surround design tailored to local conditions. Design: sub-aqueous deposition with pervious surround and underdrain collection system. Diversion of fresh water around TMF.	TMF designed for long-term stability; to minimize footprint; minimize flux to environment; and ease of decommissioning. Sub-aqueous design eliminates radioactive dust and radon. Will require a TMF Management Program and design assessment per current standards (e.g., Mining Association of Canada [MAC] Tailings Guidance).
TMF operation	TMF with subaqueous deposition of tailings to prevent freezing and radon emission. Underdrain to capture drainage for treatment at the process plant.	Should be little impact. May need some dust control for vehicles on roads. Collected water from underdrain for treatment and discharge. Secondary containment for pipelines. Installation and monitoring of groundwater wells surrounding the TMF.	If sub-aqueous system works as designed, little to no impact during operations and after decommissioning. Will require a TMF Management Program and design assessment per current standards (e.g., CDA and MAC Tailings Guidance).

table continues…

Disturbance	Description	Mitigations	Discussion
Mine ramp (decline)/foresh ore excavation	A decline will be developed to ramp through the overburden and access the ore body below the overburden.	Proper location of excavated material in dry stable area with erosion and sediment control. Material should be clean and not require water collection. Immediate stabilization and reclamation of cut slopes and embankments to minimize erosion and sediment transport.	Use of a tunneling system to safely construct a decline through the overburden. Will require dewatering.
Roadways/ including a relocated Hwy 955	Highway 955 will stay where it is, between the mine/process plant area and the TMF. This has been done at the request of local communities who did not want the road moved.	Develop an intersection with clear signage and visibility. There will be scheduled monitoring for contamination.	Discussion with the MHI will be required for the intersection requirements. Security will be required on both sides of the road. Periodic monitoring for radiation at crossing.
Mining: underground	Underground option with ventilation or access raises.	Handling of waste rock, mine water, ventilation, radiation protection, access, and egress.	Design for single pass air where workers will be present, segregate clean and dirty waste based on ARD potential, mine water collected, degassed for radon, and sent to process plant for treatment.
Ore, sub-ore, and special waste stockpile(s)	Ore storage or blending pads for rock with ARD and leaching potential to prevent the potential contamination of soil, water, and groundwater).	Bermed, double-lined storage pads. Cover with clean waste to prevent dusting. All drainage to runoff collection ponds. Installation and monitoring of groundwater wells surrounding stockpiles. Monitoring to ensure that air quality standards are met.	Water collected and treated prior to release. Ensure not upwind of living facilities to protect from dust or radon emanations. All material not milled will be returned to the underground as backfill. No stockpiles will remain at end of mine life.
Clean waste rock or aggregate	Clean overburden and waste rock with no ARD or leaching potential. Main issues are erosion of stockpiles creating sedimentation issues.	Clean waste with erosion controls and sedimentation barriers. All drainage to runoff to collection ponds or drain into sandy terrain, not directly to surface water. No liners, so some water may drain into the base.	Clean materials available for other uses and reclamation. Currently, all waste rock is expected to be used for backfilling and no residual stockpiles are expected at end of mine life.
Process plant/process plant terrace	Disturbance, runoff, dust and gaseous emissions.	Collect runoff water for treatment, keep pad areas clean, site as compact as possible, develop Wildlife Management Plan. Monitoring to ensure that air quality standards are met.	Ongoing monitoring to ensure compliance.

table continues…

Disturbance	Description	Mitigations	Discussion
Ancillary facilities, including camp, offices, shops, clean laydowns, etc.	Disturbance, contaminated and non-contaminated wastes, potable water, sewage, recycling materials.	Recycling, proper design of water and sewage facilities. Training. Domestic waste handling and hazardous waste handling.	Many recycling programs mandated by law in SK, such as electronics, tires, cardboard/paper, plastics, refundable containers, oil/oil filters, etc.
Wildfire	Is always a threat in the Athabasca region with any given area burning on average every 40 years. Threat to wildlife, people and property.	Have a comprehensive fire prevention and response plan. Work with Saskatchewan Public Safety, Wildfire group.	May include fire breaks, Fire Smart principles, evacuation plans, firefighting equipment and training.

Table 20-3: Project PERA Summary Table – Water, Contaminated and Uncontaminated, FCU – PLS Property

Disturbance	Description	Mitigations	Discussion
Runoff water	Process plant terrace, contaminated stockpiles, mine.	Collection, to process plant for treatment and eventual discharge. Maximize diversion of fresh water from project infrastructure. Full containment of plant island.	Given the sandy nature of the terrain, all areas requiring water to be collected will require some form of treatment, even if just settling out fines prior to discharge.
Mine water	Underground mine and ramps.	Collection, to process plant for treatment and eventual discharge.	Additional grouting may be required to minimize flows during operation.
TMF underdrain and sentinel wells	Tailings dewatering water drains into the underdrain system.	Collection, to process plant for treatment and eventual discharge. Security of tailings and return water pipelines.	Use of the underdrain will ensure no release of contaminants until the desired tailings density is achieved during decommissioning. May require running the treatment system for several years after production stops.
Sewage	Collect and treat from various locales. Final process to be decided.	Collection, to process plant for treatment and eventual discharge, separate sewage treatment plant or septic field.	Final sewage treatment methods have not yet been chosen. Options include TMF disposal.
Treated effluent	Discharged to Patterson Lake and the Clearwater River system.	Final estimates of quantity and quality will be needed for the EIA. Likely will use a diffuser at depth in the south basin of Patterson Lake. Regular monitoring of effluent and receiving environment.	Must meet licensed objectives. This is especially important as there is likely to be another mine discharging to the same system.
Potable water	Collected, treated, stored with reserves for firefighting.	Need inlet and water treatment facilities prior to distribution.	Inlet upstream from discharge point(s) or from groundwater wells. Consideration needed as to location of other discharge points.

table continues…

Disturbance	Description	Mitigations	Discussion
Fuels	Diesel, gasoline, lubricants, propane, LNG.	Storage licensed with ENV Environmental Protection Branch (EPB) in accordance with *Saskatchewan Environmental Code*.	Properly designed and licensed facilities with trained personnel will minimize any risk to the environment. Emergency Response Plan.
Reagents	Various, to be identified in EIA.	Storage licensed with ENV EPB in accordance with *Saskatchewan Environmental Code.*	Proper storage, likely within the process plant terrace area or in the process plant building. Site security. Emergency response plan.
Yellowcake	Produced, drummed, shipped. Prevention of release or spills.	Undertaken in accordance with federal *Packaging and Transportation of Nuclear Substances Regulations* (2015) and *Transportation of Dangerous Goods* Regulations (2001).	Proper storage and tracking. Compliant with Nuclear Non-proliferation Treaty. Additional Protocols. Site security. Emergency response plan.
Explosives	Explosives, if mishandled can cause significant damage and injuries.	Following federal regulations, properly trained, licensed personnel, separate magazines depending on the type of explosive used.	Properly handled, explosives are safe. Security will be required to prevent theft or misuse. Emergency response plan.

Table 20-4: Project PERA Summary Table – Site Air Emissions, FCU – PLS Property

Disturbance	Description	Mitigations	Discussion
Mine air exhaust	Diesel exhaust, radon, radon progeny, dust.	Dilution by having enough fresh air flow, dust control, air quality monitoring.	Modelling in the EA will provide more information.
Mine air conditioning	Greenhouse gases (GHG), probably propane.	Minimize use to the extent practicable.	Conditioning of mine air is a safety item to prevent freezing of collar or portal areas.
Generators	LNG exhaust emissions, GHG.	LNG is a cleaner option than diesel, producing less CO_2 and virtually no particulate matter, NOx, or SOx.	Use of LNG for power generation is a better air quality option than diesel in the absence of power lines. The use of renewables will be examined in the EIA.
Process plant	Various emission sources.	Protection against dust – need capture and baghouse with filters.	Emission sources will be determined and modelled in EIA.
Vehicles	Exhaust – GHG calculation.	Utilize current emissions control standards, maintain equipment, minimize idling.	Consider use of electric vehicles as they become available. Fuel in secure areas to avoid spills.
TMF	Subaqueous, so air emissions should be low.	Water cover eliminates dusting, promotes settling, and minimizes radon emanation.	Releases and long-term impacts to be defined in EIA by pathway modelling.

table continues…

Disturbance	Description	Mitigations	Discussion
Ore and special waste stockpiles	Radon, radon progeny, dust, runoff.	Proper design and monitoring.	Ensure not upwind from camp or offices.

Table 20-5: Project PERA Summary Table – Decommissioning, FCU – PLS Property

Disturbance	Description	Mitigations	Discussion
Underground	Contaminant flow to surface receptors, interaction with groundwater (GW).	Plug openings, allow to flood, monitor, grout/shotcrete/backfill to limit water movement.	Will need modelling confirmed to show limited movement of GW after closure.
Surface facilities	Potential for contaminated materials during decommissioning and demolition.	Systematically survey and decontaminate as much as possible, tear down, recycle to maximum extent possible. Dispose of materials that cannot be decontaminated in TMF, remove or cover concrete pads, clean up any contaminant spills,	Per CNSC guidelines for contaminant removal. Process plant WTP will be needed until the TMF underdrain is decommissioned. Reuse or recycle to the maximum extent practicable given the location.
Roads	Remove, scarify, revegetate once no longer needed.	Survey for contamination prior to decommissioning, remove contaminated soils to TMF for disposal.	Once clean, contour to local conditions and replant with vegetation.
TMF	Will need a cover design and implementation plan to encourage ongoing dewatering and settling.	Likely scenario is an initial cover designed to weight the tailings to encourage dewatering and compaction. Once target density is achieved, complete cap/cover in final form, seal off the underdrain, and revegetate. Monitor.	Timing would have to be modelled closer to closure. Process plant water treatment facility will be required until tailings meet density target.

Table 20-6: Project PERA Summary Table – Community and Socio-Economics, FCU – PLS Property

Disturbance	Description	Mitigations	Discussion
Consultation and Community engagement	Consultation and engagement with First Nations and Métis communities. Engagement with local communities.	Engage with Indigenous communities to establish relationships, identify potential Project impacts, identify potential mitigations, and incorporate traditional knowledge. Engage and establish relationships with all potentially impacted communities. Document all activities and participants.	It is essential that this be done for the success of the project.
Roads	Increased traffic on northern roads and through towns such as Buffalo Narrows and La Loche, and north on Hwy 955.	Work with local authorities and MHI to minimize safety risks in communities. Work with MHI to improve Hwy 955 and upgrade bridges if necessary.	Will need to complete a traffic analysis to assess the impacts of increased traffic in NW Saskatchewan. Keep local mine traffic off Highway 955 to extent practicable; site camp access is via a road parallel to the highway.
Employment	A new mining operation will create jobs and opportunities for local employment.	Engage with communities to identify local training needs and ensure a trained workforce is available for construction and operation.	Engagement can provide advance notice of education and training requirements for employment.
Business opportunities	A new mine will create opportunities for business to supply goods and services.	Work with local communities and entrepreneurs to develop businesses.	Experience elsewhere in SK indicates businesses work best when they are not solely reliant on the mine(s) for their survival, given the cyclical nature of mining. This has worked well with the eastern Athabasca mines.
Impact to community services	Potential impacts on communities range from demand on health care and social services, policing, etc.	Monitor and work with local authorities to ensure service availability. Increased employment can improve community health. Continue with engagement and sponsorship activities.	Target communities are CRDN, BNDN, BRDN, and La Loche as the nearest communities followed by the west-side (Métis communities, Buffalo Narrows, Ile-a-la Crosse, and Beauval).

The Project setting is characteristic of modern northern Saskatchewan uranium mines. Based upon decades of operation, the potential impacts and mitigations are understood and well monitored. In proposing to build the mine to at least current standards for uranium mines in Saskatchewan, FCU has designed an operation that will be protective of the environment and public health and safety with overall residual impacts minimized. The main impacts are potential losses of habitat from the Project footprint, impacts to aquatic and terrestrial habitat and biota from habitat loss and Project discharges, and impacts to local users by limiting their ability to use the land within the Project footprint. Many of the potential impacts are expected to be reversible at decommissioning. The current design applies mitigations to limit impacts; however, this cannot be fully tested until the EIA stage when a detailed environmental risk assessment will be completed.

From work with local Indigenous communities, Project design was influenced by direct feedback in three areas: the mining method, Highway 955 corridor, and effluent outfall. The negative feedback on the initially proposed open pit within Patterson Lake led to FCU undertaking a second PFS using an underground-only option. As a result, the underground only option became the preferred option and the hybrid open pit/underground mining method originally proposed was abandoned. This change eliminated virtually all direct impacts to Patterson Lake.

The second relates to the original proposal to build a highway bypass around the site instead of having Highway 955 pass through the site as it does now. Local communities made it clear that they would prefer to not have any extra driving in the area and to keep the highway exactly where it is. FCU agreed to this and will employ traffic controls and signage to keep the corridor safe; highway traffic will have the right of way. Additionally, the site plan has a service road between the camp and the mine/process plant site so that local mine traffic does not have to use the highway, although there is likely to be some local traffic between the site and Big Bear Camps to the north.

The third item identified with local communities was that they did not want a treated effluent outfall, such as a large pipe visibly discharging treated water into the lake from shore. Currently, the use of a diffuser in a deeper area of Patterson Lake's south basin is proposed in the FS.

FCU is also informed by the ongoing EA of the NexGen project where Indigenous groups identified cumulative impacts as a significant concern, and more specifically the protection of woodland caribou. The other significant area of concern was water quality as water is the basis for all living things. FCU will continue to listen to the input from our rights holder communities as FCU develops its EIA. Cumulative effects will be addressed in the EIA.

The TMF design as described in Section 18.6 of this FS incorporates a pervious surround design that considers the site-specific geotechnical and hydrogeological conditions. The design is partially above ground level in consideration of the local water table and local building materials. The purpose-built TMF avoids some of the issues related to using existing pits, which often come with structural controls that may interfere with containment. The TMF utilizes the best features of modern uranium TMFs, such as:

- Engineered tails transported as a slurry in a pipeline;
- Managed tailings emplacement to prevent grain size segregation;
- Sub-aqueous deposition that eliminates tailings freezing and minimizes dust and radon emissions;
- Tailings liner system with a pervious surround layer on the inside;
- An underdrain system to accelerate densification and collect water for treatment; and
- A tailings core that will become impervious to local groundwater flows once consolidated, thereby providing long-term stability.

Several options were considered but eliminated due to issues with ice lens formation, dusting issues (especially freeze-dried tailings dusting in winter), radon emanation, and the potential for the development of acid mine drainage. The pervious surround method allows for tailings containment and a long-term stable disposal site for the tails. While the TMF drainage through the underdrain will be collected for treatment, there will be monitoring wells in place so that groundwater can be collected and tested to ensure the system is operating correctly. Some of the wells will have pumps installed such that if there is a change in groundwater it can be collected while the cause is investigated and corrected.

The initial TMF location identified in the PFS was confirmed with subsequent geotechnical work with some layout modifications to account for the existing terrain. The TMF sits within the Patterson Lake watershed for both surface and groundwater with no flow toward Alberta from the site. The impacts to Patterson Lake are predicted to be minimal. At 10,000 years the loading to the lake of an unattenuated species (e.g., chloride) was estimated at 0.8% of the source mass over that period while no break-through of attenuated species (e.g., arsenic, radium) to the water table is expected over that same period. The current data and analysis indicate that the TMF should not create any material impact to Patterson Lake or the Clearwater River over its full lifecycle.

This preliminary examination of the potential impacts from the Project is commensurate with the current level of Project design and knowledge of Project impacts. Successful environmental protection will require more work to develop a pathways model for the proposed Project that details the predicted environmental inputs, the potential receptors, and any potential impacts. This information will then be fed back to the design team to allow for optimization of mitigation strategies. All of this will be dealt with in the EIA and subsequent licencing/permitting process.

From the current assessment, there will be some locally significant impacts that, with proper design, will be reversed at decommissioning and reclamation. The underground only mining scenario goes a long way to minimizing the impacts to Patterson Lake as it eliminates the need for a ring dyke, slurry wall, dewatering, and overburden removal that was included in the initial hybrid mining PFS.

