Preliminary Economic Assessment, NI 43-101 Technical Report, for the Loki Flake Graphite Deposit, Key Lake South Project, Saskatchewan, Canada

PREPARED FOR
Abasca Resources Inc.

EFFECTIVE DATE: AUGUST 19, 2026
719-26179.00

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UNITS OF MEASURE

above mean sea level	amsl
acre	ac
ampere	A
annum (year)	a
bank cubic metres	bm^3
barrels	bbl
billion	B
billion tonnes	Bt
billion years ago	Ga
British thermal unit	BTU
calorie	cal
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
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 (United States)	USD$
dollar (Canadian)	CAD$ or C$
dry metric tonne	dmt
foot	ft or ′
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

TETRA TECH

hour	h
hours per day	h/d
hours per week	h/wk
hours per year	h/a
inch	in. or "
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
kilowatt	kW
kilowatt hour	kWh
kilowatt hours per tonne	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 tonnes	Mt
minute (plane angle)	'
minute (time)	min
month	mo
Neutron	N
pascal	Pa
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
thousand tonnes	kt
three dimensional	3D
three dimensional model	3DM
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

ACRONYMS AND ABBREVIATIONS

Acronym	Definition
ABA	Acid-Base Accounting
ABA.V	Abasca Resources Inc. (Stock Quote)
Abasca	Abasca Resources Inc.
AGG	Airborne Gravity Gradiometer
Ai	Abrasion Index
Al	Aluminum
ANFO	Ammonium Nitrate/Fuel Oil
ARD	Acid Rock Drainage
BGC	BGC Engineering Inc.
BR	Bedrock
BT-R	Basal Till and Regolith
BWi	Ball Mill Work Index
Ca	Calcium
CAD, C$	Canadian Dollar
Capex	Capital Costs
CDA	Canadian Dam Association
Cg	Graphite Carbon
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CNSC	Canadian Nuclear Safety Commission
CRM	Certified Reference Materials
DCS	Distributed Control System
Denison	Denison Mines Corp.
DGM	Discrete Gaussian Model
DTM	Digital Terrain Model
E-houses	Electrical Houses
EA	Environmental Assessment
EAB	Environmental Assessment Branch
EIA	Environmental Impact Assessment
EIS	Environmental Impact Statement
EM	Electromagnetic
EPCM	Engineering, Procurement, and Construction Management
FCF	Free Cash Flow
Fe	Iron
FS	Feasibility Study

Acronym	Definition
gD	Vertical Gravity
gDD	Vertical Gravity Gradient
GF	Glaciofluvial Sands
GPS	Global Positioning System
Gov't	Government
HDPE	High-Density Polyethylene
HSUs	Hydro-Stratigraphic Units
HYL	High Yield Limit
I/O	Input/Output
IAA	Impact Assessment Act
ICP-MS	Inductively Coupled Plasma Mass Spectrometry
IGS-CG	Interglacial Coarse-Grained Sediments
IGS-FG	Interglacial Fine-Grained Sediments
IRR	Internal Rate of Return
Key Lake	Key Lake Operation
KLS	Key Lake South Project (the Project)
LAN/WAN	Local/Wide Network
LG	Lerchs-Grossmann
LLDPE	Linear Low-Density Polyethylene
LOI	Loss on Ignition
Loki Deposit	Loki Flake Graphite Deposit
LOM	Life of mine
M	million
MAC	Mining Association of Canada
MCLP	Misty Clifton Engineering LP
Mg	Magnesium
MIBC	Methyl Isobutyl Carbinol
ML	Metal Leaching
MRE	Mineral Resource Estimate
NAD 83	North America Datum 1983
NI 43-101	National Instrument 43-101 Standards of Disclosure for Mineral Projects
NN	Nearest Neighbour
NP	Neutralization Potential
NPAG	Not-Potentially Acid Generating
NPR	Neutralization Potential Ratio
NPV	Net Present Value

Acronym	Definition
NSG	Non-Sulphide Gangue Minerals
NTS	National Topographic System
OCSs	Operator Control Stations
OK	Ordinary Kriging
Opex	Operating Costs
P.Eng.	Professional Engineer
P.Geo.	Professional Geologist
PAG	Potentially Acid Generating
PC	Personal Computer
PEA	Preliminary Economic Assessment
PFS	Pre-Feasibility Study
PhD	Doctor of Philosophy
PLCs	Programmable Logic Controllers
POPCs	Parameters of Potential Concern
Q-Q	Quantile-Quantile
QA/QC	Quality Assurance/Quality Control
QP	Qualified Persons
REEs	Rare Earth Elements
Report	Technical Report
ROM	Run-of-Mine
RPEEE	Reasonable Prospects of Eventual Economic Extraction
RQD	Rock Quality Designation
RTO	Reverse Takeover
RWi	Rod Mill Work Index
S	Sulphur
SaskCo	101159623 Saskatchewan Ltd.
SaskPower	Saskatchewan Power Corporation
SD	Standard Deviations
SEARP	Saskatchewan Environmental Assessment Review Panel
SEQGs	Saskatchewan Environmental Quality Guidelines
SFE	Shake Flask Extraction
Si	Silicon
SI	International System of Units
SMAD	Saskatchewan Mineral Assessment Database
SPT	Standard Penetration Tests
SRC	Saskatchewan Research Council

Acronym	Definition
TDG	Transport of Dangerous Goods
Tetra Tech	Tetra Tech Canada Inc.
TIC	Total Inorganic Carbon
TMF	Tailings Management Facility
TOR	Terms of Reference
US or U.S.	United States
UCS	Unaxial Compressive Strength
UMR	Understood Mineral Resources Ltd.
USD, US$	United States Dollar
UTM	Universal Transverse Mercator
VHF	Very High Frequency
VoIP	Voice over Internet Protocol
WBS	Work Breakdown Structure
WHMIS	Workplace Hazardous Materials Information Systems
WMCZ	Wollaston-Mudjatik Contacting Zone
WRA	Whole Rock Analysis
WRSF	Waste Rock Storage Facility
WRTCF	Waste Rock Tailings Co-deposition Facility
XRD	X-Ray Diffraction

1.0 SUMMARY

1.1 Executive Summary

Tetra Tech Canada Inc. (Tetra Tech) was retained by Abasca Resources Inc. (Abasca) to prepare an independent technical report on the Key Lake South (KLS) Project (the Project), located 15 km south of the Key Lake Mine in central Saskatchewan, Canada. Abasca is a Canadian exploration company, primarily engaged in the acquisition, evaluation, and development of uranium and graphite properties. Abasca is listed on the Toronto Stock Exchange Venture with the symbol ABA. This Technical Report (the "Report") focuses on the Loki Flake Graphite Deposit (the "Loki Deposit" or the "Property"), which is 100% owned by Abasca.

This Report has been prepared by Tetra Tech in accordance with National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101), with report sections related to geology and mineral resource estimate provided by Understood Mineral Resources Ltd. (UMR). The purpose of this Technical Report is to summarize the results of a Preliminary Economic Assessment (PEA) of the Project and updated Mineral Resource Estimate announced by Abasca on July 14, 2026.

Dr. Sabry Hafez, P.Eng. PhD., of Tetra Tech, visited the Property on June 18, 2026, and conducted a general project site overview in the proposed open pit and rock storage areas. Mr. Matt Batty, MSc, P. Geo, of UMR, visited the Property from January 28 to 29, 2025. During Mr. Batty's site visit, he examined recent drill core, confirmed collar locations, reviewed drilling, logging, sampling, analytical and QA/QC procedures, and reviewed site facilities.

The Company plans to develop the Loki Flake Graphite Project using conventional open-pit mining methods, including drilling, blasting, loading, and hauling with conventional mining equipment and conventional mineral processing methods including crushing, grinding, flotation, and concentrate cleaning and dewatering. The PEA features a 2,750 tonnes-per-day (t/d) open-pit mining and processing operation with a 19-year mine life, averaging 66,500 tonnes of graphite concentrate (average 95% graphite carbon [Cg] or higher) produced annually and 1.2 million tonnes of payable graphite over the life of mine.

The PEA projects positive project economics with a pre-tax Net Present Value (NPV) of US$161M discounted at 8% and a pre-tax Internal Rate of Return (IRR) of 16.6%, and an after-tax NPV of US$130M discounted at 8% and an after-tax IRR of 16.7%. The pre-tax and after-tax simple payback periods are 6.2 years and 4.7 years, respectively.

Based on the PEA economic analysis results, Abasca believes the Project is well positioned to advance to feasibility-level engineering.

This PEA Technical Report is considered by the Qualified Persons (QP) to meet the requirements of a "Preliminary Economic Assessment" as defined in NI 43-101. The economic analysis contained in this Technical Report is based, in part, on Inferred Resources, and is preliminary in nature. Inferred Resources are considered too geologically speculative to have mining and economic considerations applied to them and to be categorized as Mineral Reserves. There is no certainty that the economic forecasts contained herein will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

1.2 Project Description and Location

The Project, which consists of 12 contiguous claims (23,974 Ha), is located approximately 540 km north of Saskatoon, the largest city in the province of Saskatchewan, and 220 km north-northwest of the town of La Ronge. The Key Lake Mine, a former uranium producer and the site of the Key Lake mill which processes Cameco's McArthur River ore, is located 15 km northeast of the Project. The Project is approximately centered at UTM NAD83 Zone 13N grid coordinates 442500E, 6322500N, on NTS map sheets 74-G-01 and 74-H-04.

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

The Project is accessible all year round by provincial highway 914, which runs through the project area. The nearest communities of Pinehouse and La Ronge are respectively located approximately 185 km and 400 km south of the project area by road. Supplies, groceries, fuel, and accommodations can be found in these communities. Accurate Industries operates a stationary modular camp 210 km north of Pinehouse on Highway 914. This camp is located within 10 km North of the Project and provides all necessary camp amenities.