The implementation of the underground-only mining option, while subject to its own challenges, will not leave a lasting presence after decommissioning. The development of the decline and the ventilation shafts through the overburden will require careful construction. The areas within overburden development will require temporary dewatering and/or freezing for safe development. Once in place, the decline and the ventilation shafts will be largely protected from water inflow or egress.

To avoid radon buildup, two ventilation shafts are proposed, one for fresh air and one for exhaust air with the mine design allowing for single pass air wherever personnel will be present. The mine air will require constant monitoring for radon and particulates to ensure work areas remain safe. Artificial ground freezing will be used underground to provide ground support, especially in the crown pillar area. This has been used in uranium mines elsewhere in the Athabasca Basin with great success. Once the ventilation shafts are completed, the headframes will be removed, thereby minimizing the visual impact of the Project from Patterson Lake.

All mineralized, non-mineralized, acid-generating, and low-grade waste rock from underground not used in the milling of ore will eventually be returned underground as backfill to maintain the stability of the workings and to dispose of potentially problematic wastes in a safe manner. At decommissioning, there should be no surface stockpiles of potentially contaminated material remaining. In fact, it is estimated that for cemented backfill there will be a significant shortage of backfill material and non-contaminated material, such as local aggregate and sand, that will have to be blended with the waste rock generated from underground development.

Overall, the main concerns regarding underground mining and the environment will be the amount of water handled, treated, and discharged to the environment. Further work is still required to understand the site water balance under all conditions. While this will be modelled in the EIA, metallurgical test work has indicated that for all COCs, if using a diffuser to discharge treated effluent into Patterson Lake, the Project will be able to meet the discharge quality achieved at operating mines in the eastern Athabasca. These discharges are based on the use of best available methodologies and result in discharges of COCs below regulated levels. The EIA will examine this closer and assess discharges considering any potential cumulative effects from another operation discharging into the same lake.

20.4 Regulatory

20.4.1 Provincial Environmental Assessment and Permitting Process

Mineral tenure is issued by the Saskatchewan Ministry of Energy and Resources and grants mineral rights subject to conditions such as the completion of certain levels and types of assessment activities. As the Project is located on Crown land, surface access is controlled through permits from the ENV during mineral exploration. For mining, a surface lease is required prior to work commencing on site. The surface lease will generally cover all areas that are predicted to be disturbed and accrues annual fees per hectare of disturbed and undisturbed land. Within the boundaries of the surface lease, the annual payments can vary as new land is disturbed or reclaimed. Surface leases are coordinated through the Ministry of Government Relations, Northern Engagement Branch, and the ENV, Lands Branch, and includes input from other government agencies where appropriate. While negotiations can start early, and in parallel with the EIA review, a precondition of the issuance of a surface lease is the successful outcome of the provincial EA process. In Saskatchewan, the EA and licensing process are sequential, as the EA process must be completed prior to the issuance of specific licenses and permits.

FCU submitted a Technical Proposal document and a draft terms of reference (TOR) to the EASB, ENV on November 22, 2021. While the intent of the submission was to self-declare the Project as a Development per the Saskatchewan *Environmental Assessment Act* (EAA), EASB requested a letter in that regard, which FCU subsequently issued on December 2, 2021. The draft TOR were finalized in July 2022 after incorporating comments from the province, the CNSC, and Indigenous reviewers.

In order to require an EIA, a project must be deemed to be a Development per Section 2(d) of the EAA. Generally, all mining projects meet the criteria and are deemed Developments. Once a project is deemed a Development, the proponent will receive a formal Ministerial Determination that the Project is a Development and an EA is required. FCU received the Ministerial Determination on December 21, 2021. In addition to the Determination Letter, there is also a public notice about the proposed project and the notices for the Project were published in local newspapers between February 12 and 19, 2022.

Though there will not be a federal Impact Assessment (IA) of the Project, the province will work closely with the lead federal agency involved in the Project, in this case the CNSC. It was incumbent on FCU to develop a TOR for the Project that fully contemplated the federal requirements in order to have a more efficient licensing process.

Once the provincial internal reviews are finished, the EASB will compile the comments and produce the Technical Review Comments document. This document and the final EIS are placed in public review for 30 to 60 days. When all the public comments are in, EASB will produce an EIA decision document for the Minister of Environment. While there are three outcomes possible, the likely outcome for a project that gets to this stage is approval of the EIA with conditions. With approval of the EIA, licensing and permitting can be completed.

While the EIA is in progress, the proponent can develop the surface lease application, and other licensing packages for review by the government. Approval cannot occur until the EIA process is completed and a positive outcome obtained. Provincially, the licensing is through the ENV EPB, which largely provides a one window approach for licensing on behalf of other branches and ministries. Other provincial agencies that will require licenses and permits, include, but are not limited to:

- Saskatchewan Ministry of Labour Relations and Workplace Safety (occupational health and safety, mining safety/Mines Regulations, 2018)
- Saskatchewan MHI (highway access and maintenance along Highway 955)

- Saskatchewan Ministry of Health (accommodations, hygiene, water, and sewage treatment)
- Saskatchewan Water Security Agency (shoreline impacts, water supplies, treated water and sewage discharge)
- Saskatchewan Ministry of Government Relations (surface lease, monitoring, social impact requirements)
- Saskatchewan Ministry of Energy and Resources (mineral tenure, royalties)

Most ministries will indicate their interest and the need for any permits in the Technical Proposal and EIA review stages and those comments will come forward in the Technical Review Comments produced by the EASB. Overall, a number of permissions, of one form or another, are required to complete a project, but when compared to the EA process, they are rarely material to the schedule or budget if organized properly.

20.4.2 Federal Impact Assessment and Licensing

The federal *Impact Assessment Act*, 2019 (IAA) and the need to produce an IA can be triggered in two ways. The first is by triggering one of the activity thresholds in the *Physical Activities Regulations,* and the second is that the project can be designated by the federal Minister of Environment and Climate Change (the Minister) in response to a request to designate the project and a supporting recommendation from the Impact Assessment Agency. Potential triggers for the Project include the amount of material being mined or milled per day, the amount of sand and gravel being produced, or the size of the airstrip serving the site. With approximately 1,000 t/day of ore being mined and milled, the Project does not trigger Sections 20 to 23 of the *Physical Activities Regulations*. The aerodrome trigger in Section 46 of the regulations will also not be triggered because FCU is not proposing to build an airstrip. Further, while sand and gravel will be required, especially for the construction of the TMF, per Sections 18 (f) and 19 (f) of the regulations, the mine will not be producing greater than 3,500,000 m^3 per year of aggregate.

The other method of being designated for an IA under the federal IAA is to be designated by the Minister. On August 17, 2022 FCU was informed that there was a request by the Métis Nation Saskatchewan to have the Project designated. FCU provided a response to this designation request to the Impact Assessment Agency on September 2, 2022. In addition, the IAA approached several government agencies, Indigenous groups, and stakeholders for input into the request. On November 14, 2022, FCU was informed that the Project will not be designated under the IAA, and as such, will not be required to complete a federal IA. The reasoning for the Minister's decision was, in part:

"*The Agency is of the view that the potential adverse effects within federal jurisdiction would be limited and managed through project design, mitigation measures, and existing legislative frameworks.*"

The main federal licensing agency for the Project, the CNSC, will need to be satisfied that the environment is protected. The CNSC will conduct an environmental protection review (EPR) for the license application in accordance with their mandate under the *Nuclear Safety and Control Act* (NSCA) to ensure the protection of the environment and the health of persons.

The CNSC and Saskatchewan ENV have historically worked closely together and the CNSC will have the ability to review the provincial EIA submitted by FCU. The regulators have been cooperating in their review of the Project to date. For instance, the CNSC reviewed and provided comments via ENV on FCU's Technical Proposal and TOR. The CNSC will act as a technical advisor and will be an active participant in the EIA process; however, the provincial EIA decision would be independent of the federal government. To help support the CNSC's ability to accept the provincial EIA results in their EPR, FCU included federal EPR requirements in the draft TOR, including incorporating the CNSC draft TOR comments into the final document.

FCU intends to initiate the CNSC licensing process in 2023 in parallel with the EIA process. This will allow for early discussion with the CNSC on the licensing process, engagement and consultation expectations, and the scope of the Project. While the option of sequentially doing the EIA and the licensing is available to the proponent, the CNSC suggests doing these two distinct processes in parallel. Effectively, while the EIA process is proceeding, the development and submission of the provincial and CNSC licensing packages proceeds in parallel. It is assumed that a successful outcome for the provincial EIA would be an important part of the CNSC's EPR, which would be presented to the Commission Tribunal as part of the licensing reviews. As in Saskatchewan, a positive environmental decision is required prior to the Commission approving any licensing packages. The CNSC license process is done on a cost recovery basis through the *Cost Recovery Fees Regulations.*

The CNSC generally grants licenses for the four distinct stages of a project in sequence. Those licensing stages are:

- Site preparation and construction
- Operation
- Decommissioning
- Abandonment

While these stages are usually separate and sequential, there is the potential for licenses to have aspects of the other licensing stages incorporated for completeness to accommodate project conditions in a single licensing action. This approach would depend upon the proponent's ability to provide the rationale and the detailed information required to support the modified licensing package.

In support of licensing, proponents are required to develop management systems complete with policies, systems/programs, procedures, and monitoring (plan, do, act, check type-system) commensurate with the proposed scope of activities. To protect human health and the environment, the CNSC focusses on their areas of safety and control in their assessment of projects, including areas of higher risk such as quality management, occupational health and safety, environmental protection, radiation protection, tailings management, and safeguards and non-proliferation, to name a few.

CNSC certified inspectors and staff perform compliance activities for uranium mines and mills. Compliance activities include inspections, licensing and license amendment requests, review of licensee reports and performance, and environmental, radiation and conventional health and safety data analysis. The inspection reports are shared with the Northern Saskatchewan Environmental Quality Committee who in turn share the information with communities.

Periodically, the CNSC commissions independent third-party monitoring of uranium mines and mills to provide a check on the results from their own monitoring and the proponents monitoring. Summary reports of mine and process plant performance in northern Saskatchewan are published by year with 2019 being the last one available at the time of writing. The CNSC regulates uranium mines based on the NSCA and appropriate regulations with more specific requirements spelled out in regulatory documents that cover most of the safety and control areas. In addition, there are several CSA standards that are applicable to the environmental aspects of the Project.

While in-water work is expected to be limited now that the mine will be an underground only operation, there may be a need to engage with Fisheries and Oceans Canada (under the *Fisheries Act*) with regard to a treated effluent discharge diffuser. Transport Canada authorization may be required if there are any in-water works with a potential to impact navigation (under the *Canadian Navigable Waters Act*) or if the headframes for the ventilation shafts, in relation to any local airstrip, need to be registered under the *Canadian Aviation Regulations*. Water quality and the monitoring of biological effects will be governed by the MDMER to the *Fisheries Act*, in addition to provincial requirements. Other federal legislation of importance to Project will be compliance with SARA and the *Migratory Birds Convention Act*.

20.5 Water Management

Water management infrastructure has been designed to maximize diversion of surface runoff water from the general site footprint or any disturbed area where it can potentially become contaminated. Contaminated water must be captured and controlled. There are two distinct areas of contamination; those areas that have clean runoff that need to be treated just for suspended solids, and all areas where runoff may be potentially contaminated with radionuclides and other material. The latter category requires collection and treatment in the process plant prior to release to the environment. Contaminated water will include water from underground, TMF underdrain water, process plant terrace and roadway runoff, vehicle wash bays, mineralized or mine waste stockpiles, and any other collected sources of potentially contaminated water.

While the site is being designed to handle a 24-hour 1:100 year storm event, it is recognized that there is the potential for a PMP event. While some of the stockpiles and ponds will have capacity, site runoff may be more difficult to handle. As such, the ability to pump excess water to the TMF for interim storage will allow the site to handle a PMP event and the resultant runoff. Contaminated water ponds will have dual HDPE liners to provide containment with a leak detection system between the two liners.

All contaminated water will be collected for treatment in the process plant's water treatment area. From there it will be batch discharged through a diffuser system in Patterson Lake once water quality is confirmed. Test work indicates that the process plant will be able to treat water to standards comparable with other operating mines in northern Saskatchewan, which are well below regulatory levels.

Six water storage ponds are planned including three monitoring ponds for treated effluent (5,000 m^3 each), one contingency pond (5,700 m^3), one settling pond near the process plant (10,200 m^3) and one run-off collection pond (14,380 m^3). Each monitoring pond and the contingency pond is sized for 5,000 m^3 of capacity or greater and will maintain one m of freeboard as contingency for a PMP event. About 1,100 m^3 of the feed settling pond capacity is reserved for a 1:100 year 24-hour storm event which comes from rain runoff collected from selected areas.

20.6 Decommissioning

As part of the regulatory process, FCU will be required to develop a Preliminary Decommissioning Plan (PDP) for inclusion in the EIA that details the steps that will be taken to decommission Project facilities and reclaim the land at the end of Project life. As part of licensing, the PDP is fleshed out and a cost estimate for implementation is prepared: the Preliminary Decommissioning Cost (PDC). The company will then be required to provide some form of surety or bond to cover the cost of carrying out the decommissioning plan. The surety is designed to cover the unlikely situation where the proponent is unable to complete the decommissioning and reclamation and the government has to step in to complete the work in a 'decommission tomorrow' scenario. While salvage of some materials is likely, these cannot be considered in the PDC. The plan and costs are periodically reviewed and updated and can be scaled to reflect the current state of the site. As operations progress, progressive decommissioning is encouraged as it lowers close-out liabilities, which can reduce the amount of a surety bond, and often reduces the cost of disturbed-land lease fees.

For a uranium mining and milling project, once operations have stopped, the first step is to conduct systematic surveys to determine the extent of contamination, if any. Contamination may be chemical or radiological. Areas that can be decontaminated will be cleaned and re-surveyed to ensure that the clean-up criteria have been met. Material that cannot be decontaminated to release standards would be disposed of on site, likely within the TMF prior to it being capped, or at an approved off-site disposal facility. The remainder of the site will be decommissioned as the facilities are no longer required with the material salvaged for reuse or recycling, or, if that is not possible, disposal.

The TMF will require time after mine operations cease for proper consolidation of the tailings, first by removing the water cover and then by progressively adding fill over the tails. During this period, the underdrain will continue to operate to handle the pore water being squeezed from the tailings as they densify. This water will have to be treated prior to release, requiring the WTP to continue operating. Once the tailings have reached their target density, the underdrain would be disconnected and sealed, the tailings cover design finalized, the TMF contoured to promote drainage and to mimic the local landscape, and all infrastructure removed. The tailings cover will likely be an engineered cap designed to minimize infiltration, and to promote drainage and reclamation. A CNSC license will be required for the foreseeable future with periodic monitoring and repair supported by a bond or fund of some sort to support this activity. Aside from the TMF, it would be expected with demonstration of successful remediation, the remainder of the site can be returned to the province.

In Saskatchewan, reclaimed land can be returned to the Crown under *The Reclaimed Industrial Sites Act* and *The Reclaimed Industrial Sites Regulations*, which establish an Institutional Control Program. This program is implemented once a decommissioned site has been deemed to be reclaimed in a stable, self-sustaining, and non-polluting manner. The property can then be transferred back to the province for monitoring and maintenance. For this to happen, the proponent pays a calculated sum into the Institutional Control Monitoring and Maintenance Fund, and the Institutional Control Unforeseen Events Fund for long term monitoring of the property and maintenance, if required. In the unlikely event that the site does not behave as predicted, the government can seek redress from the proponent if the costs exceed the funds available.

For the underground mine decommissioning, backfilling will have used all of the waste rock stored on surface, such that there will be no remaining stockpiles of any kind. The underground mining areas should be effectively isolated from the environment with the use of cemented backfill and Patterson Lake will not be affected. Any remaining freezing will be turned off once all underground facilities are properly decommissioned and sealed and no personnel are present.

20.7 Governance

FCU is governed by a Board of Directors with most of the operational aspects of the company under the Chief Executive Officer and other senior staff. While the Board has some general policies related to the environment and social engagement, these will need to be expanded upon in order to support the policy driven management systems required to meet the CNSC's safety and control requirements, to meet corporate due diligence requirements, and to align with current ESG trends. The current policy structure is acceptable for this stage of the Project, but it will need to be significantly upgraded prior to CNSC licensing. Policies to support occupational health and safety, environmental performance, radiation protection, and ESG compliance are being developed and will be put before the Board in 2023. At that time the Board will need to have a committee(s) to oversee these policies and their implementation. That implementation would include the development of management systems predicated on the ISO-style plan, do, check, act methodology.

20.8 Discussion

The extensive baseline work has described typical northern Saskatchewan terrain of the Athabasca Basin region. It has not identified anything that should significantly delay a project if proper planning and mitigations are incorporated into the Project design. Such mitigations would include, but are not limited to, offsetting for any fish habitat disturbed by the Project, possibly terrestrial habitat offsetting for woodland caribou habitat, and sufficient consultation with local First Nations, Métis, and west side communities. The primary environmental goal will be the protection of Patterson Lake and the downstream water quality in the Clearwater River system as this will likely be the focus of any concerns under the underground mining only scenario.

Overall, the Project appears to be following applicable regulations governing exploration, drilling, and land use, and FCU staff and contractors are aware of their duties with respect to environmental and radiation protection. Early in the exploration program, there were some issues related to excess clearing of trails and access to water bodies, but FCU has worked to repair those areas and reclaim them. The operations are neat and orderly, and the level of clearing and disturbance is commensurate with similar projects in northern Saskatchewan. The Project is visited frequently by Saskatchewan Conservation Officers to ensure compliance.

A high-level PERA was done to look at potential interactions of the Project with the environment. Under the underground mining-only scenario, the main area of concern is development and operation of the TMF and the protection of surface and groundwater quality. The mitigations proposed for the TMF appear protective of the environment in the long-term, post decommissioning. The TMF will use the proven sub-aqueous deposition and pervious surround methodologies, and modelling results continue to show that the TMF as proposed will be protective of the environment in the long term. The TMF design is optimized to the existing geological and hydrogeological conditions and avoids widespread dewatering during operation. The potential for impacts on Patterson Lake will be much less in the underground mining scenario and are largely related to protecting the water quality. This will need to be demonstrated in the EIA and subsequent licensing.

Most of the remaining environmental risks are similar to those at existing uranium operations, which, in the modern era, have been demonstrated to have minimal impact on the local and regional environments with proper mitigation. Regardless, for all aspects of the Project, a detailed ERA will be required to ensure that nothing is missed and that all reasonable mitigations are included in the EIA and the Project design.

The ongoing baseline done from 2013 to 2020 was adequate to include in an EIA; however, in 2021 FCU was informed by EASB that older data may not be sufficient for the EIA. FCU commissioned CanNorth to complete an updated baseline program to refresh the data and provide continuity with the data that has been collected since 2013. This updated baseline work also addressed any gaps in previous data collection including areas now identified as part of the Project footprint that had previously not been included. A refreshed Heritage Resources Study was also part of the 2022 program. Moving forward, FCU will need to do some level of monitoring to maintain the baseline database until construction starts.

The level of environmental review was commensurate with an FS and was not an exhaustive examination of all documentation nor a compliance audit, although it did include updating the PFS modelling for potential impacts from the TMF. The interpretation relies on the authors more than 40 years of experience with Saskatchewan uranium projects, and familiarity with mining and the federal and provincial requirements that accrue to such projects. The Project is at a stage where, with proper planning, areas of concern can be addressed in a timely fashion within an orderly project approvals process.

Some of the items required to support an EIA, particularly consultation, need to be undertaken in a manner that does not materially affect Project timing. This will require ongoing engagement with the CNSC and the Saskatchewan Government to ascertain the level of First Nations, Métis, and stakeholder consultation expected as well as their expectations in other areas. With the signing of agreements related to engagement and information sharing during the EIA period with the main Indigenous groups, FCU has continued to leverage its positive relations with these groups in a respectful manner. These agreements establish the basis for FCU's ongoing relationship with these groups and sets the stage for any future accommodation agreements.

FCU's level of governance continues to be adequate for the level of work on site and the EIA regulatory period, but it will require significant improvement to support the policy-driven management systems required to support a uranium project and the CNSC's safety and control requirements. FCU will be working on this next step in 2023.

The feasibility-level engineering done to support this FS will be sufficient to support the EIA process with minor amounts of additional detail, as necessary. While it is not sufficient to support most of the licensing requirements, and that additional work will be started in 2023, the FS work provides evidence that the Project can be constructed in a manner that protects the environment and public health and safety.

21.0 CAPITAL AND OPERATING COST ESTIMATES

21.1 Capital Costs

An initial capital cost of $1.155 billion was estimated for the FS. All currencies in this section are expressed in Canadian dollars, unless otherwise stated. Where required, costs have been converted using fixed currency exchange rates of US$0.75/EUR€0.72/AUD$1.09 to C$1.00. The capital cost estimate is consistent with an AACE Class 3 estimate with the expected accuracy of ±15%. Table 21-1 presents a summary of project initial capital costs.

Table 21-1: Initial Capital Cost Summary

Major Area Description	Cost ($ million)
Mining	176
Processing	141
Infrastructure	159
TMF	235
Direct Costs	**711**
Indirect Costs	198
Owner's Costs	109
Contingency	137
Total Initial Capital Cost	**1,155**

The costs stated in Table 21-1 include only initial capital, which is defined as all costs to build the facilities that mine, transport, and process ore to produce first concentrate. Costs incurred during ramp-up of the mine and process plant in year 1, through commercial production, are included in the operating costs in Section 21.2.

The FS cost estimate was prepared with a base date of Q3/Q4 2022. The estimate does not include any escalation past this date. Vendor quotations were obtained for major equipment including vendors-provided prices, delivery lead times, spare allowances, and freight costs to project location. The quotations used in the FS cost estimate were obtained in Q3/Q4 2022.

For non-major equipment, costing is based on in-house data or quotes from other projects in the region.

All equipment and material costs include Incoterms FCA. Other costs such as spares, taxes, duties, freight, and packaging are covered separately in the cost estimate as indirect costs.

21.1.1 Mining

Within the mining initial capital costs, the significant expenditures include the portal development and decline, lateral development, ventilation raises and systems, mobile equipment, and mine infrastructure which are presented in Table 21-2. Note that costs related to ground freezing and refrigeration plant occur as Sustaining Capital, as they are incurred after the pre-production period. Any mining done by the Owner during year -1 has been designated as capitalized pre-production operating costs and is included in indirect costs.

Table 21-2: Mining Initial Capital Direct Cost Summary

Major Area Description	Cost ($ million)
Mine Development	89.0
Underground Mining Equipment	13.8
Underground Mine Infrastructure	17.6
Ventilation and Services	43.4
Mine Surface Infrastructure	11.7
Total Mining Initial Capital Direct Cost	**175.5**

21.1.2 Processing

Capital costs developed for the process plant are consistent with the process methodology described in Sections 13 and 17. Process plant costs were divided between process plant and infrastructure related to the process plant (Table 21-3).

Table 21-3: Process Plant Initial Capital Direct Cost Summary

Major Area Description	Cost ($ million)
Ore Handling and Grinding	45.2
Leaching	5.6
Liquid / Solids Separation	18.2
SX	30.8
Precipitation	12.7
Tailings Neutralization	6.3
Product Drying and Packaging	6.9
Reagents	10.6
Mobile Equipment	4.2
Total Process Plant Initial Capital Direct Cost	**140.5**

21.1.3 Infrastructure

The Project is located in a region of Saskatchewan with road access, but devoid of other infrastructure requirements, notably an electrical transmission line. A power generation option trade-off study was undertaken during the FS to determine the optimal method of providing power to the Project. Options included the construction of a high-voltage transmission line from various take-off points, and an on-site powerplant. A subsequent review of diesel power plants and LNG power plants showed that an LNG power plant is the preferred option for power generation.

In addition to the power plant, other major infrastructure expenditures include a TMF, fuel storage, site preparation, maintenance shop, administration and dry facility, effluent treatment facility, site roads, and permanent camp facility. The summary of infrastructure initial capital direct costs are shown in Table 21-4.

Table 21-4: General Infrastructure Initial Capital Direct Cost Summary

Major Area Description	Cost ($ million)
Site Preparation	3.8
Site Roads	5.5
Surface Drainage and Ponds	3.8
TMF	234.9
Effluent Treatment	34.5
Power Plant	19.8
Power Distribution	19.7
Water (Raw, Potable, Process)	7.0
Acid Plant	34.6
Admin Building and Dry	2.0
Maintenance Shop	8.7
Permanent Accommodations	15.4
Assay Laboratory	3.2
Utilities	1.5
Total General Infrastructure Initial Capital Direct Cost	**394.4**

21.1.4 Initial Capital Indirect Costs and Contingency

Initial capital indirect costs were applied to each of the respective areas of capital expenditure and are lumped into major categories including EPCM requirements, Owner's costs, pre-production operating costs, construction indirects, temporary facilities, construction power, temporary camp and buildings, freight, spare parts, first fills, commissioning, and non-recoverable taxes.

Contingencies were applied to the capital costs that are consistent with an AACE Level 3 cost estimate. Initial capital indirect costs and contingency are summarized in Table 21-5.

Table 21-5: Initial Capital Indirect Cost and Contingency

Major Area Description	Cost ($ million)
Camp and Catering	23.3
Construction Indirects	93.9
EPCM	66.7
CM Vendor Reps, Spares, First Fills and Commissioning	13.8
Owner's Costs – General	90.2
Owner's Costs – Mgmt. Reserves	19.0
Total Initial Capital Indirect Cost	306.9
Contingency	136.9

21.1.5 Sustaining Capital and Closure Costs

Capital costs that were incurred following year -1 were considered sustaining capital. This includes all capital expenditure related to ongoing mine development, the ground freezing program, mobile equipment replacements, expansions to the TMF, expansion to the power plant, sustaining capital costs related to the process plant and surface infrastructure, and an estimate of closure costs. Sustaining capital and closure costs are summarized in Table 21-6.

Table 21-6: Sustaining Capital and Closure Costs

Major Area Description	Cost ($ million)
Mine Development	258.6
TMF	42.8
Power Plant	25.1
Construction Indirects	8.3
EPCM	3.0
Contingency	46.0
Total Sustaining Capital Cost	**383.8**
Closure Cost	73.8
Total Sustaining Capital and Closure Cost	**457.6**

21.1.6 Capital Cost Exclusions

The following items are not included in the capital cost estimate:

- Force majeure
- Schedule delays, such as those caused by:

  - Major scope changes
  - Unidentified ground conditions
  - Labour disputes
  - Environmental permitting activities
  - Abnormally adverse weather conditions
- Receipt of information beyond the control of the EPCM contractors
- Salvage value for assets only used during construction
- Cost of financing (including interests incurred during construction)
- Federal sales taxes (i.e. GST/HST)
- Royalties or permitting costs, except as expressly defined
- Schedule acceleration costs
- Forward inflation
- Abnormal price fluctuations due to external events such as pandemic, geopolitical instability or interruptions in logistics, supply chain and world trade
- Cost of this study and future studies
- Escalations beyond effective date of this study
- Growth factors in design and engineering
- Uncertainties in geotechnical or hydrogeological conditions
- Sunk costs

21.2 Operating Costs

Operating costs were estimated for the FS and allocated to either mining, processing, or G&A and site services. LOM operating costs are summarized in Table 21-7.

Table 21-7: Average LOM Operating Cost Summary

Major Area Description	LOM Cost (C$ millions)	Average Annual (C$ millions)	Unit Cost (C$/t proc)	Unit Cost (C$/lb U_3O_8)
Mining	458.8	45.9	152.55	5.05
Processing	489.6	49.0	162.78	5.39
G&A and Site Services	234.9	23.5	78.12	2.59
Total[1]	**1,183.3**	**118.3**	**393.45**	**13.02**

Note: 1. Sums may not add due to rounding.

On average, approximately 325 workers are required to support the operations, including the manpower required for mining, processing, site services camp services and onsite and offsite operation support management teams. The worker headcount for mining operation varies from 76 in Year 10 to 172 in Year 2.

Mining takes place during years -1 to year 10 (note that year -1 mining costs are capitalized). Mine operating costs are summarized in Table 21-8.

21.2.1 Mining Operating Costs

Mining takes place during years -1 to year 10 (note that year -1 mining costs are capitalized). Mine operating costs are summarized in Table 21-8.

Table 21-8: LOM Mining Operating Costs

Major Area Description	LOM Cost (C$ millions)	Average Annual (C$ millions)	Unit Cost (C$/t proc)	Unit Cost (C$/lb U_3O_8)
Labour	156.5	13.0	52.04	1.73
Mine Consumables	127.1	10.6	42.27	1.40
Equipment Operations and Maintenance	62.0	5.2	20.63	0.68
Power Consumption	104.1	8.7	34.62	1.15
In-Fill Drilling	9.0	0.8	2.99	0.10
Total[1]	**458.7**	**38.2**	**152.55**	**5.06**

Note:
1. Sums may not add due to rounding.

21.2.2 Processing Operating Costs

Process operating costs are primarily composed of labour, power consumption, and consumables. Consumables consist of reagents, grinding media, mill liners, and LNG. LOM Process operating costs are summarized in Table 21-9.

Table 21-9: LOM Process Operating Costs

Major Area Description	LOM Cost (C$ millions)	Average Annual (C$ millions)	Unit Cost (C$/t proc)	Unit Cost (C$/lb U_3O_8)
Labour	99.9	10.0	33.20	1.10
Power	58.3	5.8	19.40	0.64
LNG for calciner	4.2	0.4	1.40	0.05
Reagents	282.0	28.2	93.75	3.10
Maintenance	45.2	4.5	15.03	0.50
Total[1]	**489.6**	**48.9**	**162.78**	**5.39**

Note:
1. Sums may not add due to rounding.

21.2.3 General and Administration Operating Costs

G&A costs include allowances for flights to and from the work site, camp and catering costs, insurance premiums, marketing and accounting functions, general maintenance of camp and other surface buildings, and onsite and offsite services. Additionally, allowances were made for departments of personnel that are atypical of a mine setting but are necessary for uranium mining in Canada. Allowances were made for reimbursable fees paid to the CNSC. G&A operating costs are summarized in Table 21-10.

Table 21-10: General and Administrative Operating Costs

Major Area Description	LOM Cost (C$ millions)	Average Annual (C$ millions)	Unit Cost (C$/t proc)	Unit Cost (C$/lb U_3O_8)
Labour[1]	95.1	9.5	31.62	1.05
Camp and Catering	37.7	3.8	12.54	0.41
Flights and Logistics	38.5	3.9	12.81	0.42
G&A Expenses[2]	63.6	6.4	21.16	0.70
Total[3]	**234.9**	**23.5**	**78.12**	**2.59**

Notes:
1. Includes on-site labour, off-site labour and site services labour.
2. Includes on-site expenses, off-site expenses, site services expenses and camp power.
3. Sums may not add due to rounding.

21.2.4 Power Operating Costs

The price to supply power to the Project has been calculated as $0.19/kWh. This was calculated by summing the power demand across the entire site, adding in an allowance for maintenance of the generators, and including a portion of labour to operate and maintain the plant.

21.2.5 Labour Operating Costs

Labour costs have been estimated based on comparable projects. Personnel requirements are developed based on the operational requirements, shift, equipment attendance, safety, training, and maintenance requirements. LOM average on-site staffing requirements during operation are provided in Table 21-11**.** The staffing is based on two 12-hour shifts per day on a two-week fly-in/fly-out basis.