The topography over the project area is generally flat lying with low ridges and hills reaching up to several tens of metres in relief. The area is characterized by numerous lakes, creeks, and ponds. Annual temperatures generally range between -40°C in the winter and 35°C in the summer. Mean temperatures are -25°C and 25°C in the winter and summer, respectively. Average annual rain- and snow-fall are 225 mm and 2,150 mm, respectively. Lake ice typically thaws in late April and returns in October.

1.4 History

The qualified persons have not verified the information of the adjacent properties, and that the information of the adjacent properties is not necessarily indicative of the mineralization on the project that is the subject of the technical report.

Considerable exploration activity by multiple operators has occurred on the Project. Early regional exploration activities between the late 60s and throughout the 70s focused on ground prospecting, lake water geochemistry, and airborne radiometric and electromagnetic (EM) surveys as well as some shallow drilling. Although these early activities led to the discovery of the nearby Gaertner and Deilmann deposits that comprise Key Lake, no significant uranium mineralization was discovered over the current Project area during that time. During the early 80s, anomalous radioactivity in pegmatites was discovered during ground prospecting surveys over the Project area and shallow follow-up drilling was done during the mid-2000s with limited technical success.

101159623 Saskatchewan Ltd. (SaskCo) acquired ownership of the project through staking in 2011 and 2012. Various compilation reports and interpretations of historical data were undertaken by SaskCo, both in-house and by independent consultants, in 2011, 2012, and 2013, resulting in identifying target areas and recommendations for future exploration work. Field programs, including airborne and ground geophysical surveys, geological mapping, soil and lake sediment geochemical surveys, and radon-soil-surveys were conducted by SaskCo in 2014 and 2015.

In 2016, SaskCo carried out a diamond drilling program to test geological features interpreted by the previous exploration data, including the 2014 HeliFALCON Airborne Gravity Gradiometer survey, EM surveys, geochemistry surveys, and geological mapping. A total of 4,553 m were drilled comprising 26 NQ holes in two target areas: 15 holes totalling 2,744 m in the Campbell target area and 11 holes totalling 1,809 m in the Mustang target area. The best drillhole of the Mustang drilling was 256 ppm U over 0.4 m from 48.8 to 49.2 m in hole KS-MS16-07. Drilling at the Campbell target area included multiple graphitic fault zone intersections up to 40 m in length. This was the discovery of what is now referred to as the Loki Flake Graphite Deposit, although only representative samples were collected through the zone and no samples were evaluated for graphite at the time. In 2022, Condor Consulting Inc. was engaged in further interpreting and modelling available geophysical survey data to assist with SaskCo's in-house technical team for structural analysis, target prioritizing, and detailed drillhole planning.

At the end of 2022, Abasca Resources Inc. acquired the project through reverse takeover.

1.5 Exploration and Drilling

In 2023, Abasca continued uranium exploration efforts and completed 25 diamond drill holes totaling 10,135 m over winter and summer drill programs. Geochemical assays confirmed anomalous uranium intersections at the Mustang target area where 9 of the 11 drillholes intersected anomalous uranium (> 100 ppm U). KLS-23-004, the best hole of the campaign, intersected 1260 ppm U between 310.5 and 310.6 m. Additionally, in the fall of 2023, samples from the 2016 Campbell drill core were re-analysed for graphite. Highlighted intercept from KS-CC16-12 includes a half-metre sample grading 22.2 % Cg.

Based on the results of the 2023 graphite analyses, in 2024 Abasca delineated what is now known as the Loki Flake Graphite Deposit (Loki Deposit). A total of 20 diamond drill holes for 5,499 m, were drilled into the middle section of the known extents of the conductor. All holes intersected significant graphitic intervals (≥ 1 % Cg over at least 1.0 m) similar to those drilled in 2016. This includes highlighted intercepts in KLS-24-031 (7.18 % Cg over 52 m) and KLS-25-043 (9.01 % Cg over 60 m). An additional 2 holes, totaling 912 m, were drilled on a parallel conductor north of the Loki Deposit. These holes intersected a new graphite zone (Thor Zone) up to 50 m wide. This includes a highlighted intercept in KLS-24-046 (8.91 % Cg over 27 m). A total of 5 holes were also drilled along the Mustang-Seager Lake trend, totaling 2,681 m where drilling intersected pervasive silicification zones and local graphitic fault zones.

Continued delineation at the Loki Deposit was completed over two drill programs in 2025, comprising 51 diamond drill holes for a total of 11,950.6 m. The programs focused on extending the 2024 delineated zone at 100 m grid-spacing as well as infill drilling of the 2024 drilling area. One hole was abandoned at 72 m due to poor ground conditions. The programs successfully delineated the extension along strike by 300 m toward the northwest and 400 m toward the southeast. Highlighted intercepts include KLS-25-101 (10.11 % Cg over 49.7 m), and KLS-25-097 (9.49 % Cg over 56 m). One diamond drill hole (KLS-25-072) totalling 429 m was drilled at the Thor Zone (400 m up-dip from KLS-24-046), where 72 m of graphite mineralization was intersected over 3 zones. Highlights intercepted in KLS-25-072 include 7.36% Cg over 8 m, 9.78% Cg over 32 m, and 6.39% Cg over 32 m.

Geotechnical data were collected at the Loki Deposit throughout the 2025 summer program. Additionally, 6 sonic holes totalling 235.2 m were drilled to characterize the overburden and to test the unconformity for uranium mineralization. A total of 3 holes had vibrating wire piezometers installed and 3 other holes had Standpipe Piezometers installed to analyze hydrogeological features.

As of the effective date of this report, the Company completed a winter drill program at the Loki Deposit in 2026 totaling 13 diamond drill holes for a total of 2,961 m. One hole was abandoned at 36 m due to poor ground conditions. The program focused on extending the inferred-spaced drilling along strike by 200 m towards the southeast as well as collecting geotechnical data. An additional 12 sonic holes, totalling 459 m were drilled to characterize the overburden and to test the unconformity for mineralization. An additional 5 diamond drill holes, totalling 1,556 m, were drilled at the Thor Zone, of which one hole was abandoned at 33 m due to poor ground conditions. Highlighted intercepts include KLS-26-112 (7.78 % Cg over 18.75 m, 3.23 % Cg over 21 m, and 7.8 % Cg over 14 m) and KLS-26-115 (4.7 % Cg over 6 m, 3.36 % Cg over 10 m, 3.48 % Cg over 68 m, and 7.42 % Cg over 14 m).

1.6 Geology and Mineralization

The Loki Deposit is underlain by the prospective uranium hosting rocks of the Wollaston-Mudjatik contacting zone (WMCZ) in the southeastern Athabasca Basin. The world's largest high-grade uranium deposits are associated with the unconformity between the Athabasca Basin and the Wollaston-Mudjatik basement. Most of the uranium occurrences and deposits associated with the Athabasca Basin are located near the boundary between the Mudjatik and Wollaston domains as either unconformity-related or basement-hosted type. The Loki Deposit is located in the southern strike extent of these deposits and in the same regional magnetic low structure that hosts them. Alteration at the Deposit shows characteristics of basement-hosted uranium deposits.

Through geochronological studies, uranium mineralization in the Athabasca Basin have shown that most deposits were formed between 1,330 Ma and 1,380 Ma (Cumming and Krstic, 1992), and as early as 1,590 Ma at the Millennium Deposit (Cloutier et al, 2009) and 1,521 Ma at the McArthur River Mine (Cameco Corporation, 2012) with ages of remobilization near 1,350 Ma. The uranium deposits in the region generally occur at the unconformity between the lowermost Athabasca Group and the underlying crystalline basement rocks, or completely within the basement rocks. The deposits are commonly localized to the intersection of faults and the unconformity, or at a paleotopographic basement ridge. No significant uranium mineralization has been discovered at the Loki Deposit.

Graphite mineralization at the Loki Deposit occurs within regionally extensive fault zones. Although there are numerous graphitic faults at the Deposit, most drillhole intersections do not exceed 15 m in length outside of the area near the Loki Deposit where intersections lengths are on average approximately 35 m and extend along strike for over 1.5 km. The Loki Deposit is a moderately dipping, southeast-striking graphitic fault zone that is orthogonal to the predominant orientation of regional structures. The deposit has an approximate strike-length of 1,700 m and extends from the lower contact of the overburden, approximately 55 m below surface, to 335 m below surface where it remains open.

1.7 Data Verification

Abasca has a robust QA/QC process in place, as described in Section 11.0. Assay results were actively monitored throughout the drill programs and QA/QC results were summarized. Several failures for standard reference materials were documented, resulting in the re-assaying of entire sample batches. Most of the reference materials performed as expected within tolerances of two to three standard deviations of the mean grade. The QP is satisfied that the QA/QC process is performing as designed to ensure the quality of the assay data.

In 2025, the QP independently verified 353 assay records from 8 drill holes against 8 different laboratory certificates. No major discrepancies were found.

Mr. Batty visited the Project from January 28 to 29, 2025. During the QP's two-day site visit, he reviewed ongoing and recent core from the Deposit, confirmed the location of three collar locations with a handheld GPS, verified the geological setting, and reviewed drilling, logging, sampling, analytical and QA/QC procedures.

In Mr. Batty's opinion, the Loki Project exploration data are free of any material or systematic errors and are considered well validated and of sufficient quality for use in this Technical Report.

Dr. Sabry Hafez, P.Eng. PhD., of Tetra Tech, visited the Project on June 18, 2026 and conducted a general project site overview in the proposed open pit and rock storage areas.

Dr. Hafez considers the information supporting the mining, mine planning and economic-analysis portions of the PEA to be adequate for the purposes of this Technical Report.

1.8 Mineral Processing and Metallurgical Testing

A preliminary test program for recovering the graphite from the Loki Flake Graphite Deposit was conducted by SGS Lakefield during 2025 and 2026. The test program included chemical characterization, mineralogical analyses, grindability testing, and flotation testing, graphite concentrate and flotation tailings characterization.

The test results indicate that the samples respond well to conventional flotation concentration. The flowsheet uses separate flotation with staged regrinding for coarse and fine fractions, respectively. The final concentrate grades are expected to be approximately 95% Cg or higher. There was no significant difference in the graphite grade between the coarse cleaner concentrate (+ 150 µm) and the fine cleaner concentrate (-150 µm).

Further test work should be conducted to optimize the flotation conditions and improve graphite flotation performances, including conservation of coarse graphite flakes. Also, additional purification test work on flotation concentrates should be conducted to understand the optimum purification treatments, including ultra-purity graphite chemical analysis.

The test results indicate that:

- At the grinding fineness of around P80 300 µm, more than 99% graphitic carbon can be recovered in the rougher concentrate.
- For 6 mesh master samples, more than 40% graphitic carbon can be concentrated into the flash flotation froth.
- It was also noticed that the performances of the flash flotation was not stable.
- The yield of the combined flash and rougher flotation concentrate was approximately 40% with a graphite grade of 20.7%.
- There was no significant difference in the graphite grade between the coarse cleaner feed (+150 µm) and the fine cleaner feed (-150 µm).
- It seems that at the tested flotation conditions, it would be difficult for the coarse cleaner circuit to produce a graphite concentrate with a graphite grade greater than 95%.
- The cleaner flotation on the fine fraction rougher concentrate produces better metallurgical performances in upgrading graphite concentrate. It is possible to achieve a concentrate with a greater than 95% graphite grade.

1.9 Mineral Resource Estimation

The updated Mineral Resource estimate for the Deposit was completed by Matt Batty, MSc, P. Geo of UMR. The effective date of the enclosed mineral resource is April 23, 2026, coincident with the date the last assay received.

Samples were composited to 8 m lengths within the single modelled graphite domain. The composites were reviewed for outliers and declustered, resulting in a decided representative dataset for the domain. Major local lithologies observed in the deposit area were also modelled.

A block model was constructed to encompass the modelled wireframes and the blocks were populated using Ordinary Kriging (OK) as informed by the directional variogram. The lithology domains were assigned a density based on specific gravity measurements observed within the domains or observations from analogous deposits. A linear regression was used to assign density to blocks with estimated graphite grades.

The block model was validated via volume comparison, mean grade comparison, visual inspection, swath plots, and change of support comparison.

The reported Indicated Mineral Resources are generally supported by an approximate drill hole spacing of 50 m and a minimum of three drill holes informing the estimate, whereas the Inferred Mineral Resources are supported by wider drill spacing of 100 m with a minimum of three holes.

In the QP's opinion, the estimation methodology is consistent with standard industry practice and the Indicated and Inferred Mineral Resource Estimates for Loki Deposit are reasonable and acceptable.

The Mineral Resource estimate for the Deposit is based on 79 diamond drill holes. The Mineral Resources are summarized in Table 1-1 and are reported at a break-even cut-off grade of 2.30% Cg within a conceptual open pit design. No Mineral Reserves have been estimated at the Project.

Table 1-1: Loki Flake Graphite Mineral Resources (April 23, 2026)

Category	Cg Grade Cut-off (%)	Tonnage (Mt)	Cg Grade (%)	Contained Cg (Mt)
Indicated	2.30	6.99	8.27	0.58
Inferred	2.30	15.83	6.93	1.10

Notes:

1. The reporting standard for the Mineral Resource Estimate uses the terminology, definitions and guidelines given in the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Mineral Reserves (May 2014) as required by NI 43-101.
2. Reported Mineral Resources are constrained to a conceptual pit-shell above a cut-off grade of 2.30% Cg.
3. Numbers may not add up due to rounding.
4. The effective date of this Mineral Resource estimate is April 23, 2026.
5. The qualified person knows of no environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors that may materially affect the Mineral Resource Estimate in this report.
6. Mineral Resources are not Mineral Reserves and have not demonstrated economic viability.

1.10 Mineral Reserve Estimation

There are no current Mineral Reserve estimates on the Project.

1.11 Mining Methods

The Project is planned to be developed using conventional open-pit mining methods, including drilling, blasting, loading, and hauling with conventional mining equipment. Material will be drilled and blasted, then loaded into diesel-powered haul trucks using a fleet of hydraulic shovels and front-end wheel loaders.

To identify the optimal pit size and mining sequence, Tetra Tech employs the Lerchs–Grossmann (LG) algorithm to evaluate the net value of individual blocks within the block model. An assumed graphite concentrate price of US$1,450 per tonne has been applied. Life-of-mine (LOM) operating costs were estimated based on similar projects or operations.

The table and figure below show the project mine production plan.

Table 1-2: Mine Production Plan

Production Year	Total Mined (Mt)	Waste (Mt)	Strip Ratio	Mill Feed (Mt)	Grade (%)
-1	7,499,919	7,499,919	-	-	0.00
1	9,999,901	9,149,202	10.8	763,616	8.50

Production Year	Total Mined (Mt)	Waste (Mt)	Strip Ratio	Mill Feed (Mt)	Grade (%)
2	9,999,840	9,243,484	12.2	843,640	7.05
3	9,999,980	8,804,283	7.4	1,002,839	7.91
4	9,999,970	9,012,357	9.1	1,001,241	6.64
5	9,999,913	8,643,356	6.4	1,002,531	7.76
6	9,999,803	9,034,431	9.4	1,001,241	6.54
7	9,999,978	8,628,075	6.3	1,001,241	7.92
8	8,999,978	8,348,799	12.8	1,001,241	6.22
9	8,499,832	7,413,899	6.8	1,001,241	7.03
10	7,999,916	6,638,336	4.9	1,001,199	8.03
11	5,999,761	5,056,954	5.4	1,001,241	6.52
12	4,499,942	3,498,543	3.5	1,001,241	6.96
13	3,499,819	2,461,676	2.4	1,001,241	7.58
14	2,499,840	1,532,322	1.6	1,001,241	7.79
15	2,499,899	1,430,300	1.3	1,001,281	7.91
16	2,499,914	1,372,698	1.2	1,002,552	8.13
17	1,816,662	676,773	0.6	1,001,241	8.30
18	-	-	-	1,001,241	5.11
19	-	-	-	240,396	4.04
Total	**126,314,868**	**108,445,407**	**6.1**	**17,871,701**	**7.27**

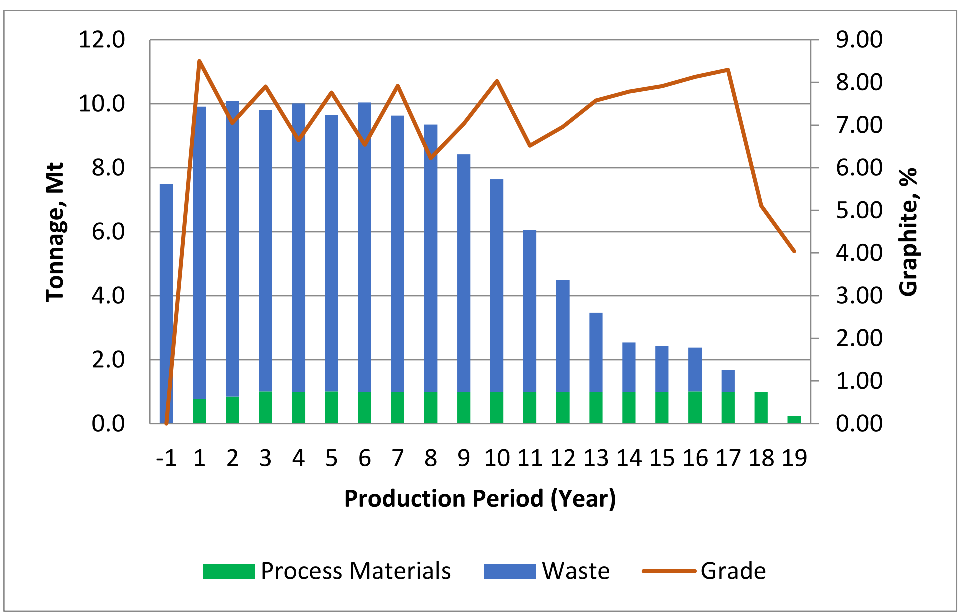


Figure 1-1: Mine Production Plan

The life-of-mine plan comprises 17 years of active mining operations followed by two years of stockpile processing. Mining is conducted at a consistent rate of 10.0 Mt of material annually during the first seven years and slowly diminishing from Y8 to Y17, with a LOM average strip ratio of 6.1:1. The process plant is designed for a nominal throughput of 2,750 tonnes per day (approximately 1.0 million tonnes per year). Average payable graphite production over the 19-year mine life is estimated at approximately 1.23 million dry tonnes per year. Mill throughput is planned at 75% of nameplate capacity in Year 1, 85% in Year 2, and 100% from Year 3 onward, allowing for a controlled start-up and optimization of plant performance. The mill feed grade fluctuates between 6.2% and 8.5% from Year 1 to Year 17, before reaching 4% to 5% during the processing of the stockpiled material from Year 18 to Year 19.