Table 21-11: On-site Staffing Requirements During Operation

Labour Type	LOM Average
Site Administration	
Management/HR	6
Supervisors	7
Health, Safety and Security	14
Coordinators & Technicians	9
Warehousing	3
Trades	21
Labourers/Apprentices	16
Camp Management	4
Camp Services	23
Subtotal	**103**
Mine Operations	
Superintendents	4
Foremen	7
Technicians	12
Crews	88
Labourers	10
Admin	3
Subtotal	**124**
Mill Operations	
Superintendents/Managers	6
Foremen	8
Operators	30
Lab. Technicians	7
Trades	24
Labourers/Apprentices	22
Subtotal	**97**
Total	**324**

22.0 ECONOMIC ANALYSIS

22.1 Introduction

Tetra Tech prepared an economic analysis of the FS based on both pre-tax and post-tax basis in a single economic analysis model. For the 10-year LOM and 3.007 Mt Mineral Reserve, the following pre-tax economic analysis results were calculated using the metal price of US$65/lb U_3O_8 and the exchange rate of C$1.00 to US$0.75:

- 35.5% IRR
- 2.3-year payback on $1.155 billion initial capital
- $2.095 billion NPV at an 8% discount rate

FCU engaged an accredited Canadian accounting firm to review the tax component of the model for the post-tax economic analysis for this FS with the inclusion of applicable provincial royalties and corporate income taxes.

The following post-tax economic analysis results were calculated using the metal price of US$65/lb U_3O_8 and the exchange rate of C$1.00 to US$0.75:

- 27.2% IRR
- 2.6-year payback on $1.155 billion initial capital
- $1.204 billion NPV at an 8% discount rate

22.2 Pre-Tax Economic Analysis

The production schedule has been incorporated into the 100% equity economic analysis model to develop annual recovered metal production from the relationships of tonnage processed, head grades, and recoveries.

Metal revenues were calculated based on the metal recoveries and LOM concentrate production schedule presented in Section 17 and the metal pricing information presented in Section 19. Operating cost for mining, processing, site services, G&A, tailing management and handling and effluent treatment, as well as off-site charges (smelting, refining, transportation, and royalties) were deducted from the revenues to derive annual operating cash flow.

Initial and sustaining capital costs as well as closure and reclamation costs have been incorporated on an annual basis over the mine life and deducted from the operating cash flow to determine the net cash flow before taxes. Initial capital expenditures include costs accumulated prior to first production of concentrate, including all pre-production mining costs. Sustaining capital includes expenditures for mining and processing additions, replacement of equipment, and TMF expansions.

Initial and sustaining capital costs applied in the economic analysis are $1.155 billion and $0.383 billion, respectively.

Economic analysis accounts for physical reclamation costs at various times during and after the LOM.

Pre-production construction period is estimated to be three years, based on the project execution schedule presented in Section 24. NPV and IRR reported in this section are estimated at the start of this three-year period.

Pre-tax economic analysis results are summarized in Table 22-1.

Table 22-1: Summary of the Pre-Tax Economic Analysis

Description	Unit	Value
Metal Price	US$/lb U_3O_8	65
Exchange Rate	US$:C$	0.75
Undiscounted NCF	$ billion	4.508
NPV (at 8%)	$ billion	2.095
IRR	%	35.5
Payback	Years	2.3

22.3 Post-Tax Economic Analysis

At the metal price and exchange rate used for this FS, total estimated taxes and royalties payable on PLS revenues and profits over the 10-year LOM are:

- Provincial Revenue Royalties $571 million
- Provincial Profit Royalties $728 million
- Corporate Income Tax $992 million

The post-tax undiscounted annual cash flows are illustrated in Figure 19-4.

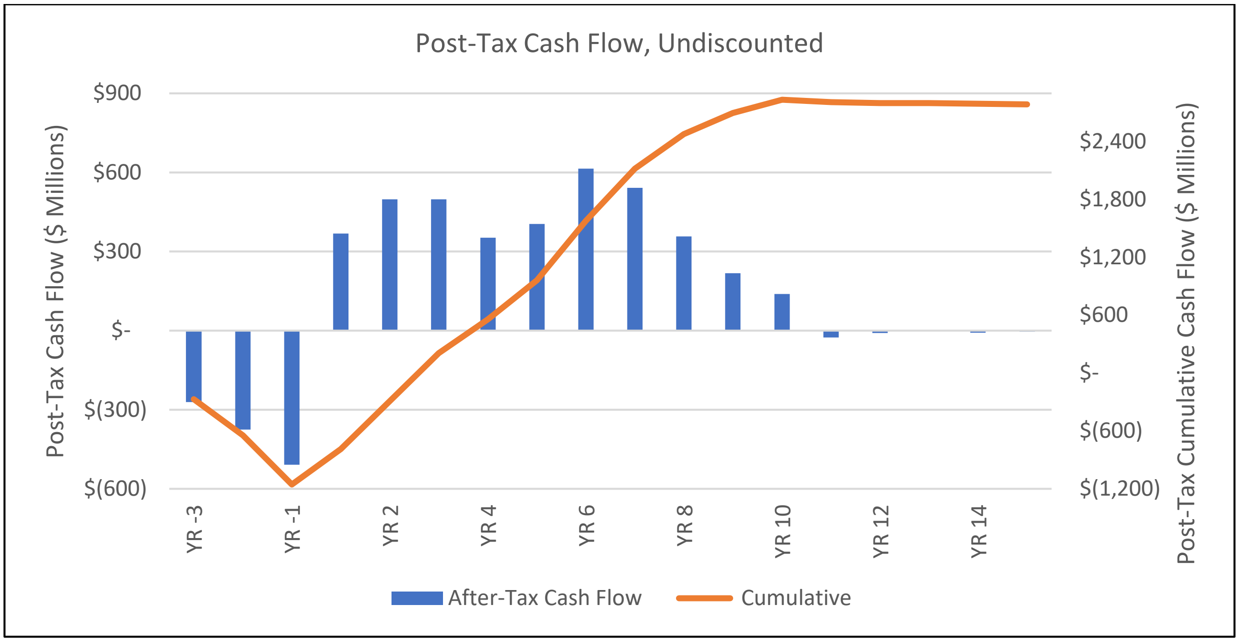


Figure 22-1: Post-tax Undiscounted Annual and Cumulative Cash Flow

Post-tax economic analysis results are summarized in Table 22-2.

Table 22-2: Summary of the Post-Tax Economic Analysis

Description	Unit	Value
Metal Price	US$/lb U_3O_8	65
Exchange Rate	US$:C$	0.75
Undiscounted NCF	$ billion	2.787
NPV (at 8%)	$ billion	1.204
IRR	%	27.2
Payback	Years	2.6

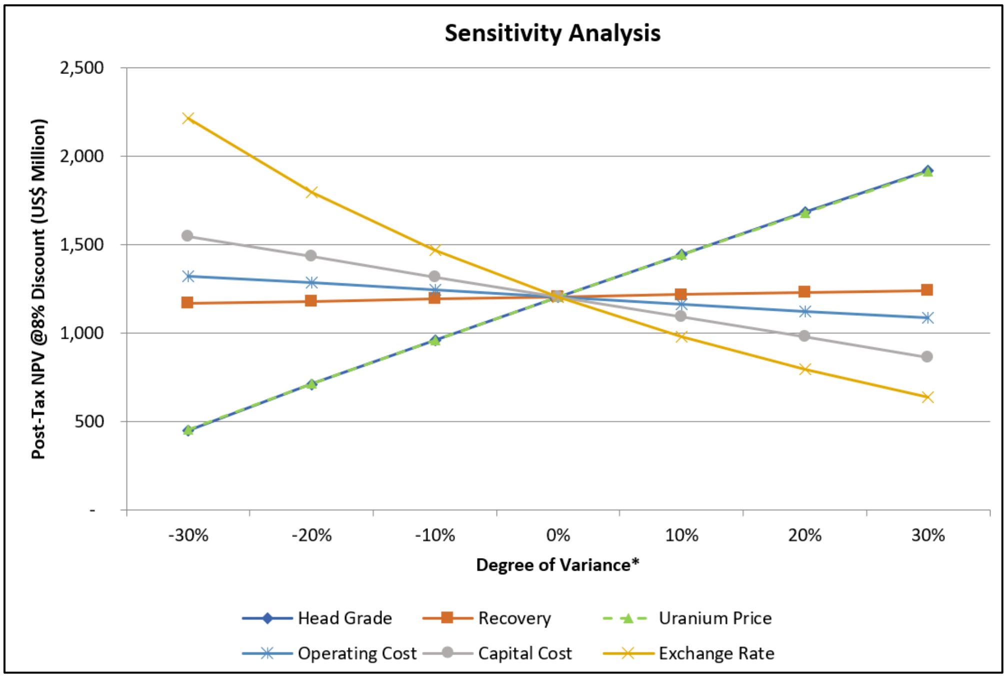
22.4 Sensitivity Analysis

Tetra Tech investigated the sensitivity of NPV, IRR, and payback period to the key variables. Each of key variables was changed between -30% and +30% in 10% increments while holding the other variables constant, except for metal recovery in %, which was changed in 0.5 increments. Sensitivity analyses were carried out on the following key variables:

- Uranium metal price (U_3O_8)
- Head Grade
- Exchange rate
- Capital costs

- Operating costs
- Metal process recovery

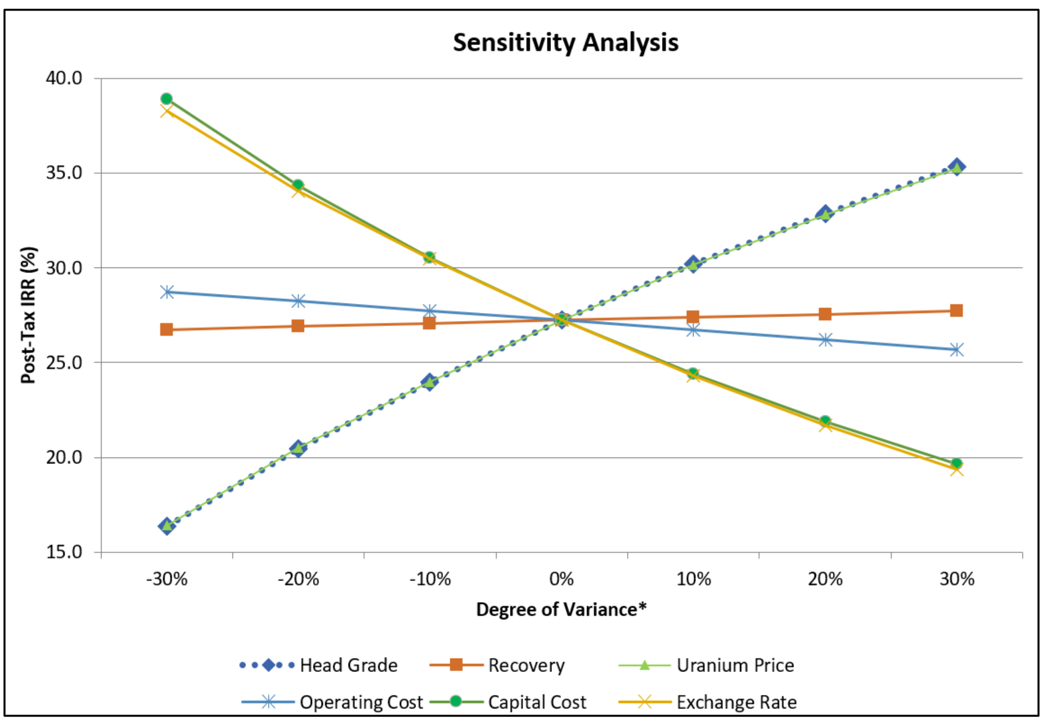
The analyses are presented graphically as financial outcomes in terms of post-tax NPV and IRR. The NPV is most sensitive to exchange rate, uranium price and head grade, followed by capital costs, operating costs, and metal recovery. The IRR is most sensitive to capital costs, exchange rate, uranium price and head grade, followed by operating costs, and metal recovery. The post-tax NPV and IRR sensitivities are presented in Figure 22-2 and Figure 22-3, respectively.



*Except for metal recovery, at 0.5% intervals between 95.5% and 98.5%

**Both head grade and metal price exhibit the same sensitivity

Figure 22-2: Post-tax Sensitivity Analysis (NPV_8)



*Except for metal recovery, at 0.5% intervals between 95.5% and 98.5%

**Both head grade and metal price exhibit the same sensitivity

Figure 22-3: Post-tax Sensitivity Analysis (IRR)

22.5 Taxes, Provincial Royalties, and Depreciation

Taxes and depreciation for the Project were modelled based on input from FCU's tax advisor.

The Project will be subject to Federal and provincial income taxes, as well as Saskatchewan royalty taxes.

To calculate income taxes, a federal rate of 15% and Saskatchewan provincial rate of 12% was applied to calculated taxable income. Taxable income consists of gross revenues less taxable deductions for the year. Taxable deductions include operating costs, reclamation costs, deductions of discretionary tax attributes (cumulative Canadian exploration expenses [CEE], cumulative Canadian development expenses [CDE], non-capital losses and capital cost allowance (CCA) on capital assets) and Saskatchewan uranium royalties.

To develop the income tax and depreciation model, all forecast capital costs were assigned to either of CDE or CCA.

In addition, FCU has opening income tax balances consisting of CEE and operating losses that were applied in the tax model. Up to 30% of the CDE balance can be applied in any given year. Generally, most mining equipment and structures that are considered depreciable fall under Class 41 of Canadian tax codes, which can be depreciated at 25% on a declining balance annually.

In Saskatchewan, multiple royalties are applied to uranium projects. Royalties generally fall into two categories: revenue royalties and profit royalties. An explanation of the various royalties is provided below:

- Resource Surcharge of 3% of net revenue (where net revenue is defined as gross revenue less transportation costs directly related to the transporting of uranium to the first point of sale).
- Basic Royalty of 5% of net revenue (as defined above), less a Saskatchewan Resource Credit of 0.75% of net revenue, for an effective royalty rate of 4.25%.
- Tiered profit royalty, with a 10% royalty rate on the first C$24.14 (indexed to inflation) profit/kg of yellowcake, followed by 15% royalty on profits exceeding C$24.14 /kg.

In the tiered profit royalty, the basic royalty and resource surcharge are not deductible for calculating profit royalties. Profits for the purposes of royalties are calculated by taking the net revenue, subtracting the full value of operating costs, capital costs, and exploration expenditures. Revenue royalties were included in the "pre-tax" cash flow results, while profit royalties are considered a tax, and are included in "post-tax" results.

Federal and provincial taxes were applied at a rate of 15% and 12%, respectively. Table 22-3 provides a summary of the taxes and royalties paid to the provincial and federal government.

Table 22-3: Summary of Taxes and Royalties

Description	Unit	Value
Provincial Revenue Royalties	$ million	571
Provincial Profit Royalties	$ million	728
Corporate Income Tax	$ million	992
Total Government Royalties and Taxes	**$ million**	**2,291**

The following assumptions are used in the tax model:

- Commercial production commences in taxation year immediately following the last year of construction.
- Capital expenditures in the cash flow model for depreciable property are included in CCA Class 41(b) or 41.2(b), depending on when the expenditures are incurred.
- Capital expenditures in the cash flow model incurred for the purpose of bringing the mine into production are included as CDE.
- No Scientific Research and Experimental Development has been modelled.
- Pre-existing tax attributes of FCU were included in the model for non-capital losses, undepreciated capital cost, cumulative Canadian exploration expense, cumulative Canadian development expense, and ITCs.
- Actual taxes payable will be affected by future corporate activities and/or material changes to tax rates or eligible deductions, both of which have not been considered.

23.0 ADJACENT PROPERTIES

The PLS Property is contiguous with claims held by various companies and individuals. As of the effective date of this Technical Report, the PLS Property is contiguous with claims registered in the names of NexGen Energy Ltd. to the east, Fission 3.0 Corp. to the south, Forum Uranium Corp. to the southwest, Dale Resources to the west, T. Young to the west and southwest, Canalaska Uranium Ltd. to the north, and a consortium consisting of Areva Resources Canada (39.5%), Cameco (39.5%), and Purepoint Uranium Group Inc. (21%) to the north and northeast (Figure 23-1).

SLR has not relied upon information from the adjacent properties in the writing of this Technical Report.