1.12 Recovery Methods

Based on the available metallurgical reports and test work results, the optimized flowsheet was developed for this study. The process flowsheet is designed for 2,750 t/d and follows a conventional flotation circuit to produce a high-grade graphite concentrate, which will be further processed at an offsite purification facility. The processing plant (see simplified process flow diagram below) will consist of the following:

- A run-of-mine (ROM) mill feed stockpile,
- A primary crusher operating in open circuit,
- A secondary crusher operating in closed circuit with a vibrating screen,
- A crushed mill feed stockpile with reclaim feeders,
- A ball mill grinding circuit operating with flash flotation and a classification hydrocyclone cluster,
- A rougher flotation circuit comprising rougher and rougher-scavenger flotation, followed by a rougher-scavenger tailings dewatering circuit, including thickening and filtration processes to generate tailings filter cakes for tailings dry stacking at a lined tailings management facility (TMF) to mitigate the impacts of tailings acid generation potentials on the environment,
- A rougher concentrate upgrading circuit comprising
  - One-stage polishing regrinding followed by one stage of cleaner flotation
  - Second-stage polishing mill followed by three stages of cleaner flotation,
- A fourth-cleaner concentrate sizing circuit to separate the upgraded rougher concentrate into coarse and fine graphite concentrate streams,
- A coarse concentrate regrind mill, followed by four additional stages of cleaner flotation (5th to 8th cleaners) to produce a final coarse graphite concentrate,
- A fine concentrate regrind mill, followed by five additional stages of cleaner flotation (5th to 9th cleaners) to produce a final fine graphite concentrate,
- A concentrate dewatering and product handling circuit, including concentrate thickening, filtration, drying, final product sizing, and packaging.

The simplified process flowsheet is shown in Figure 1-2.

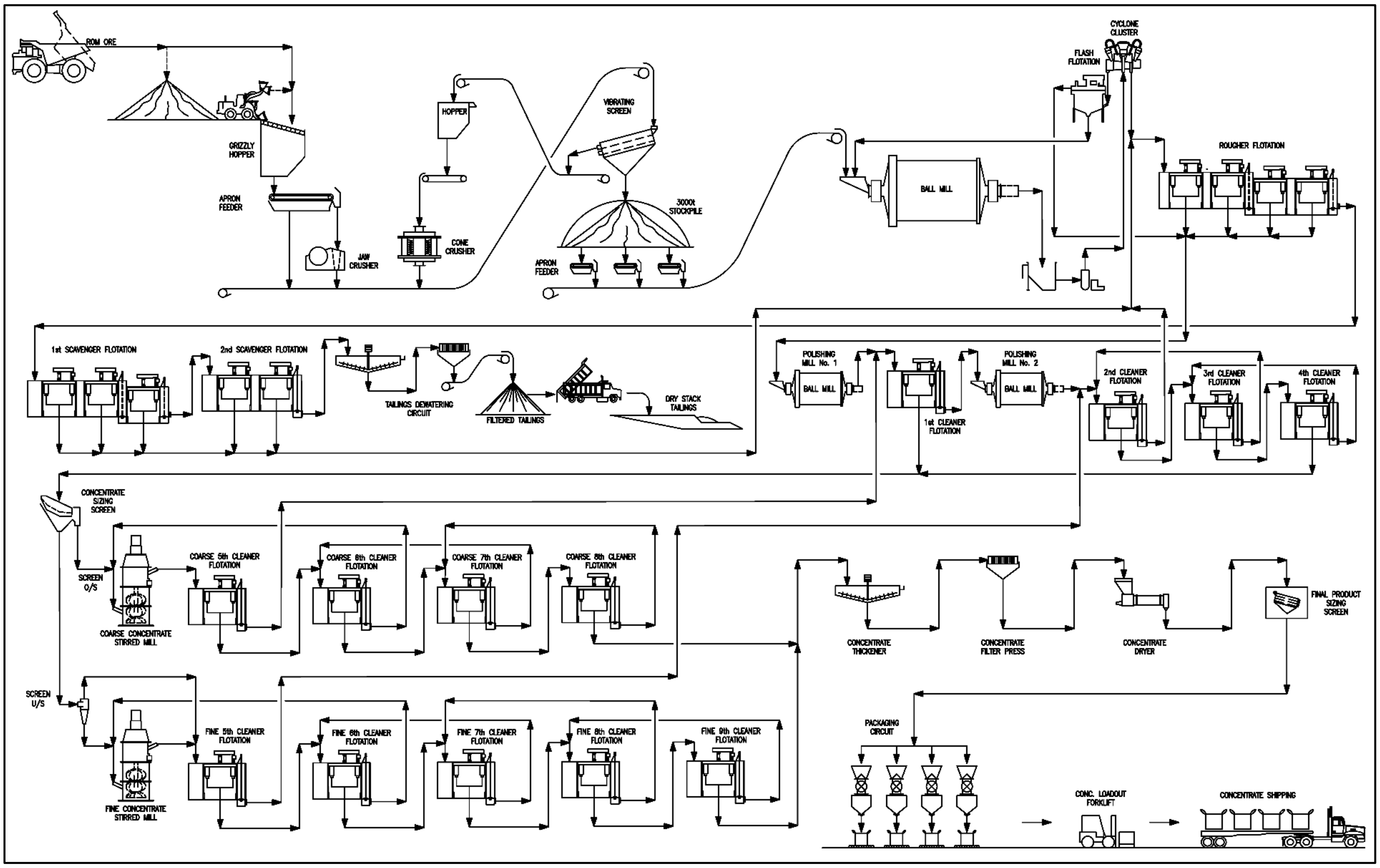


Figure 1-2: Simplified Process Flowsheet

1.13 Project Infrastructure

The Project benefits from existing transportation infrastructure, proximity to the provincial power network and favourable site characteristics that support future development, including:

- **Road Access:** The Project is located approximately 15 kilometres south of the Key Lake mill. Provincial Highway 914, a north-south all-weather highway in Saskatchewan, crosses the Project, providing ideal access.
- **Power:** The provincial power grid is merely 15 km from the Project, which can provide long-term economical and reliable electrical power for the Project.
- **Water:** Several surface water bodies are located near the project site and are expected to provide suitable water sources for future operations, subject to detailed engineering and permitting.
- **Waste Rock Tailings Co-deposition Facility (WRTCF):** The WRTCF was designed to accommodate 17.9 Mt of tailings and 108.4 Mt of waste rock over the life of the mine. The WRTCF will consist of commingled dry stacked tailings and waste rock.
- **Camp and Services:** Existing accommodation camps nearby have the potential to support construction and operations. Camp services are expected to be provided by third-party contractors, creating potential business and employment opportunities for nearby Indigenous communities.
- **Overall Site General Arrangement:** The overall site arrangement is presented in Figure 1-3.

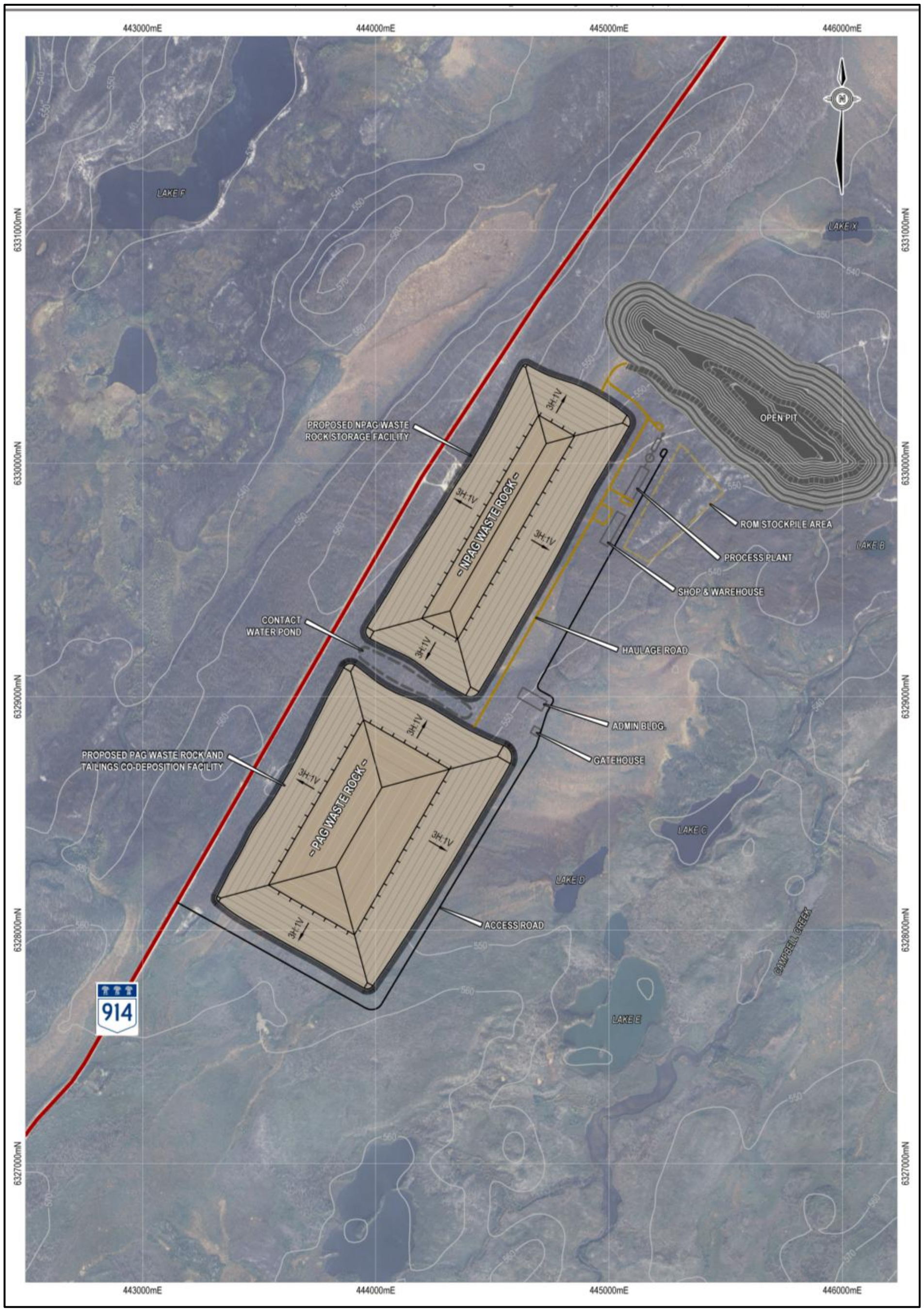


Figure 1-3: Overall Project Site General Arrangement Plan

1.14 Market Studies and Contracts

The graphite concentrate produced by the Project is expected to yield an average grade of 95% Cg or higher. The current project concept is to ship the concentrate from the Project to Saskatoon, and from there the product would be consolidated for shipment to domestic and foreign buyers and for Abasca's purification process facility planned for future. The concentrate price used for the economic analysis is US$1,450/t. The concentrate handling and transportation costs are included in the economic analysis. The marketing study summary is presented in Section 19.0. Currently, there have been no concentrate sale contracts established for the Project.