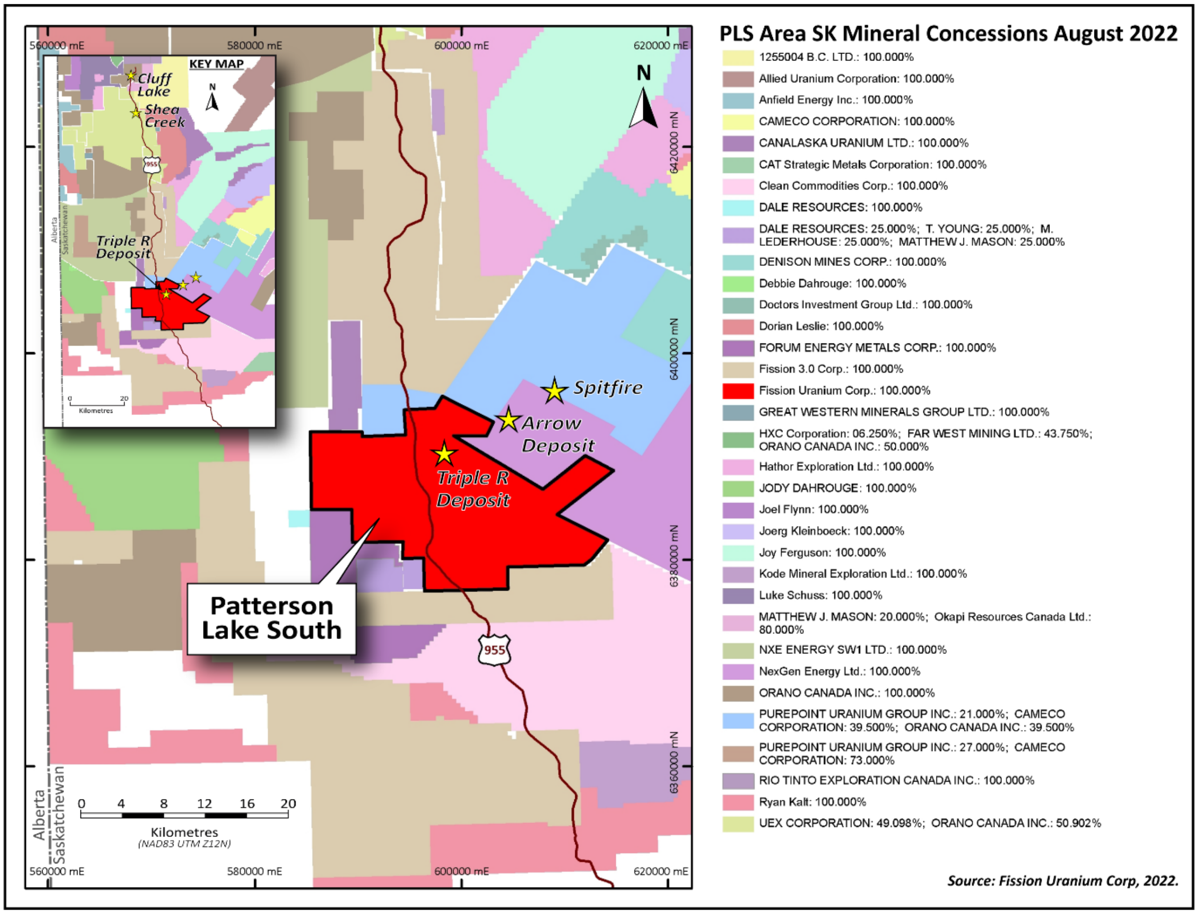


Figure 23-1: Adjacent Properties

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Execution Plan

The Project Execution Plan describes how the PLS Project could be constructed. The plan is in the preliminary stage and briefly defines the elements required to execute construction management for PLS successfully. It should be noted that the development approach will be dictated by the majority owner operatory of the Project when it goes into construction and may deviate from that described herein.

24.1.1 Introduction

The PLS Project will require three years to construct, depending on the scope of early works completed prior to full project approval. The construction scope is intended to meet the following key objectives:

- Deliver an optimized, safe, and environmentally-compliant constructed project following the systems and procedures in place
- Perform construction activities safely, striving for zero recordable incidents
- Expedite factory site and process components, preassembly, modularization, and testing to minimize site construction hours and hazards
- Maximize contracting opportunities of major scope components, for local communities, stakeholders, and Indigenous groups.
- Ensure that regulations, license agreements, applicable specifications, and standards are met
- Complete construction within the agreed schedule, not exceeding the budget, and delivering the full scope as described in the construction authorization

24.1.2 Early Works Plan

An Early Works Plan will be developed to ensure certain key infrastructure and support services are in place early during construction and functioning efficiently for a successful construction program.

The following planning and field construction focus areas must be addressed in the Early Works Plan:

24.1.2.1 Planning

- Permit review and renewal plans
- Construction procedures and develop construction work packages
- Supply chain management plan for planning, tendering, evaluating, and purchasing equipment and materials
- Staffing, recruiting, and labour relations plan
- Contracting strategy and plan for lump sum and unit rate contracts, vetted and approved by the Owner
- Health, safety, and security management plan and manual
- Logistics supply and materials management plan for early material requirements

- Site access plan
- Site-wide general services plan such as waste transfer operation, generator operation and maintenance, site garbage removal, sanitary pump out services
- Site rules and regulations plan, including site security
- Environmental and cultural sensitivity awareness training plan
- Health and hygiene program
- Site safety and security orientation program
- Natural hazards management plan
- Employee transportation plan for early construction program; air and ground planning required
- Environmental management plan to manage sediment control, waste, spills, fueling, etc., and wildlife management plan for early construction activities
- Community relations plan
- Quality management system
- Safety and emergency response plans, including content related to medical facilities and medical attention, emergency medevac, etc.
- Final development execution plan

24.1.2.2 Field Construction

- Establish explosive supply storage and controls
- Identifying and proving borrow pits
- Sourcing road materials and aggregates; setting up crushing and screening facilities
- Build roads to the Mine Site and TMF
- Establish the aggregate plant, aggregate wash plant, batch plant installation and supply of cement and aggregates
- Develop fuel supply and storage locations on-site immediately upon achievement of road access
- Establish temporary construction power – standalone power supply systems (gensets) in containers with fuel systems; build temporary water treatment/settling ponds

24.1.3 Construction Schedule

The FS construction schedule was compiled per the AACE recommended scheduling guidelines (level of detail is at Level 2), with a Class 3 definition. The construction schedule is estimated to be three years and has been designed to accommodate major seasonal and environmental constraints.

The major critical path revolves around the underground mine development and includes the installation of major infrastructure such as fresh and return air shafts, decline slurry wall construction, and dewatering system installation. After installing the decline dewatering system, six months dewatering period is envisioned before the decline development.

A preliminary construction schedule has been developed with a start date for the construction program assumed for early year -3. Contractors are to begin construction on the FAS and decline slurry wall, followed by dewatering system installation in early year -3.

Clearing, grubbing, and cutting/filling/compacting the site for early works efforts are projected to occur in early year -3. The concrete pour for the process plant is planned to occur in spring year -2, with all concrete being poured in the warm weather months of year -2, followed by erecting the building envelope to allow the construction for winter months. As buildings are closed, the mechanical, piping, HVAC, electrical, and instrumentation trades can begin construction in a staggered sequence.

The site preparation for the TMF will commence in late year -3, followed by the commencement of TMF construction to 554 masl.

The overall construction duration is estimated at 36 months. The high-level schedule is shown in Figure 24-1.

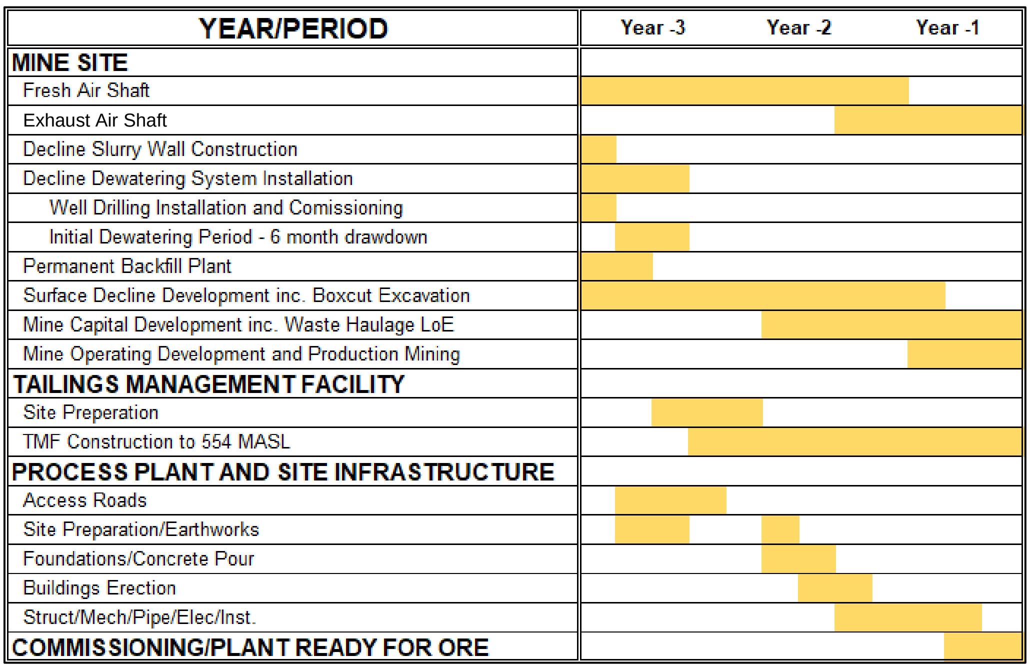


Figure 24-1: Construction Schedule Summary (Level 1)

24.1.4 Engineering and Procurement

Engineering and procurement activity will be managed by teams of professionals who will report through the EPCM contractor's directorate. The engineering team will provide the required drawings, specifications, and documents to the procurement team to purchase all equipment and materials for the construction and to allow field construction of the scope to the design intent. The EPCM contractor’s scope will include process facility and infrastructure engineering, including managing speciality contractors for major air vent shafts, slurry walls, and decline development. Mine designs will be developed and delivered by the Owner's team.

The procurement team will receive the engineering documentation, obtain multiple quotations that meet engineering specifications, and provide a purchase recommendation to the EPCM director. After the EPCM director's approval, the procurement team will purchase equipment and materials and arrange all logistics to deliver the items to the construction site ready for installation. The procurement team will also establish engineering and field construction service contracts.

24.1.5 Construction Management

The construction management team will manage all activities related to the construction management scope, including all construction activity in the mine, process, and infrastructure areas. Mining activity, environmental monitoring and reporting, and community affairs will be the accountability of the Owner's Team.

The construction management team will oversee the installation of all materials and equipment according to engineering and manufacturers' specifications and build the facilities to satisfy the design intent and be fully operable. The construction management team is also accountable for construction activity and site until handed over to the Owner following dry commissioning.

24.1.6 Construction Supervision and Contractor Management

The objective of all site construction activities is the timely and cost-effective completion of the construction facilities safely to the design intent and required standards following the schedule. While ensuring that standards are maintained, construction supervision staff will provide all oversight management to contractors in achieving this objective.

The contracts management group will fall under the responsibilities of the site procurement manager, will use an integrated data management system to track contractor invoicing, changes, and requests for information. The EPCM contractor will develop a comprehensive set of procedures in conjunction with and approved by the Owner. These procedures will outline the requirements for the execution of the administrative activities.

24.1.7 Contracting Packaging and Strategy Overview

The preliminary construction strategy includes dividing the construction into contract packages. During the contractor expression of interest and pre-qualifications phase and the advancement of detailed engineering, the contract packages will be combined to reduce the total number of contracts and form a final contracting strategy for the construction.

24.1.8 Site Organization Structure

The EPCM site organization structure has been developed to provide a balanced combination of senior managers, area managers, engineers, superintendents, and discipline specialists, to provide the Owner and contractors with continuous support during the installation period. A high-level organizational chart is provided in Figure 24-2.

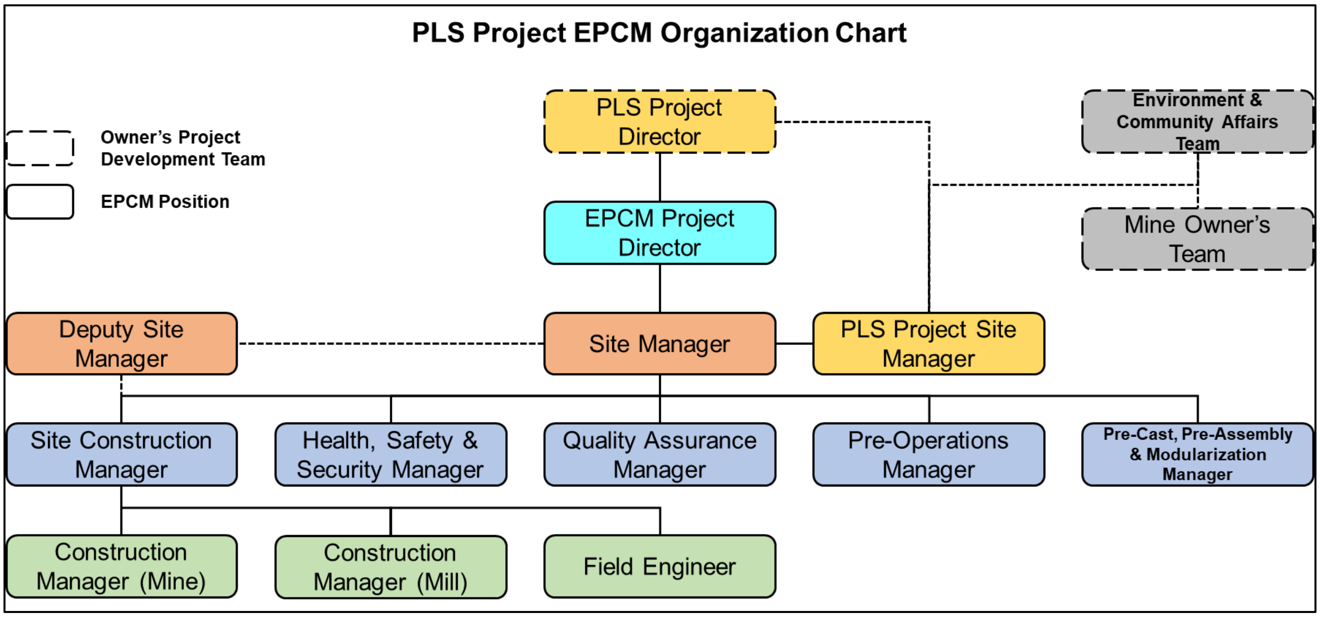


Figure 24-2: EPCM Organizational Chart

The site organization and staffing plan have been designed by work type (e.g., engineering vs. cost controls) with the geographical constraints of a construction site incorporated. The construction site will have a dedicated health and safety manager and multiple health and safety representatives to assist contractors with the daily issues and training requirements.

24.1.9 Pre-commissioning/Commissioning

The commissioning period starts in any specific work area after all materials and equipment have been installed to design specifications. The EPCM contractor certifies installation is complete and hands the area to the commissioning team. Commissioning starts after equipment or material installation for a system or work area is complete and ends when ore starts to be processed to yield a revenue stream (i.e., the battery limit between year -1 and year 1).

The EPCM contractor's scope includes commissioning through dry commissioning when work areas are handed over to the Owner's commissioning team. The Owner's commissioning team executes wet and process commissioning with select operating staff and professionals that are separate in reporting line and accountable from the operations staff. When these commissioning phases are complete, they will be handed over to Owner's operations staff, who will operate the facilities in year 1 and on to normal operation.

24.2 Owner's Implementation Plan

The Owner's implementation plan described attempts to provide a preliminary outline of the key responsibilities and actions the Owner's Team will take, including interaction with EPCM contractors during the construction stage.

An office in Saskatoon, SK, will facilitate support functions close to the construction site to provide effective support. The implementation plan described in this section highlights some key tasks required for execution by the Owner's team throughout construction. There are two initial critical tasks for the Owner's Team, starting with identifying and hiring the PLS project director, who will initially select a team and do the same for their respective teams. This process is expected to be repeated throughout construction until the entire organization has been built while directing and supporting construction in various roles.

The second initial critical Owner's team task is the engagement of an EPCM contractor early in the development schedule to drive the majority of the scope that resides outside of the Owner's direct responsibility. The type of contractual arrangement between the Owner and EPCM contractor has not been established.