1.15 Environmental Studies, Permitting, and Social or Community Impact

In 2025, Abasca engaged CanNorth Environmental Services to conduct environmental studies on the Project. The purpose of the studies was to initiate data collection for components that require baseline datasets, specifically hydrology, water chemistry, and bathymetry to support a future Environmental Impact Assessment. Studies and data collection will continue into 2026 and 2027, including further work on aquatic, terrestrial and heritage resources as well as work on meteorological, hydrogeological and geochemical conditions. This work will advance project development and support future submissions to meet regulatory requirements.

1.16 Capital and Operating Costs

The capital costs (Capex) and operating costs (Opex) for the Project have been estimated and are summarized in Table 1-3.

Table 1-3: Summary of Capital and Operating Costs

Description	Total Capital Cost (US$ million)	Average Unit Operating Cost (US$/t processed)
Initial Capital Costs	216.5	-
Sustaining Capital and Closure Costs	56.4	-
LOM On-site Operating Costs	-	46.34

Notes:

1. Initial Capex: US$216M, including contingency of US$24.3M and US$22.5M initial mining operating costs and mining equipment lease downpayment.
2. Sustaining Capex and Closure Costs: US$56.4M from Year 1 to Year 19, followed by a 4-year period of closure and reclamation.
3. Opex: US$46.34/t processed, including mining and waste rock management (44%), processing (28%), G&A and site services (27%), and interest on mining equipment lease (1%).

All costs are reflected in 2026 Q2 US Dollars unless otherwise specified. The expected accuracy range of the cost estimates is within ±35%. Where applicable, costs in this report have been converted from Canadian Dollars to US Dollars using a currency exchange rate of CAD1.00:USD0.72.

1.17 Economic Analysis

The results of the project economic analysis are presented in Table 1-4. All dollar figures are expressed in US dollars and all units in metric, unless otherwise noted.

The PEA is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the PEA will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 1-4: Economic Analysis Results

Description	Unit	Value
Metal Price		
Graphite Price (Average)	**US$/t**	**1,450**
Production		
Mine Life	Year	19
Mill Feed Tonnage, Life of Mine (LOM)	Mt	17.9
Mill Feed Grade, Life of Mine (LOM)	% Cg	7.27
Concentrate Grade, Life of Mine (LOM) Average	% Cg	95.0
Graphite Recovery to Concentrate	%	92.3
Graphite Produced, Life of Mine (LOM)	dry tonnes	1,263,000
Annual	dry tonnes	66,500
Operating Costs, Life of Mine (LOM)	US $M	828
Unit	US$/t proc.	46.34
Operational Revenue	US $M	940
Capital Costs		
Initial Capex	US $M	216
Sustaining & Closure	US $M	91
Total Capital Costs	US $M	307
Economic Results		
Discount Rate	%	8%
Pre-Tax Unlevered Cash Flow (UCF)	**US $M**	**633**
Pre-Tax Net Present Value (NPV) @ 8%	**US $M**	**161**
Pre-Tax Internal Rate of Return (IRR)	**%**	**16.6%**
Pre-Tax Simple Payback Period	**Year**	**6.2**

Description	Unit	Value
After-Tax Unlevered Cash Flow (UCF)	**US $M**	**477**
After-Tax Net Present Value (NPV) @ 8%	**US $M**	**130**
After-Tax Internal Rate of Return (IRR)	**%**	**16.7%**
After-Tax Simple Payback Period	**Year**	**4.7**

Notes

1. *Initial Capex represents upfront expenditures to construct and commission the mine, plant, and supporting infrastructure.*
2. *Sustaining Capex represents ongoing capital expenditures required to maintain production during the life of mine.*
3. *Payback represents the years from start of commercial production to achieve cumulative positive after-tax free cash flow, including sustaining capital.*
4. *Exchange rate assumption: $0.72 USD per $1.00 CAD.*
5. *After-tax payback period is shorter than pre-tax payback period due to the tax refund receivables during the earlier project years, as forecasted by Abasca's tax expert.*

1.18 Adjacent Properties

Situated immediately north of the Project is the Key Lake mill, a joint venture of Cameco Corporation (83%) and Orano (17%), operated by Cameco. The mill has been in operation since 1983 and has processed 535 million pounds of uranium concentrate (Cameco, 2022). The operation now processes ore from the McArthur River mine.

Exploration adjacent to the Project is primarily focused on greenfields uranium exploration. CanAlaska's Key Lake Extension property is immediately to the west of the Project and Geiger Energy's Highrock Lake and Bear projects are respectively to the east and south of the Project (formerly owned by Forum Uranium and Baselode Energy, respectively).

The qualified person has not verified the information of the adjacent properties, and that the information of the adjacent properties is not necessarily indicative of the mineralization on the project that is the subject of the technical report.

1.19 Interpretation and Conclusions

Mineral Resource Estimate

The QP's related interpretation and conclusions are summarized below.

- The Loki Deposit is a broad graphite bearing shear-zone ranging in thickness along an orientation of 127° azimuth and -45° dip to the southwest. The QP created one vein wireframe to constrain the estimate in the predominate orientation of mineralization. The wireframe is approximately 1,700 m long in the strike direction with an upper contact 55 m below surface and extends to 335 m below surface. The thickness of the modelled graphite ranges between 10 and 55 m with an approximate average of 35 m.

- The 2026 Mineral Resource Estimate has an effective date of April 23, 2026, coincident with the date of the last assay result received from the analytical laboratory.

- The Mineral Resource is composed of Indicated and Inferred Mineral Resources. The Indicated Mineral Resource totals 6.99 million tonnes at an average grade of 8.27 % Cg for a total of 0.58 million tonnes Cg, and the Inferred Mineral Resource totals 15.83 million tonnes at an average grade of 6.93 % Cg for a total of 1.10 million tonnes Cg.

- The resource was reported at a cut-off grade of 2.30 % Cg within a conceptual open pit design. No Mineral Reserves have been estimated at the Project.
- In conjunction with infill drilling, the testing of uranium contamination and graphite flake quality will be important for upgrading portions of the deposit from Inferred to Indicated or Indicated to Measured Mineral Resources.
- The lithology domains were assigned a density based on specific gravity measurements observed within the domains or observations from analogous deposits. A linear regression was used to assign density to blocks with estimated graphite grades.
- In the QP's opinion, the project exploration data are free of any material or systematic errors, well validated and of sufficient quality for use in this Technical Report.
- The QP identified uranium contamination of graphite mineralization as a potential risk. The QP believes that the likelihood of realizing this risk in a material sense is minimal given that the graphitic shear does not appear to host significant uranium grade, but recognizes the risk remains until confirmed otherwise. Beyond this risk, the QP has not identified any other significant risks or uncertainties that could reasonably be expected to affect the reliability or confidence in the Mineral Resource.
- The QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource Estimate that are not discussed in this Technical Report.

Mining

The life-of-mine plan comprises 17 years of active mining operations followed by two years of stockpile processing. Mining is conducted at a consistent rate of 10.0 Mt of material annually during the first seven years and slowly diminishing from Y8 to Y17, with a LOM average strip ratio of 6.1:1. The mill feed grade fluctuates between 6.2% and 8.5% from Year 1 to Year 17, before reaching 4% to 5% during the processing of the stockpiled material from Year 18 to Year 19.

Processing

According to the test results from the preliminary test program for recovering the graphite from the Loki Deposit samples conducted by SGS Lakefield during 2025 and 2026, it is QP's opinion that the mineralization shows typical metallurgical response to the conventional flotation concentration method.

1.20 Recommendations

Mineral Resource Estimate

The Mineral Resource related recommendations are summarized below.

- Mineral resources are uncertain because of variability at all scales and sparse sampling. Geostatistical techniques can be used to quantify the uncertainty and the expected reduction of uncertainty in resources as a function of data spacing. The QP recommends that a drill hole spacing study be completed on the deposit to inform drill hole spacing for Indicated Mineral Resource classification. After completion of the drill hole study, definition drilling should be planned and executed accordingly. The drill hole spacing study is estimated to cost $45,000.
- The QP recommends future sample testing include impurity removal testing.
- Customer specifications for flake graphite are typically based on physical properties, particularly flake size, in addition to chemical characteristics. It is recommended that Abasca complete more comprehensive testing for graphite quality. The physical project testing is estimated to cost $150,000.

- Additional delineation drilling is recommended to further upgrade Inferred to Indicated Mineral Resources. Drilling required is estimated to include approximately 6,000 m for $2,500,000.

Mining

Several cost saving opportunities will be studied in the future. They include autonomous (driverless) haulage trucks, trolley assist (using electrical power to drive haul trucks up the pit ramp) and regenerative braking (haul trucks generating electricity and charging the site power grid while travelling downhill).

Processing

Further test work should be conducted to further optimize processing conditions, in particular, to conserve graphite flake size. Variability test work should be conducted to better understand metallurgical performances for the mineralization samples collected from different lithological and alteration zones and spatial locations.