The Owner will manage any early engineering work required to prepare design documents that support permit applications or renewals and compliance reports for permits issued by the Province of Saskatchewan and the Government of Canada.

A conceptual onboarding plan for specific G&A functions will be developed before the turnover of constructed areas of the site from EPCM to the Owner's team and is programmed to be well before the turnover. This will allow sufficient time for developing internal PLS Project processes and procedures to facilitate a smooth mine start-up. The early onboarding plan is intended to cover gaps in service areas that may not have been detected in this early design stage.

24.3 Health, Safety, Environmental, and Security

Health, Safety, Environmental, and Security (HSES) programs and initiatives are essential to project success. These programs will be in accordance with the conditions of the provincial EA certificates, any necessary certificates under the Saskatchewan Mines Regulations, 2018, to the Saskatchewan Employment Act, and other regulatory approvals as they are obtained. A fully integrated program will be implemented to achieve a “zero-harm” goal. To achieve this, key project stakeholders will be asked to share in the responsibility by providing the leadership and commitment to attain the highest HSES standards and values. A high level of communication, motivation and involvement will be required to develop HSES practices, including alignment with site contractors on topics such as safety training, occupational health and hygiene, hazard and risk awareness, safe systems of work, and job safety analysis. Tools will be implemented for performance tracking and accountability, including procedures for incident management.

All design and engineering stages incorporate criteria for responsible management of process flows, effluent, and waste products to meet established capture and containment guidelines. The design also incorporates basic clean plant design standards, including operational safety and maintenance access requirements. The project design team will conduct a Hazard and Operability Analysis (HAZOP) during the detailed design stage for each area of the plant. This systematic team approach will identify hazards associated with operability that require attention to eliminate undesirable consequences. Environmental protection will be incorporated in the design of the main processes of the plant, as well as in the transportation, storage, and disposal of materials within and outside the plant's boundaries.

PLS’s HSES management system will include the following elements:

- An HSES policy
- Planning, implementation, and operation
- Occupational health and hygiene
- Radiation protection
- Incident investigation

- Emergencies and contingencies
- Verification and corrective actions
- Environmental monitoring
- Review by management

PLS will provide a well-equipped first-aid facility, ambulance, and fire engine for project-wide use. The first aid facility will be staffed with medical practitioners and nurses who will be available 24 h/d to ensure continuous coverage. The first-aid staff will live at the camp. Contractors will be expected to provide basic first aid stations for their workers at the site. The main first aid station will be located by the truck shop, and satellite first aid stations will be located by the underground mine and process plant.

PLS will supply a 24-hour staffed site security program during initial field mobilization. Access to the site will be controlled at the principal road entrances at Highway 955, where the gatehouses will be constructed. All personnel required to be at the site must complete safety induction training; no personnel will be allowed at the site without this training.

All personnel required at the site will stay at the construction/operation camp. No private vehicles will be allowed on the principal mine site. Personnel will be transported by highway or air as appropriate.

The Owner will develop HSES policies and procedures that, at a minimum, address the following requirements for all future stages of the project and operation:

- Permit to work
- Ground disturbance
- Driving
- Confined spaces
- Hot work
- Working at height
- Lifting operations
- Radiation protection
- Energy isolations
- Pressure testing
- Plant and equipment
- Housekeeping
- Machine guarding
- Employee health

24.4 Risk and Mitigation

Risks are inherent to any major project, and early risk identification allows mitigating strategies to be devised and resources to be allocated for their implementation, thus enhancing the Project's benefits.

The FS project team conducted a detailed risk assessment using an industry-standard assessment methodology. A series of potential project risks were identified and assessed according to the type of risk, the effectiveness of mitigation controls, consequences, level of exposure to the risk, and probability of occurrence. The risks identified are in addition to general risks associated with mining projects, including, but not limited to:

- General business, social, economic, political, regulatory, and competitive uncertainties
- Changes in project parameters as development plans are refined
- Changes in labour costs or other costs of production
- Adverse fluctuations in commodity prices
- Failure to comply with laws and regulations or other regulatory requirements
- The inability to retain key management employees and shortages of skilled personnel and contractors.

The following definitions have been employed in assigning risk probability factors to the various aspects and components of the Project:

1. Rare: The risk is very unlikely to occur during the Project life.
2. Unlikely: The risk is more likely not to occur than occur during the Project life.
3. Possible: There is an increased probability of the risk occurring during the Project life.
4. Likely: The risk is likely to occur during the Project life.
5. Almost Certain: The risk is expected to occur during the Project life.

The following definitions have been employed in assigning risk consequence factors to the various aspects and components of the Project:

1. Low: Risks that are average or typical for a deposit of this nature and could have a relatively insignificant impact on the economics. These generally can be mitigated by normal management processes combined with minor cost adjustments or schedule allowances.
2. Minor: Risks that have a measurable impact on the estimated quality but are insufficient to impact the economics significantly. These generally can be mitigated by normal management processes combined with minor cost adjustments or schedule allowances.
3. Moderate: Risks that are average or typical for a deposit of this nature but could significantly impact the economics. These risks are generally recognizable and, through good planning and technical practices, can be minimized so that the impact on the deposit or its economics is manageable.
4. Major: Risks that have a definite, significant, and measurable economic impact. This may include basic errors or substandard quality based on estimate studies or project definitions. These risks can be mitigated through further study and expenditure that may be significant.

5. High: Risks that are largely uncontrollable, unpredictable, unusual, or are considered not to be typical for a deposit of a particular type. Good technical practices and quality planning are no guarantee of successful exploitation. These risks can significantly impact the deposit's economics, including significant schedule disruption, significant cost increases, and degradation of physical performance. These risks cannot likely be mitigated through further study or expenditure. Environmental/social non-compliance, particularly regarding Equator Principles and International Finance Corporation Performance Standards, may be included in this category.

The risk items were ranked according to the consequence of the event and the likelihood of occurrence, as indicated in Figure 24-3. The top ten risks are highlighted in red in the figure.

Probability/ Likelihood					
Almost Certain					
Likely			**1**		
Possible			2, **7**, **8**, 9, **10**, **11**, 14, **15**, **16**, **17**, 18, 21, 22, 23, 24, 25, 26		
Unlikely			27	13	**12**
Rare					3, 4, **5**, 6, 19, 20
	Low	**Minor**	**Moderate**	**Major**	**High**
	Consequence / Impact				

Figure 24-3: Risk Consequence versus Likelihood Matrix

The risks identified for the Project are summarized in Table 24-1. The risks listed are typical for mine development projects. No fatal flaws have been identified.

Table 24-1: Summary of Major Risks and Mitigation Strategies

Risk Number	Project Element	Description	Project Impact	Likelihood	Consequence	Mitigation
1	Environmental	Forest fires / natural disasters	A forest fire and natural disaster pose risks to site and personnel	Likely	Moderate	▪ Work with Saskatchewan public safety to develop a smart fire plan for the site ▪ A seismic review was completed, including decline/ramp access, and was also addressed in the underground geotechnical report
2	Environmental	Failure to meet direct contact water quality	Risk of getting notices of non-compliance, fines, suspending operations or getting shut down. Escalating consequences	Possible	Moderate	▪ Monitoring ponds to be constructed. Robust effluent WTP. Recommendation to use a diffuser
3	Environmental	Failure of the TMF containment structures/liners	Risk of groundwater contamination	Rare	High	▪ Monitoring sentinel wells ▪ Effective design and optimization
4	Environmental	Sewage	Sewage going to Patterson Lake	Rare	High	▪ Rotating biological contactor unit is planned ▪ Sludge will go to TMF ▪ A septic pond is planned
5	Permitting	Final acceptance of EIS and permitting is not achieved	Project delays in permission to construct	Rare	High	▪ Continue with the regular community and regulatory consultation meetings ▪ Provincial regulations have been followed, and appropriate documents have been submitted ▪ Draft EIS to be submitted ▪ Consultation agreements with all of the identified First Nations impacted and Metis Nations
6	Permitting	The surface lease agreement is not approved.	Project delays in permission to construct.	Rare	High	▪ Complete in parallel with the EA process
7	Mining	Resource tonnage and/or metal grade are over-estimated	Loss of project financial KPIs	Possible	Moderate	▪ Infill drilling is required in areas classified as indicated or inferred ▪ There is upside potential to increase resources along strike at depth. Allowance for infill and exploration drilling included in plans ▪ Consider drilling from the surface for measured resources (first two years or payback period) and scissored holes
8	Mining	Overburden characteristics and impact on the decline and shaft development method	Project delays in construction	Possible	Moderate	▪ Continue investigation on development methodologies ▪ Early engagement of contractors and suppliers (decline-specific) ▪ Feasibility confirmation from third-party contractors (RORCON)
9	Mining	Ground conditions within the mineralized rock cause unmanageable ground conditions (MSZ)	Project delays in construction.	Possible	Moderate	▪ The design adheres to BGC geotechnical report recommendations
10	Mining	Unanticipated poor ground conditions are encountered that affect the location and design of long-term underground excavations, mine development rates, and stoping design parameters	Project delays in construction. Project financial KPIs are not met	Possible	Moderate	▪ Continue to conduct geotechnical drilling before and during construction and underground geotechnical drilling during mine development ▪ Support requirements are adjusted to meet conditions encountered ▪ Upgrade the structural geology model ▪ The design adheres to BGC geotechnical report recommendations
11	Mining	Production rate not meeting process plant capacity	Loss of project financial KPIs	Possible	Moderate	▪ Mine sequence incorporates optionality ▪ Flexibility on various mining fronts is available ▪ t per vertical m are well within the benchmark range ▪ Critical production equipment has spare capacity

table continues…

Risk Number	Project Element	Description	Project Impact	Likelihood	Consequence	Mitigation
12	Mining	Underground flooding	Project delays in construction. Project financial KPIs are not met	Unlikely	High	▪ Ensure spare pumping capacity is available (especially during the construction phase) ▪ Predevelopment grouting and probing before entering an area ▪ Bulk freeze wall planned in high-risk areas ▪ Testing of permeability of rock with two additional pumping wells in winter
13	Mining	An uncontrolled inflow of CHF	Risk of failure of containment.	Unlikely	Major	▪ Staged filling, barricades, rock band ▪ Free draining system
14	Mining	Ensuring higher ground support selection/design is adequate	Increased costs	Possible	Moderate	▪ Proper engineering design and ongoing assessment of rock conditions ▪ Ground Control Management plan ▪ Have ground support and supply contingency ▪ Ensure excavation control of mine control intersection and stopes sizes ▪ Ensure proper mine development contractor plan
15	Mining	Ventilation	Inadequate ventilation for personnel	Possible	Moderate	▪ Ensure adequate single-pass ventilation in all areas with personnel ▪ Stench gas system for personnel evacuation ▪ A failure mode system is in place if fans go down ▪ Radiation monitoring underground ▪ The primary ventilation system is under negative pressure
16	Process	Throughput not realized	Project financial KPIs are not met	Possible	Moderate	▪ Proper engineering design ▪ Equipment redundancy
17	Process	Uranium recovery of the process plant does not meet expectations	Project financial KPIs are not met	Possible	Moderate	▪ Engineering design is supported by test work ▪ Design factor and additional surge capacity
18	Process	Hazardous material	Exposure to corrosive and toxic chemicals above administrative or regulatory levels; radiological exposure in certain parts of the mill or cleaning circuits.	Possible	Moderate	▪ Containment areas, secondary containment ▪ Separate fire suppression system for SX plant ▪ Explosion-proof equipment and circuits as needed ▪ Automated packaging system for yellowcake ▪ Adequate HVAC system to be installed
19	Water Management	Water permit penalty changing discharge requirements	No major concerns.	Rare	High	▪ Consider the location of the potable water draw ▪ Reuse of process and reclaim water to minimize freshwater usage
20	Infrastructure	Aviation	Plane goes down / loss of staff	Rare	High	▪ Ensure the airline complies with license standards
21	Health & Safety	Working underground	Project delays in construction. Project financial KPIs are not met	Possible	Moderate	▪ Procedures, PPE, following standards/regulations, proper training ▪ Elimination/Minimization by design ▪ Health and safety management system
22	Health & Safety	Confined space		Possible	Moderate	
23	Health & Safety	Underground mobile equipment (remote)		Possible	Moderate	
24	Health & Safety	Crane and Lifts		Possible	Moderate	
25	Health & Safety	Isolation of stored energy		Possible	Moderate	
26	Health & Safety	Electrical isolation		Possible	Moderate	
27	Reputation	Poor safety and environmental performance	Project delays in construction. Project financial KPIs are not met	Unlikely	Moderate	▪ Follow regulations and procedures ▪ Regular community updates ▪ Ensure critical management procedures

25.0 INTERPRETATION AND CONCLUSIONS

25.1 Geology and Mineral Resources

The Triple R deposit is considered an example of a basement-hosted vein-type or fracture-filled uranium deposit. Mineralization is known to occur at five locations on the PLS Property, from west to east: 1) R1515W, 2) R840W, 3) R00E, 4) R780E, and 5) R1620E, the most significant of which is the R780E zone. To date, FCU and its predecessors have completed a total of 844 drill holes, totalling 227,775 m across the PLS Property. Drilling includes exploration, geotechnical, metallurgical, water wells, and hydrogeology drill holes. The core from the first drilling programs was stored at the Big Bear Lodge on Grygar Lake, but since August 2013, all the core has been stored at a purpose-built storage facility located west of Patterson Lake. In the SLR QP's opinion, the logging and sampling procedures meet or exceed industry standards and are adequate for Mineral Resource estimation. SLR reviewed and verified the resource database, including a review of the QA/QC methods and results, verifying assay certificates against the database assay table, standard database validation tests, and three site visits, including a drill core review. No limitations were placed on SLR's data verification process.

Based on the data validation and the results of the standard, blank, and duplicate analyses, the SLR QP is of the opinion that the sampling methods, chain of custody procedures, and analytical techniques are appropriate and meet acceptable industry standards. The assay and bulk density databases are of sufficient quality for Mineral Resource estimation at the Triple R deposit.

The Mineral Resources are based on a US$50/lb uranium price at a COG of 0.25% U_3O_8 and a potential underground scenario. Indicated Mineral Resources total 2.69 Mt at an average grade of 1.94% U_3O_8 for a total of 114.9 Mlb U_3O_8. Inferred Mineral Resources total 0.64 Mt at an average grade of 1.10% U_3O_8 for a total of 15.4 Mlb U_3O_8. Gold grades were also estimated and averaged 0.61 g/t for the Indicated Mineral Resources and 0.44 g/t for the Inferred Mineral Resources. Mineral Resources are inclusive of Mineral Reserves. The SLR QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the current Mineral Resource estimate.

25.2 Metallurgy and Process

A series of bench scale and bulk tests were conducted at SGS Lakefield to support the feasibility level design of the process plant. High uranium extractions were achieved in a 12-hour leach, averaging 98.4% for all the tests, regardless of composite type, leach solid density, feed grind size, head grade, oxidant type, oxidation potential and free acid levels. The CCD simulation showed that a six-stage thickener circuit would achieve a 99.5% wash efficiency. The SX pilot plant test showed 99.9% uranium recovery in the extraction stage and 99.4% stripping efficiency. Yellowcake precipitation using hydrogen peroxide and magnesia for pH control produced products averaging 80% U_3O_8, within refinery specifications. The uranium grade in the calcined yellowcake product was 95% U_3O_8 at a calcination temperature of 450°C. The effluent treatment test yielded a treated effluent meeting MDMER guidelines.

Tetra Tech completed the process plant design using conventional and proven uranium extraction processes and equipment and has drawn on its knowledge of other Athabasca uranium plants. The processing plant has been designed to process the ore at a nominal throughput of 1,000 t/d to produce market-grade uranium concentrate.

The ore will be ground in an SSAG grinding circuit to 80% passing 150 µm and then leached using sulphuric acid and hydrogen peroxide at 50°C. The leached slurry will be fed to a counter-current decantation and clarification circuit to produce the pregnant leach solution. An SX circuit will purify and concentrate uranium in the pregnant leach solution for yellowcake precipitation. Hydrogen peroxide and MgO will be used for yellowcake precipitation. The precipitated yellowcake will be calcined at 450°C before packaging in drums for shipment to the customers. The produced tailings will be neutralized and then deposited in the TMF. The effluent and contact water will be reused in the process plant. Any excess water will be treated, monitored, and sampled to meet the applicable environmental standards and regulations before discharge.