A small scale demonstration flotation process plant is planned to be engineered and built to demonstrate the forecasted economical and technical viability of the Project.

Feasibility Study

This Report shows that the Project is considered to be technically and economically viable as a long-life graphite development project. The Project should be advanced toward a Feasibility Study. The costs associated with all the recommended work are summarized in Section 26.

2.0 INTRODUCTION

2.1 Overview

The Project is currently 100% owned by Abasca Resources Inc. (Abasca). Abasca is a Canadian exploration company, primarily engaged in the acquisition, evaluation, and development of uranium and graphite properties. Abasca is listed on the Toronto Stock Exchange Venture with the symbol ABA.V. Abasca commissioned Tetra Tech Canada Inc. (Tetra Tech) to prepare an independent technical report on the Key Lake South (KLS) Project (the Project), located in Saskatchewan, Canada, in accordance with NI 43-101 Standards of Disclosure for Mineral Projects. The purpose of this Technical Report is to summarize the results of a Preliminary Economic Assessment (PEA) of the Project and updated Mineral Resource Estimate announced by Abasca on July 14, 2026. The report sections relevant to geology and the mineral resource estimate were provided by Understood Mineral Resources Inc. (UMR).

Description and location of the Project are presented in Section 4.0 of this Technical Report.

The Report includes the Mineral Resource Estimate of the Project that was previously disclosed in the NI43-101 Technical Report titled “Technical Report On The Key Lake South Project with updated Mineral Resource Estimate for the Loki Flake Graphite Deposit, Saskatchewan, Canada.” with the effective date of April 23, 2026 which was filed on SEDAR on August 27, 2026.

The Qualified Persons (QPs) that authored the Technical Report are independent of Abasca and the Project.

2.1.1 2026 PEA Press Release

The results of the 2026 PEA were disclosed in Abasca’s press release dated August 19, 2026, in accordance with CIM Guidelines and NI43-101. This report is filed in support of the disclosure of the 2026 PEA results.

2.1.2 2026 MRE Press Release

An updated Mineral Resource Estimate (“MRE”), effective date April 23, 2026, was disclosed in the NI43-101 Technical Report titled “Technical Report On The Key Lake South Project with Updated Mineral Resource Estimate for the Loki Flake Graphite Deposit, Saskatchewan, Canada.” with the effective date of April 23, 2026 which was filed on SEDAR on August 27, 2026, in accordance with CIM Guidelines and NI43-101 and replaces the previous MRE with an effective date of April 10, 2025.

2.2 Sources of Information

The key information sources for this report were:

- Documents referenced in Section 3.0 (Reliance on Other Experts),
- Inspection of the Project as detailed above and in Section 12,
- Drill data, geologic models, cross sections, and other geologic data that were transferred to the QP via a data sharing platform in the first and second quarter of 2026,
- Additional information provided by Abasca personnel or consultants where required,
- Documentation and other sources of information listed in Section 27 including from the public domain.

2.3 Effective Dates

The overall effective date of the Report is August 19, 2026.

The effective date of the Mineral Resource estimate is April 23, 2026.

2.4 Qualified Persons

The names of the Qualified Persons (QPs) of this report and their QP certificates, describing the QPs' education, membership in recognized professional associations, independence from Abasca, and relevant work experience, are included in Section 28.0.

2.5 Personal Inspections

The following QPs conducted site visits of the Project:

- Mr. Matt Batty, MSc, P. Geo, of UMR, visited the Project from January 28 to 29, 2025. During Mr. Batty's site visit, he examined recent drill core, confirmed collar locations, reviewed drilling, logging, sampling, analytical and QA/QC procedures, and reviewed site facilities.
- Dr. Sabry Hafez, P.Eng. PhD., of Tetra Tech, visited the Project on June 18, 2026 and conducted a general project site overview in the proposed open pit and rock storage areas.

3.0 RELIANCE ON OTHER EXPERTS

This report has been prepared by the QPs for Abasca. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to the QPs at the time of preparation of this report
- Assumptions, conditions, and qualifications as set forth in this report
- Data, reports, and other information supplied by Abasca and other third-party sources

The Authors carefully reviewed the available Project information and believe that the information used to prepare this Technical Report is valid and appropriate, considering the purpose of the current Technical Report, and therefore accept responsibility for the information other than as described below.

The Authors are not qualified to provide an opinion or comment on issues related to legal agreements, mineral titles, royalties, taxation, or environmental matters.

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

3.1 Mineral Tenure, Surface Rights

For the purpose of this report, the geology QP relied on Abasca to provide all pertinent information concerning the legal status of Abasca, as well as current legal title information for the mineral claims and material environmental information that relate to the Project. Mr. Brian McEwan, Vice President of Exploration and Development for Abasca, provided an internal memorandum summarizing the status of claims as of the effective date of this report. The memorandum is titled 'Abasca KLS Mineral Claims' and dated April 8, 2026. The relevant information related to these matters is summarized in Section 4 of this Technical Report.

The Authors did not attempt to verify the legal status of the claims that comprise the Project. However, the mineral claim status is available online and was confirmed by the author via the Government's Mineral Administration Registry Saskatchewan (MARS) website as of the effective date of this Report.

3.2 Royalties and Incumbrances

Abasca holds a 100% interest in the Project. There are no additional royalties, back-in rights, or encumbrances on the project or potential production, other than the standard royalties due to the Government of Saskatchewan.

3.3 Taxes

Dr. Sabry Hafez, P. Eng., PhD., has not independently reviewed the taxation information.

Dr. Sabry Hafez, P. Eng., PhD., has fully relied upon the taxation information derived from the experts retained by Abasca, as applied in Section 22.5 of the Report.

3.4 Environmental Studies, Permitting and Social or Community Impact

Mr. Hassan Ghaffari, P. Eng., MASc. has fully relied upon the information regarding Environmental Studies, Permitting and Social or Community Impact of the Project from the experts retained by Abasca, as applied in Section 20.0 of the Report.

4.0 PROPERTY DESCRIPTION AND LOCATION

The Project is located approximately 220 km north-northwest of the town of La Ronge (Figure 4-1). The Key Lake Mine, a former uranium producer and the site of the Key Lake mill which processes Cameco's McArthur River ore, is located 15 km northeast of the Project. The Project is centered at UTM NAD 83 Zone 13N grid coordinates 442500E, 6322500N, on NTS map sheets 74-G-01 and 74-H-04.

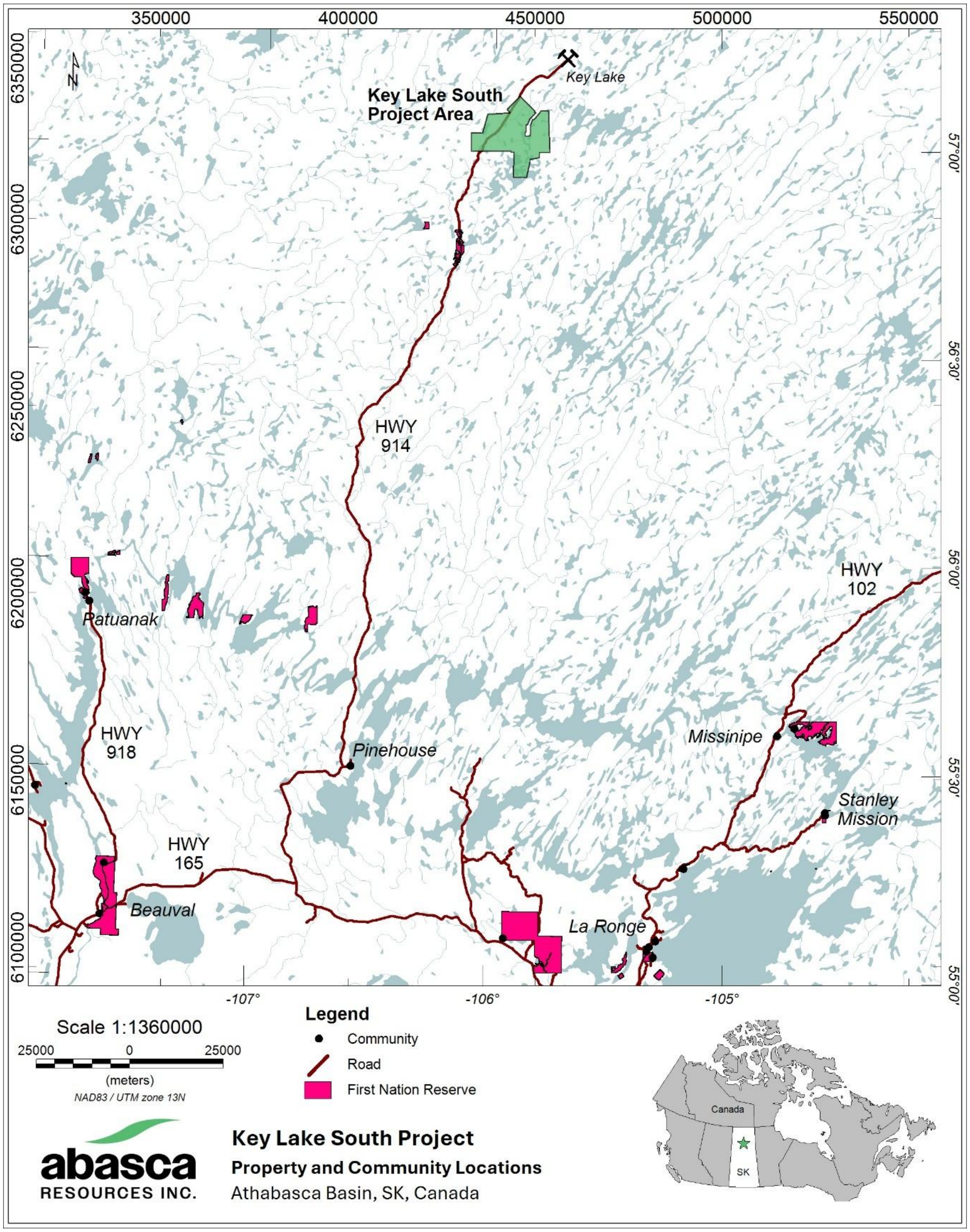


Figure 4-1: Key Lake South Project Location Map

4.1 Land Tenure

Under Saskatchewan law, claims are staked through an online registry. The map-designated coordinates of the claims are the legal limits of said claims. The physical limits can be verified by consulting the Government's Mineral Administration Registry Saskatchewan (MARS) website.

The Project comprises twelve contiguous mineral dispositions, covering 23,974 hectares over National Topographic System (NTS) areas 74G01, 74A13, and 74H04 (Figure 4-2). A summary of the tenure information, as extracted from the MARS website, is presented in Table 4-1. All claims are 100 percent owned by Abasca and are in good standing with expiry dates varying between April 10, 2031 and September 12, 2031. The total annual assessment requirement is $25 per hectare, totaling $599,357. The total area of the Project was reduced at the beginning of 2026 from 23,977 Ha to 23,974 Ha due to the Saskatchewan government adjusting for partial cell claims created when the MARS system was first introduced. The change in size has not materially affected or impacted the mineral occurrences or prospects of the Project.

Table 4-1: Mineral Dispositions

Mineral Disposition	Issuance Date	Size (Ha)	Annual Assessment (CND)	Expiration Date*
S-112088	October 18, 2011	1,754	$43,845.70	September 12, 2031
S-112288	October 17, 2011	1,175	$29,375.88	August 15, 2031
S-112289	October 17, 2011	4,143	$103,579.40	August 15, 2031
S-112290	October 17, 2011	4,965	$124,116.15	August 15, 2031
S-112291	October 17, 2011	4,901	$122,520.05	August 15, 2031
S-112430	June 14, 2012	194	$4,857.25	April 10, 2031
S-112431	June 14, 2012	175	$4,370.85	April 10, 2031
S-112432	June 14, 2012	148	$3,694.13	April 10, 2031
S-112433	June 14, 2012	101	$2,516.13	April 10, 2031
S-112434	June 14, 2012	344	$8,599.18	April 10, 2031
S-112435	June 14, 2012	2,810	$70,244.25	April 10, 2031
S-112436	July 31, 2012	3,266	$81,638.08	June 12, 2031
	Total	**23,974**	**$599,357.05**	

*Expiration Date refers to the Review Date after all available banked expenditure credits have been used.

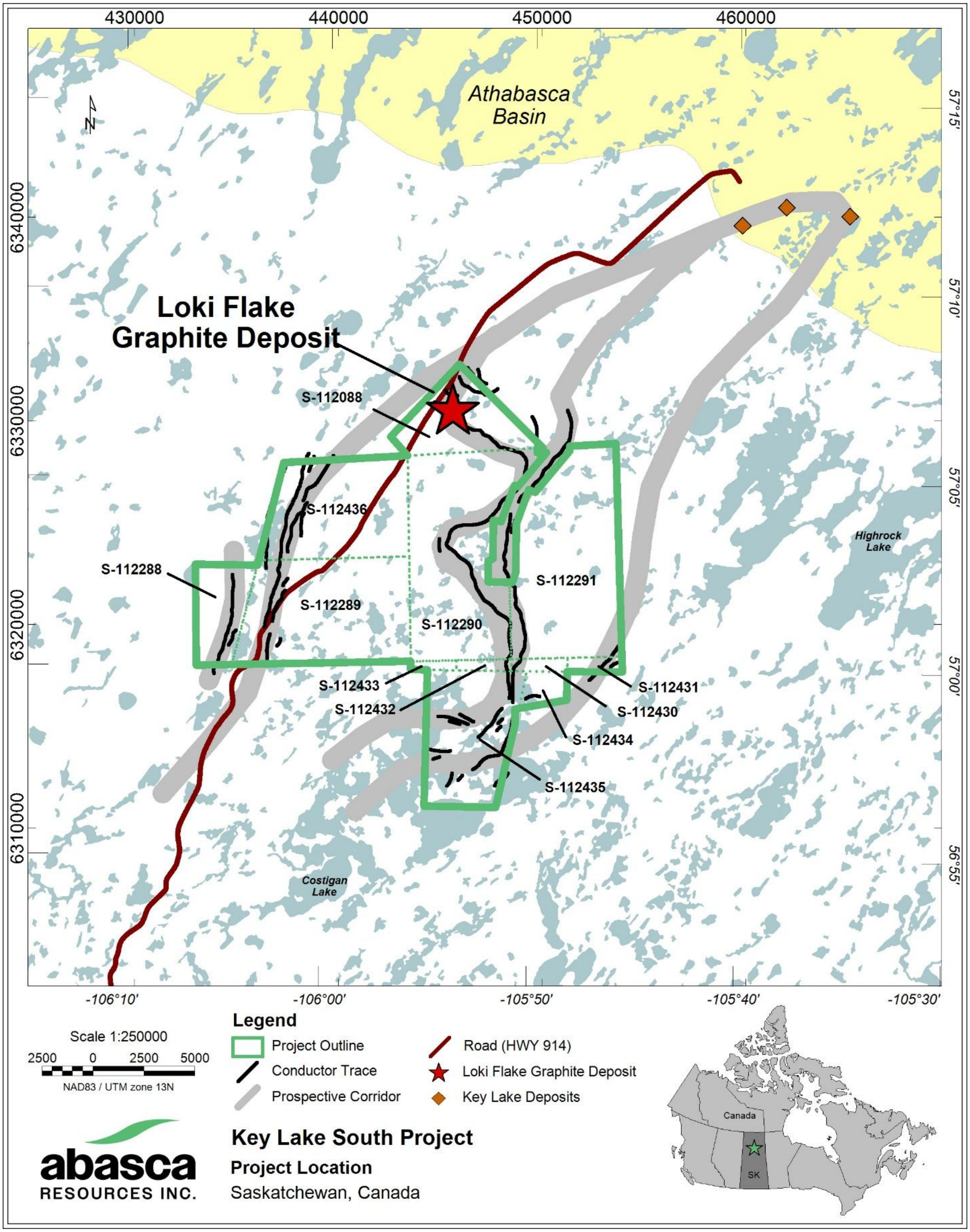


Figure 4-2: Mineral Dispositions of the Key Lake South Project

4.2 Mineral Rights

In Canada, natural resources fall under provincial jurisdiction.

In Saskatchewan, mineral resources are owned by the Crown and managed by the Saskatchewan Ministry of Energy and Resources using the Crown Minerals Act and the Mineral Tenure Registry Regulations, 2012. Staking for mineral dispositions in Saskatchewan is conducted through the online staking system, MARS. These dispositions give the stakeholders the right to explore the lands within the disposition area for economic mineral deposits.

Mineral dispositions for the Project were staked between 2011 and 2012, prior to the implementation of MARS. Accordingly, ground staking methods were employed by a private company (101159623 Saskatchewan Ltd. or "SaskCo") to secure these dispositions, which have been held and explored by them until the end of 2022. The project was then transferred to Abasca Resources Inc. by reverse takeover of a TSX Venture listed corporation and is 100%-owned.

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. Surface facilities constructed in support of mineral extraction require a surface lease. Surface leases have 33-year maximum terms and are also renewable.

Abasca does not have surface rights associated with the mineral claims that comprise the Project.

4.3 Royalties and Other Encumbrances

Abasca holds a 100% interest in the Project. There are no additional royalties, back-in rights, or encumbrances on the Project or potential production, other than the standard royalties due to the Government of Saskatchewan.

4.4 Permitting

Mineral exploration on land administered by the Ministry of Environment requires that surface disturbance permits be obtained prior to exploration activities, including 1) Crown Land Work Authorization, 2) Aquatic Habitat Protection Permit, 3) Temporary Work Camp Permit, and 4) Forest Product Permit. Additionally, the Saskatchewan Mineral Exploration and Government Advisory Committee (SMEGAC) have developed the Mineral Exploration Guidelines for Saskatchewan to mitigate environmental impacts from industry activity and facilitate governmental approval for such activities. Applications to conduct exploration work need only to address the relevant topics of those listed in the guidelines. The QP understands that Abasca has all required permits to conduct its proposed mineral exploration.

The Saskatchewan Ministry of Environment issued permits (ENV# 24-13-M0305, 25-13-M0005) authorizing Abasca to carry out its exploration activities and to construct and maintain a temporary work camp in the southwestern part of the project area.

For other permitting requirements for the Project, as it relates to mine development, refer to Section 20.3.

4.5 QP Comment

There are no known environmental issues or liabilities potentially affecting the Project and all the proper permits required to conduct exploration activities on the Project for all exploration campaigns have been obtained.

The authors are not aware of any other significant factors or risks that would affect access, title, or the ability to perform work on the Project.

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

5.1 Accessibility

The Project is situated near the southeastern margin of the Athabasca Basin, approximately 15 km southwest of the Key Lake uranium mill and 185 km north of Pinehouse, Saskatchewan (see Figure 4-1 in Section 4). The Project is accessed by provincial highway 914 that goes through the western part of the project area. The western region of the project area can be accessed by historical trails whereas the central and eastern regions are best accessed by helicopter.

5.2 Climate

The Project is located on the margin of the Athabasca sedimentary basin region which is coincident with the Athabasca Plain Ecoregion and Boreal Shield Ecozone. The Athabasca Basin to the north of the Project covers about 100,000 square kilometers in northern Saskatchewan and the northeastern corner of Alberta, Canada. Annual temperature of the basin and surrounding region generally ranges from -40°C in the winter to 35°C in the summer. The mean temperatures for January are -25°C and 18°C for July. Annual average rainfall is 225 mm and snowfall 2,150 mm. Lake ice thaws in late April and returns in late October.