The process plant will treat approximately 3 Mt ore over the LOM and produce 90.9 Mlb yellowcake concentrate with 95% U_3O_8.

25.3 Mining

The Triple R de posit is contained primarily within metamorphosed basement lithologies and, to a much lesser extent, within overlying Meadow Lake Formation sedimentary rocks. Bedrock is overlain by 50 m to 100 m of sandy overburden, with high grade mineralization located near the bedrock-overburden contact. Although the bedrock is generally competent, rock strengths in the mineralization have been degraded by radiological alteration. The deposit extends under Patterson Lake, and a key technical challenge to developing the operation will be water control related to Patterson Lake and saturated sandy overburden.

The FS mine plan has been designed around the R780E, R00E and R840W orebodies. The underground mining method will be primarily longhole retreat stoping using longitudinal methods based on the current geotechnical analysis and orientation of mineralization. Drift and fill mining will be used within the crown pillar zones of the deposit. Where the mineralization thickness allows, mining will progress across the orebody from the hanging wall drive to footwall drive. In the longitudinal areas of mining, the lenses will be mined from the southwest to the northeast in a bottom-up sequence except for one zone in the later part of the mine life which will be mined underneath a sill pillar. Mining is planned at a nominal rate of 1,000 tpd ore.

The mine will be accessed using a decline originating to the south of the R840W deposit, in close proximity to the mill stockpile and waste stockpile areas. The decline will include a box cut into the overburden and a portal face collared into the overburden headwall. The first stage of the decline will be developed through overburden geological units for approximately 350 m on a gradient of minus 15 %. The decline is planned to be excavated using a tunnel shield method with segmented concrete liner which will require the length of the decline to be dewatered to below the tunnel horizon. Following this, the decline will transition through a section of weak sedimentary bedrock for 65 m before it reaches competent bedrock. Ventilation will be via a 6.5 m diameter FAS and a 5.0 m EAS. Secondary personnel egress will also be installed in the FAS using a self contained modular ladderway affixed to the shaft liner.

A key component of the underground design is the concept of using artificial ground freezing to extract a portion of the crown pillar – the mineralized material that approaches the overburden layer. The crown pillar areas will be partially recovered using bulk artificial ground freezing techniques and an overhand drift and fill mining method. The artificial ground freezing reticulation is planned to be installed from an offset underground drift below the crown pillar areas in both the R00E and R780E zones. The refrigeration plants will be constructed on surface and will pump refrigerated brine solution underground via the FAS to the crown pillar areas.

An updated estimate of Mineral Reserves for the Project based on the underground mine plan that captures the majority of the High-Grade Indicated Resources in the R780E, R840W and R00E zones showed a 3.01 Mt of probable reserves at 1.41% U_3O_8 grade, equating to 93.7M lb of contained U_3O_8. The Project will have a three-year construction period, followed by ten years of mining. Mineral Reserves are estimated using an average long-term uranium price of US$65/lb U_3O_8 and an exchange rate of C$1.00/US$0.75. The QP is not aware of any mining, metallurgical, infrastructure, permitting, or other relevant factors that could materially affect the Mineral Reserves estimate.

25.4 Infrastructure

The Project will require the development of several infrastructure items. Project infrastructure will include:

- Fresh and exhaust air ventilation shafts, a decline for ore transport from underground to the surface, a freeze plant, dewatering wells, a backfill plant and an intermediate settling/polishing pond
- Process facilities including ore stockpile, process plant, SX plant, acid plant, effluent treatment facility, surface run-off and monitoring ponds, and assay laboratory
- A TMF to safely manage the tailings and water associated with mill feed processing, tailings transport and disposition systems and a reclaim water system
- On site connective access roads among site infrastructure and Highway 955 with site access controls
- Truck shop, machine shop and warehouse, fuel storage and fuel farm
- LNG storage, power plant and distribution system
- WRMF
- Accommodation and administration offices
- Communications infrastructure

25.5 Environmental, Permitting and Social Considerations

The extensive baseline work has described the typical northern Saskatchewan terrain of the Athabasca Basin region. It has not identified anything that should significantly delay a project if proper planning and mitigations are incorporated into the Project design. The primary environmental goal will be the protection of Patterson Lake and the downstream water quality in the Clearwater River system, as this will likely be the focus of any concerns under the underground mining-only scenario. Overall, the Project appears to be following applicable regulations governing exploration, drilling and land use, and FCU staff and contractors are aware of their duties to environmental and radiation protection.

At a high level, PERA was done to look at potential interactions of the project with the environment. Under the underground mining-only scenario, the main area of concern is the development and operation of the TMF and the protection of surface and groundwater quality. The mitigations proposed for the TMF appear protective of the environment in the long-term, post-decommissioning. Most of the remaining environmental risks are similar to those at existing uranium operations, which, in the modern era, have been demonstrated to have minimal impact on the local and regional environments with proper mitigation.

The level of environmental review was commensurate with an FS and was not an exhaustive examination of all documentation nor a compliance audit, although it did include updating the PFS modelling for potential impacts from the TMF. The feasibility level engineering done to support this FS will be sufficient to support the EIA process with minor amounts of additional detail, as necessary. While it is not sufficient to support most of the licensing requirements, and that additional work will be started in 2023, the FS work provides evidence that the Project can be constructed in a manner that protects the environment and public health and safety.

25.6 Capital and Operating Costs

The total estimated initial and sustaining capital cost for the design, construction, installation, and commissioning of the Project is $1,539 million. This estimate has been prepared in accordance with the AACE Class 3 cost estimate standards with the expected accuracy of ±15%. The average operating cost is estimated at $393/t ore processed, or $13.02/lb U_3O_8 produced.

25.7 Economics

Economic analysis of the FS demonstrates that the Project is economically viable using the stated price assumptions, cost estimates and technical parameters generated by the FS. Using an annual discount rate of 8%, the Project base case post-tax cash flow evaluates to an NPV of $1.204 billion and an IRR of 27.2%. The post-tax payback period is 2.6 years when discounted at 8% per year based on the metal price of US$65/lb U_3O_8 and the exchange rate of C$1.00 to US$0.75.

26.0 RECOMMENDATIONS

This section presents the recommendations for the Project. Overall, it is recommended that the Project be advanced to the next stage.

The QPs make the following recommendations to advance the Project:

26.1 Geology and Mineral Resources

- The R1620E zone was last drilled during the winter 2017 program; additional drilling is recommended once operations are underway. FCU estimates approximately 20 holes would be required to fully delineate the R1620E zone.
- The R840W zone is currently defined by 91 drill holes with a total grid east-west strike length of 425 m. Additional drilling is recommended. This should be done as an underground drill program once operations are underway, via a dedicated exploration drift.
- The R1515W zone was last drilled during the winter 2018 program; additional drilling is recommended, which is to be drilled from underground once operations are established.
- SLR recommends resuming the use of a secondary laboratory check with any future exploration drilling. Umpire laboratory duplicates have never failed QA/QC and triggered a repeat analysis at either laboratory.
- A number of other wireframe solids make up a smaller portion of the Mineral Resources. Most of the secondary domains are oriented similarly to the MZ. Some, including R00E, were modelled with a horizontal orientation. Additional drilling is recommended to better define the geometry of mineralization. This should be done as an underground drill program once operations are underway.

26.2 Geotechnical and Hydrogeology

- Geotechnical drill holes to collect additional information should be used to confirm the geological interpretations and the geotechnical parameters of the rock mass and discontinuities. The drilling program should include packer testing above, below, and across/within faults or geologic contacts.
- Televiewer surveying should be conducted in all drill holes to collect oriented discontinuity data and to assist with the identification and interpretation of large-scale structures, and piezometers should be installed to monitor groundwater pressures both before and during mining. Additional laboratory testing should be completed, including UCS, BTS, triaxial, frozen creep, and frozen UCS testing.
- The current structural model developed by BGC has been developed through a desktop study of available geotechnical and geological data. It is recommended that the high confidence faults with the potential to impact major infrastructure be targeted in future drilling campaigns.
- The strength and quality of the Resource geotechnical unit is highly variable. The preliminary variability assessment conducted as part of the current study should be updated as additional data is collected during detailed design and as mining commences, and ground performance can be assessed.
- The mine plan should account for the potential for open historical drill holes, particularly those collared on Patterson Lake, because they present an inflow risk. BGC recommended that FCU compile a “vertical opening register” that summarizes the as-built and completion data for all historical drill holes and is kept updated as additional drilling and other vertical development occurs.

- BGC recommended that the crown pillar design assumptions be confirmed using a probe and grout program prior to excavation within 15 m of the inferred basement bedrock contact. Accurate and precise confirmation of the basement bedrock and sedimentary bedrock units are critical to crown pillar stability, and a poor understanding of those contacts increases the potential for crown pillar instability.
- Given the relatively high predicted unmitigated seepage rates to the underground, and the sensitivity of the analyses to the bedrock hydraulic conductivity derived from the pumping tests, BGC recommends that FCU complete two additional pumping tests in the location of the proposed mine under Patterson Lake. Ideally, one pumping test would be conducted in each of the hanging wall and footwall of the deposit, and two observation VWP nests would be installed for each test.
- As further hydraulic testing is complete and additional data is collected, the numerical groundwater flow model developed for the decline and underground hydrogeological assessment should be re-evaluated. This should include updating the steady state and transient calibrations of the model and revising model predictions.
- Opportunities exist to further optimize the current mine plan to provide additional reduction in mine seepage rates. This could include:
  - sequential mining of individual ore bodies followed by hydraulic isolation of mined-out areas using tight backfill and strategically placed bulkheads,
  - accelerated mining in areas where higher seepage is expected, followed up by hydraulic isolation, and
  - sequencing development of accesses and drifts so it coincides with stope development.
- It is also recommended that historical records for any exploration programs in the area by others, together with FCU's borehole abandonment records, be reviewed and results incorporated into future mine planning.

26.3 Mineral Reserves and Mining

- An analysis of the planned underground stope shapes has been performed and shows a net dilution of 24% for the PLS project. The QP recommends that a detailed dilution study be conducted that further defines the expected performance of the various ore zones, given the large range in ore strengths as part of future studies.
- MineFill recommends the following future work:
  - Additional test work on the hydraulic fills to optimize the designs. The additional testing should include rheology, additional percolation tests, and uniaxial compression tests over a range of solids contents.
  - Additional flow modelling to optimize the CHF reticulation to stopes
- An analysis of the technical and economical viabilities of substituting the diesel vehicles with battery electric vehicles.

26.4 Metallurgy & Process

The QP recommends further metallurgical testing on samples from the PLS deposit focused on accessing cost-effective alternative methods that could reduce the overall footprint and project cost without hampering the overall uranium recovery.

- The solid-liquid separation of the leached slurry was conducted for conventional thickeners. A test program for high-rate, high-density and/or high-compression thickeners is recommended to help reduce the thickener size and overall CCD circuit footprint.

- Alternative methods of solid-liquid separation of leached slurry, such as pressure and vacuum filtration, should be tested. Belt filters instead of thickeners have shown wide acceptance in similar operations.
- The use of Polyox 301 for silica removal from the pregnant leach solution should be further explored and optimized.
- The test results indicated that nanofiltration can provide the potential for recovering 50% of the sulphuric acid in the pregnant strip liquor for recycling to the process but would accompany a 15% recycling of uranium silica from the solution. The application of nanofiltration for acid recovery should be further explored and optimized.
- Alternative clarification methods, such as inclined plate settling and/or dynamic/pin bed clarification, should be tested to help reduce the plant footprint and have shown applications in Southern Africa.
- Given the presence of 0.3 g/t gold in the leach residue, the potential to add a gold recovery circuit into the existing design should be evaluated if the market forces change.

26.5 Infrastructure

- Perform a logistics study for the Project. Emphasis should be placed on the three traffic bridges on route to the site to define the allowable load sizes and weights that the bridges can accommodate.
- Further geotechnical investigation of proposed locations of FS surface infrastructure facilities.
- Investigation and possible adaptation of newest building technologies for enhancing the energy efficiency of buildings and mechanical equipment.

26.6 Environmental, Licensing and Permitting

- Proceed to develop an EIA consistent with the ToR and input from engagement activities that supports the provincial and federal requirements.
- Initiate the CNSC licensing process in 2023 in parallel with the EIA process to avoid undue delays to the proposed schedule.
- Continue the engagement and consultation program already underway as the information from the engagement agreements is critical for a successful EIA.
- Carry out a detailed ERA to ensure that all reasonable mitigations are included in the EIA and incorporated in the Project design.
- Continue bio-physical monitoring to maintain the currency of the existing environmental database.
- Continue to explore options to reduce the environmental footprint of the Project. Examine sustainability options in the EIA..
- Explore shared services options with other companies operating in the area (e.g., environmental data sharing, infrastructure, etc.) in order to reduce impacts.
- Continue to participate in the woodland caribou discussions for two zones in Saskatchewan: SK1, the Boreal Shield (which includes the Athabasca Basin), and SK2W, the Boreal Plain, which includes the Site.
- Ensure that future work on site is of sufficient detail to support the EIA and CNSC licensing process.

26.7 Cost Estimate for FS Recommendations

The following budget is proposed for work carrying through to the completion of front-end engineering design, including completing an EA and licensing process (Table 26-1).

Table 26-1: Cost Estimate for FS Recommendations

Item	Budget ($ millions)
Geology Mineral Resource	4.5
Geotechnical Studies	7.1
Metallurgical Studies	1.0
Underground Mining – Geotechnical Drilling	1.7
Underground Mining – Hydrogeological Drilling	0.8
Underground Mining – CHF Testwork	0.1
Front-End Engineering Design	9.8
Exploration Drilling	24.0
Social Permitting	3.5
EA and Licensing	20.0
Total	**61.7**

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28.0 QP CERTIFICATES

CERTIFICATE OF QUALIFIED PERSON

I, Hassan Ghaffari, M.A.Sc., P.Eng., do hereby certify:

- I am a Director of Metallurgy with Tetra Tech Inc. with a business address at Suite 1000, 10th Floor, 885 Dunsmuir Street, Vancouver, BC, V6C 1N5.
- This certificate applies to the technical report entitled “Feasibility Study, NI 43-101 Technical Report, for the PLS Property”, with an effective date of January 17, 2023 (the “Technical Report”).
- I am a graduate of the University of Tehran (M.A.Sc., Mining Engineering, 1990) and the University of British Columbia (M.A.Sc., Mineral Process Engineering, 2004).
- I am a member in good standing of the Engineers and Geoscientists British Columbia (#30408).
- My relevant experience includes 31 years of experience in mining and mineral processing plant operation, engineering, project studies and management of various types of mineral processing, including hydrometallurgical processing for mineral deposits.
- I am a “Qualified Person” for the purposes of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for those sections of the Technical Report that I am responsible for preparing.
- I have not conducted a personal inspection of the PLS Property.
- I am responsible for Sections 2, 3, 18, 19, 21.1 (excl. mining capital costs), 22, 24 and related disclosure in Sections 1, 25, 26, and 27 of the Technical Report.
- I am independent of Fission Uranium Corp. as Independence is defined by Section 1.5 of NI 43-101.
- I have no prior involvement with the PLS Property that is the subject of the Technical Report.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Effective Date: January 17, 2023
Signing Date: February 28, 2023

(Signed and Sealed) Hassan Ghaffari

Hassan Ghaffari, M.A.Sc., P.Eng.
Director of Metallurgy
Tetra Tech Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Jianhui (John) Huang, Ph.D., P.Eng., do hereby certify:

- I am a Senior Metallurgist with Tetra Tech Inc. with a business address at Suite 1000, 10th Floor, 885 Dunsmuir Street, Vancouver, British Columbia, V6C 1N5.
- This certificate applies to the technical report entitled “Feasibility Study, NI 43-101 Technical Report, for the PLS Property”, with an effective date of January 17, 2023 (the “Technical Report”).
- I am a graduate of North-East University, China (B.Eng., 1982), Beijing General Research Institute for Non-ferrous Metals, China (M.Eng., 1988), and Birmingham University, United Kingdom (Ph.D., 2000).
- I am a member in good standing of the Engineers and Geoscientists British Columbia (#30898).
- My relevant experience includes over 36 years involvement in mineral processing for base metal ores, gold and silver ores, and rare metal ores, and mineral processing plant operation and engineering including hydrometallurgical mineral processing for various mineral mineralization.
- I am a “Qualified Person” for purposes of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for those sections of the Technical Report that I am responsible for preparing.
- I conducted a personal inspection of the PLS Property on October 13 and 14, 2021 and witnessed leaching, solvent extraction and settling tests conducted by SGS, Lakefield on April 19, 2022.
- I am responsible for Sections 13, 21.2 (excl. mining operating costs) and related disclosure in Sections 1, 25, 26, and 27 of the Technical Report.
- I am independent of Fission Uranium Corp. as Independence is defined by Section 1.5 of NI 43-101.
- I have no prior involvement with the PLS Property that is the subject of the Technical Report.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Effective Date: January 17, 2023
Signing Date: February 28, 2023

(Signed and Sealed) Jianhui (John) Huang

Jianhui (John) Huang, Ph.D., P.Eng.
Senior Metallurgist
Tetra Tech Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Patrick Donlon, FAusIMM, FSAIMM, do hereby certify:

- I am an Associate Senior Metallurgist with Tetra Tech Vancouver with a business address at 1000-885 Dunsmuir Street, Vancouver, British Columbia, V6C 1N5.
- This certificate applies to the technical report entitled “Feasibility Study, NI 43-101 Technical Report, for the PLS Property”, with an effective date of January 17, 2023 (the “Technical Report”).
- I graduated with an Extraction Metallurgy National and Higher National Diploma from Johannesburg University’s Technikon Witwatersrand School of Mining, South Africa, in 1986.
- I have 36 years’ experience in technical support and management of metallurgical operations, and as a consultant, lead design engineer and principal metallurgist engaged in the design of metallurgical processes. My expertise is in mineral processing and recovery of metals including gold, platinum, uranium, and ferrous metals.
- I am a Fellow of the Australian Institute of Mining and Metallurgy (member number 308860), and I am a Fellow of the South African Institute of Mining and Metallurgy (member number 701397). I am in good standing with these professional institutions.
- I am a “Qualified Person” for purposes of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for those sections of the Technical Report that I am responsible for preparing.
- I have not conducted a personal inspection of the PLS Property.
- I am responsible for Section 17 and related disclosure in Sections 1, 25, 26, and 27 of the Technical Report.
- I am independent of Fission Uranium Corp. as Independence is defined by Section 1.5 of NI 43-101.
- I have no prior involvement with the PLS Property that is the subject of the Technical Report.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Effective Date: January 17, 2023
Signing Date: February 28, 2023

(Signed and Sealed) Patrick Donlon

Patrick Donlon, FAusIMM, FSAIMM
Principal Metallurgist
Tetra Tech

CERTIFICATE OF QUALIFIED PERSON

I, Mark Wittrup, M.Sc., P.Eng., P.Geo., CMC, do hereby certify:

- I am Vice-President Environmental and Regulatory Affairs with Clifton Engineering Group Inc. at 10509– 46th Avenue NE, Calgary, Alberta, T2C 5C2.
- This certificate applies to the technical report entitled "Feasibility Study, NI 43-101 Technical Report, for the PLS Property", with an effective date of January 17, 2023 (the "Technical Report").
- I am a graduate of the University of Saskatchewan in 1988 with a Master of Science, Geology, and Lakehead University in 1979 with an Honours Bachelor of Science, Geology.
- I am registered as a Professional Engineer and Geologist in the Province of Saskatchewan (#05325), and a Professional Engineer in the Provinces of Alberta (#182977), British Columbia (#183022), Manitoba (#43989) and Yukon Territory (#2739). I have worked as an environmental engineer and geologist for a total of 41 years since obtaining my undergraduate degree. My relevant experience for the purpose of the Technical Report is:
  - 31 years with a major uranium mining company with 3 years uranium exploration, and over 28 years environmental and regulatory experience specifically related to uranium mines and nuclear facilities globally, with a focus on Northern Saskatchewan;
  - Project manager for the federal and provincial environmental assessment, approvals and permitting processes and main author of the EIS for a high-grade uranium mine;
  - Four years Assistant Deputy Minister, Environmental Protection and Audit, and Commissioner Environmental Assessment, Saskatchewan Ministry of Environment;
  - Participated in the implementation of the IAEA Additional Protocols with a major uranium mining company and have participated on revising the IAEA NORM Guidelines;
  - Have worked on environmental/regulatory projects directly related to 13 uranium mines and advanced properties in Canada, United States, Australia, Kazakhstan and Greenland; and,
  - I am one of the section authors (governance and implementation) for the Canadian Mining and Metallurgy ESG Guidelines scheduled for publication in 2023.
- I am a "Qualified Person" for purposes of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for those sections of the Technical Report that I am responsible for preparing.
- My most recent personal inspection of the PLS Property was on June 13 and June 14, 2019. I have been to the site and area a total of five times.
- I am responsible for Section 20 and related disclosure in Sections 1, 25, 26, and 27 of the Technical Report.
- I was appointed a special advisor to Fission Uranium's Board in 2018. I receive no direct financial consideration for this appointment. The advice and observations provided to the Board are consistent with the advice and observations detailed in this Technical Report. I am independent of Fission Uranium Corp. as Independence is defined by Section 1.5 of NI 43-101.
- I have had involvement with the PLS Property that is the subject of the Technical Report, in acting as a Qualified Person for the "Technical Report on the Preliminary Economic Assessment of The Patterson Lake South Property, Northern Saskatchewan, Canada" with the report effective date of September 14, 2015, the "Technical Report on the Pre-Feasibility Study on The Patterson Lake South Property, Northern Saskatchewan, Canada" with the report effective date of May 30, 2019 and the "Technical Report on the Pre-

Feasibility Study on The Patterson Lake South Property using Underground Mining Methods, Northern Saskatchewan, Canada" with the report effective date of September 19, 2019.

- I have read NI 43-101 and the sections of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Effective Date: January 17, 2023
Signing Date: February 28, 2023

(Signed and Sealed) Mark Wittrup

Mark Wittrup, M.Sc., P.Eng., P.Geo., CMC
Vice-President Environmental and
Regulatory Affairs
SLR Consulting Ltd.

CERTIFICATE OF QUALIFIED PERSON

I, Wayne Clifton, P.Eng.. do hereby certify:

- I am CEO and Senior Principal with Clifton Engineering Group Inc. at 340 Maxwell Crescent, Regina, Saskatchewan S4N 5Y5.
- This certificate applies to the technical report entitled “Feasibility Study, NI 43-101 Technical Report, for the PLS Property”, with an effective date of January 17, 2023 (the “Technical Report”).
- I am a graduate of University of Saskatchewan located in Saskatoon, SK at which I earned my Bachelor Degree in Civil Engineering (B.Eng. 1963) and Masters in Civil Engineering (MSc. 1965), followed by an MSc and DIC in Soil Mechanics (Geotechnical Engineering) from Imperial College, University of London, 1966. I have practiced my profession continuously since graduation. My summarized career experience is as follows:
  - 1966-73: Senior Geotechnical Engineer, Saskatchewan Highways and Transportation
  - 1973 to present: Consulting Geotechnical Engineer
  - 1978 to 2018: CEO and Senior Principal, Clifton Associates Ltd
  - 2018 to present: CEO and Senior Principal, Clifton Engineering Group Inc.
- I am licensed by the Professional Engineers, Geologists and Geophysicists of AB (No.M33284); Professional Engineers and Geoscientists of SK (No.1758); and Professional Engineers and Geoscientists of ON (No. 100556418).
- I am a “Qualified Person” for purposes of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for those sections of the Technical Report that I am responsible for preparing.
- I have not visited the site, but have relied on the observations of my staff as they have conducted geotechnical work on site.
- I am responsible for Sections 18.6 (TMF) and related disclosure in Sections 1, 25, 26, and 27 of the Technical Report.
- I am independent of Fission Uranium Corp. as Independence is defined by Section 1.5 of NI 43-101.
- I have no prior involvement with the PLS Property that is the subject of the Technical Report.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Effective Date: January 17, 2023
Signing Date: February 28, 2023

(Signed and Sealed) Wayne Clifton

Wayne Clifton, P.Eng.
CEO and Senior Principal
Clifton Engineering Group Inc.

SLR International Corporation
1658 Cole Blvd, Suite 100, Lakewood, CO 80401

CERTIFICATE OF QUALIFIED PERSON

I, Mark B. Mathisen, C.P.G., do hereby certify:

- I am Principal Geologist with SLR International Corporation of Suite 100, 1658 Cole Boulevard, Lakewood, Co., USA 80401.
- This certificate applies to the technical report entitled "Feasibility Study, NI 43-101 Technical Report, for the PLS Property", with an effective date of January 17, 2023 (the "Technical Report").
- I am a graduate of Colorado School of Mines in 1984 with a B.Sc. degree in Geophysical Engineering.
- I am a Registered Professional Geologist in the State of Wyoming (No. PG-2821) and a Certified Professional Geologist with the American Institute of Professional Geologists (No. CPG-11648). I have worked as a geologist for a total of 30 years since my graduation. My relevant experience for the purpose of the Technical Report is:
  - Mineral Resource estimation and preparation of NI 43-101 Technical Reports.
  - Director, Project Resources, with Denison Mines Corp., responsible for resource evaluation and reporting for uranium projects in the USA, Canada, Africa, and Mongolia.
  - Project Geologist with Energy Fuels Nuclear, Inc., responsible for planning and direction of field activities and project development for an in situ leach uranium project in the USA. Cost analysis software development.
  - Design and direction of geophysical programs for US and international base metal and gold exploration joint venture programs.
- I am a "Qualified Person" for purposes of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for those sections of the Technical Report that I am responsible for preparing.
- My most recent personal inspection of the PLS Property was from August 6 to 8, 2018.
- I am responsible for Sections 4 through 12, 14, 23 and related disclosure in Sections 1, 3, 25, 26, and 27 of the Technical Report.
- I am independent of Fission Uranium Corp. as Independence is defined by Section 1.5 of NI 43- 101.
- I have had involvement with the PLS Property that is the subject of the Technical Report, in acting as a Qualified Person for the "Technical Report on the Pre-Feasibility Study on The Patterson Lake South Property, Northern Saskatchewan, Canada" with the report effective date of May 30, 2019 and the "Technical Report on the Pre-Feasibility Study on The Patterson Lake South Property using Underground Mining Methods, Northern Saskatchewan, Canada" with the report effective date of September 19, 2019.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.

- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: January 17, 2023
Signing Date: February 28, 2023

(Signed and Sealed) Mark B. Mathisen

Mark B. Mathisen, C.P.G.
Principal Geologist
SLR International Corporation

CERTIFICATE OF QUALIFIED PERSON

I, Maurice Mostert, P.Eng., FSAIMM., M.Sc., do hereby certify:

- I am an independent mining contractor and owner of Mostert Mining Consulting with a business address at Suite 504, 999 Canada Place, Vancouver, British Columbia, V6C 3T4, Canada.
- This certificate applies to the technical report entitled “Feasibility Study, NI 43-101 Technical Report, for the PLS Property”, with an effective date of January 17, 2023 (the “Technical Report”).
- I am a Mining Engineer by profession (University of the Witwatersrand–M.Sc. Mining Engineering.), registered with the Southern African Institute of Mining and Metallurgy as a Fellow (FSAIMM) and a graduate of the University of South Africa (UNISA), holding a Bachelor of Technology (B-Tech) Degree. I have practiced my profession for more than 20 years. I have been directly involved in mining and mineral processing projects in Canada, the United States, South America, South Africa, and Europe.
- I am the holder of a Professional Engineer’s (P.Eng.) designation through the Professional Engineers Association of Saskatchewan, member number 72839, as well as Engineers and Geoscientists of British Columbia, member number 5282.
- I am a member in good standing of the Canadian Institute of Mining and Metallurgy, member number 708681.
- I am a “Qualified Person” for purposes of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for those sections of the Technical Report that I am responsible for preparing.
- I conducted a personal inspection of the PLS Property on October 13 and 14, 2021.
- I am responsible for Sections 15, 16 (except 16.2 and 16.3), 21 (mining costs) and related disclosure in Sections 1, 25, 26, and 27 of the Technical Report.
- I am independent of Fission Uranium Corp. as Independence is defined by Section 1.5 of NI 43-101.
- I have no prior involvement with the PLS Property that is the subject of the Technical Report.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Effective Date: January 17, 2023
Signing Date: February 28, 2023

(Signed and Sealed) Maurice Mostert

Maurice Mostert, P.Eng., FSAIMM., M.Sc.
Owner
Mostert Mining Consulting

CERTIFICATE OF QUALIFIED PERSON

I, Catherine Schmid, M.Sc., P. Eng., do hereby certify:

- I am Senior Geotechnical Engineer, with BGC Engineering Inc., with a business address at 234 St. Paul Street, Kamloops, BC, Canada, V2C 6G4.
- This certificate applies to the technical report entitled “Feasibility Study, NI 43-101 Technical Report, for the PLS Property”, with an effective date of January 17, 2023 (the “Technical Report”).
- I graduated from Queen’s University with a Bachelor of Applied Science degree in Geological Engineering in 2002 and a Master’s of Science in Engineering in 2005.
- I am a Professional Engineer, registered with the Association of Professional Engineers and Geoscientists of Saskatchewan (APEGS), member number 16362.
- I have practiced my profession continuously since 2002 and specialize in underground rock mechanics and provide operational support for underground mine and tunneling operations. I have worked at mining operations and projects throughout Canada, the United States, Africa and Europe.
- I am a “Qualified Person” for purposes of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for those sections of the Technical Report that I am responsible for preparing.
- My most recent personal inspection of the PLS Property was conducted from June 16 to June 21, 2021.
- I am responsible for Sections 16.2 and related disclosure in Sections 1, 25, 26, and 27 of the Technical Report.
- I am independent of Fission Uranium Corp. as Independence is defined by Section 1.5 of NI 43-101.
- I have no prior involvement with the PLS Property that is the subject of the Technical Report.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Effective Date: January 17, 2023
Signing Date: February 28, 2023

(Signed and Sealed) Catherine Schmid

Catherine Schmid, M.Sc., P.Eng.
Senior Geotechnical Engineer
BGC Engineering Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Randi Thompson (nee Williams), M.A.Sc., P. Eng., do hereby certify:

- I am a Principal Hydrogeological Engineer, with BGC Engineering Inc., with a business address at 234 St. Paul Street, Kamloops, BC, Canada, V2C 6G4.
- This certificate applies to the technical report entitled "Feasibility Study, NI 43-101 Technical Report, for the PLS Property", with an effective date of January 17, 2023 (the "Technical Report").
- I graduated from the University of Wisconsin – Madison with a Bachelor of Science degree in Geological Engineering in 2001 and from the University of British Columbia – Vancouver with a Master of Applied Science in Geological Engineering in 2005.
- I am a Professional Engineer, registered with the Association of Professional Engineers and Geoscientists of Saskatchewan, member number 58129.
- I have practiced my profession continuously since 2001 and specialize in the assessment and characterization of hydrogeological conditions at mine sites. I have worked at mining projects throughout Canada and the United States.
- I am a "Qualified Person" for purposes of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for those sections of the Technical Report that I am responsible for preparing.
- I have not conducted a personal inspection of the PLS Property.
- I am responsible for Section 16.3 and related disclosure in Sections 1, 25, 26, and 27 of the Technical Report.
- I am independent of Fission Uranium Corp. as Independence is defined by Section 1.5 of NI 43-101.
- I have no prior involvement with the PLS Property that is the subject of the Technical Report.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Effective Date: January 17, 2023
Signing Date: February 28, 2023

(Signed and Sealed) Randi Thompson

Randi Thompson, M.A.Sc., P.Eng.
Principal Hydrogeological Engineer
BGC Engineering Inc.