Exploration activities can be carried out year-round, however access is limited to the Project during the months of April and May due to the abundance of lakes, muskeg and wet conditions that occur during the spring thaw.

5.3 Local Resources

The nearest communities are Pinehouse and La Ronge are respectively located approximately 185 km and 400 km south of the project area by road. Supplies, groceries, fuel, and accommodations can be found in these communities. Accurate Industries operates a stationary modular camp is located within 10 km North of the Project via Highway 914 and provides all necessary camp amenities.

5.4 Infrastructure

All infrastructure currently on the Project is non-permanent. The Government of Saskatchewan requires a surface lease to be issued for all permanent structures. There is access to fresh water close to the Project and the hydroelectric grid is located near the Project within approximately 15 km of the Project which supplies electricity to the operations at Key Lake. The nearest wireless service via the signal from the Key Lake mill. Standalone wireless service is available in some parts of the project area, particularly in the north at the Project; however, a signal booster can be used in other parts of the project area.

5.5 Physiography

Topography is generally flat lying with low ridges and hills reaching up to several tens of metres in relief. A strong north-easterly structural grain is evident in the topography. The area is characterized by numerous lakes, creeks, and ponds. Soil thickness on the ridges and hills is minimal and bedrock exposure is scarce but is found within recent burns and along large ridges. Rare outcrops are typically clustered and covered in lichens and moss. The vegetation is characteristic of subarctic tundra, dominated by small coniferous and deciduous trees, as well as shade-tolerant shrubs with grasses and feathermoss carpets at their base. Low-lying areas between hills and ridges consist of poorly drained muskeg swamps with scattered tamarack and black spruce.

6.0 HISTORY

6.1 Prior Ownership

The claims defining the Project were originally staked in 2011 and 2012 by private company 101159623 Saskatchewan Ltd. ("SaskCo"). These claims partially overlap or wholly encompass claims previously held by various operators between 1969 and 2010, including Yukon Geothermal Company Ltd., Inexco Mining Co., Scurry-Rainbow Oil Ltd.–Western Mines Ltd., Getty Minerals Company Ltd., E&B Explorations Ltd., Uranerz Exploration & Mining Ltd., Minatco Ltd., International Uranium Corporation, and Denison Mines Corporation. The past operators did not have title to the claims when SaskCo completed the original staking.

In December 2022, Abasca acquired a 100% right, title and interest in the mineral claims that comprise the Key Lake South Project ("KLS") located in the southeastern Athabasca Basin Region in northern Saskatchewan, Canada from SaskCo for 25,639,288 common shares of the Company (the "Transaction"). The Transaction constituted a "reverse takeover" ("RTO") pursuant to the policies of the TSX Venture Exchange.

6.2 Exploration and Development History

The QP has not verified the information of the adjacent properties, and that the information of the adjacent properties is not necessarily indicative of the mineralization on the project that is the subject of the technical report. The reviewed historical results from 1969 to 2010 have not been verified by the author and there is a risk that any future confirmation work and exploration may produce results that substantially differ from the historical results. These results are considered relevant to assess the mineralization and economic potential of the project, not to the Mineral Resource disclosed in this document.

The following compilation of historical exploration activities in the Key Lake South area are modified from Zhou et al. (2018).

Early regional exploration activities between the late 1960s and throughout the 1970s focused on ground prospecting, lake water geochemistry, and airborne radiometric and electromagnetic (EM) surveys as well as some shallow drilling. Although these early activities led to the discovery of the nearby Gaertner and Deilmann deposits that comprise Key Lake, no significant uranium mineralization was discovered over the current project area during that time.

In the early 1980s, Uranerz Exploration and Mining discovered radioactive pegmatites during ground prospecting surveys with values ranging from 0.02% to 22% U_3O_8 at Davies Creek, located on the current S-112088 claim, and in outcrops along Highway 914 on the current S-112289 claim with values of 0.18% to 0.94% U_3O_8. Radioactive sands and boulders ranging from 0.004% to 0.005% U_3O_8 (30 to 46 ppm U) and some radioactive gabbro and mafic dykes with values ranging from 0.025% to 0.861% U_3O_8 (200 to 7300 ppm U) were also discovered in the Twin Lakes area during this period.

In 2005, International Uranium Inc./Denison Mines Corp. (Denison) conducted ground HLEM and magnetic surveys at Twin Lakes, as well as a regional GeoTEM survey. A follow-up drill program of nine drill holes totalling 759 metres targeted a conductor at shallow depths. Drilling intersected massive graphite and massive sulphides but no anomalous radioactivity. Only a limited portion of the stratigraphy above the graphitic horizon was tested.

In 2006, Denison carried out an AeroTEM survey followed by HLEM surveys. Six holes were drilled along the Wollaston-Mudjatik transition to test a conductor located roughly parallel to and just west of Highway 914; Denison intersected some graphitic pelites that corresponded to the target conductors and some associated weak uranium mineralization in pegmatite. Two of the holes were drilled on the claim now held as S-112289. Denison still controls a large portion of this conductor system but a strike extent of this conductor of nearly 6 km occurs on the project's claim S-112289.

In the winter of 2009, Denison conducted HLEM and magnetometer surveys in the Graham Lake and Campbell Creek-Zimmer Lake areas and identified conductive units in both these areas.

SaskCo completed geological mapping and geochemical surface sampling in 2014 as well as a project-wide airborne HeliFalcon gravity survey, and together with historical data, culminated in 14 defined target areas prospective for uranium mineralization. Two target areas, Mustang and Campbell, were drilled in 2016 for a total of 4,553 metres. Additional geophysical compilation and reprocessing was done in 2022, including 3D inversions of the airborne magnetic data and the 2014 HeliFalcon gravity survey.

Previous exploration in the area is summarized in Table 6-1, highlighting the most relevant historical exploration, organized by company and year. Report numbers refer to those in the Saskatchewan Mineral Assessment Database (SMAD). The drilling completed on the project prior to 2016 is summarized in

Table 6-2. Drilling completed in 2016 and later is described in Section 10.

Table 6-1: Previous Work Conducted on the Project

Company	Report Year	Work Year(s)	Work Preformed	Assessment Report No.
Yukon Geothermal Company Ltd.	1969	1969	Airborne Radiometric Surveys and Ground Prospecting	74G01-0004
Inexco Mining Co.	1973	1973	Airborne Radiometrics, Geological and Geochemical Survey.	74H-0011
Scurry – Rainbow Oil Ltd – Western Mines Ltd.	1976	1976	Prospecting, Mapping, Lake Water and Sediment Sampling, VLF-EM and Magnetic Surveys	74H04-0027
Getty Minerals Company Ltd.	1976	1976	Airborne Radiometric Survey, Prospecting, Geologic Mapping, Lake Water and Sediment Sampling, VLF-EM, Magnetic and Track-Etch. Surveys.	74H04-0033
E&B Explorations Ltd.	1977	1977	Prospecting, VLF-EM, Magnetics, Lake Sediment Sampling	74G01-0014
Uranerz Exploration & Mining Ltd.	1977	1977	Prospecting, Lake Water and Sediment Sampling, Muskeg Geochemistry, Track ETCH.	74H-0020
Denison Mines Ltd.	1977	1977	Airborne EM, Magnetics, Drilling, EM, Radiometric Lake Sediment Sampling.	74H04-0025
Getty Minerals Company Ltd.	1977	1977	VLF-EM, Magnetic, Geological Mapping, Lake Sediment, Radon Survey, Prospecting, Stripping and Trenching.	74H04-0034

Company	Report Year	Work Year(s)	Work Preformed	Assessment Report No.
Scurry – Rainbow Oil Ltd – Western Mines Ltd.	1977	1977	Ground EM and Magnetic Surveys, Deep Over Burden Drilling and Analyses	74H04-0035
E&B Explorations Ltd.	1978	1978	Drilling (78-S-1 to 78-S-6)	74G01-0015, 74G01-0021
E&B Explorations Ltd.	1978	1978	EM, and Magnetics	74G01-0016
Denison Mines Ltd.	1978	1978	Drilling, Ground EM, Max/Min, Magnetics, Radiometric Surveys, Prospecting, Geological Mapping, Lake Sediment Sampling	74H04-0028, 74H04-0029
Scurry – Rainbow Oil Ltd – Western Mines Ltd.	1978	1978	Diamond Drilling (CS-78-01 to CS-78-15), Airborne EM	74H04-0048
Scurry – Rainbow Oil Ltd – Western Mines Ltd.	1978	1978	Prospecting, Geological Mapping, VLF-EM	74H04-0053
E&B Exploration	1979	1979	Drilling, EM, and Magnetics, Overburden Drilling	74G01-0028
Getty Minerals Company Ltd.	1979	1978 – 1979	Ground EM, Magnetics	74H04-0044
Scurry – Rainbow Oil Ltd – Western Mines Ltd.	1979	1979	Diamond Drilling (CS-79-01 to CS-79-33)	74H04-0058
Getty Minerals Company Ltd.	1980	1979 – 1980	Drilling: SB-8-1 to SB-8-05	74A13-0033
Getty Minerals Company Ltd.	1980	1980	Drilling: HRN-80-101 to HRN-80-106	74H04-0063
Getty Minerals Company Ltd.	1980	1977 – 1980	Summary Report (Geology, Geophysics and Drilling Activities)	74H04-0064
Uranerz Exploration & Mining Ltd.	1981	1981	Doppler Satellite Survey	74G-0009
Scurry – Rainbow Oil Ltd – Western Mines Ltd.	1981	1981	Diamond Drilling (CS-81-01 to CS-81-08)	74H04-0060
Uranerz Exploration & Mining Ltd.	1982	1982	Airborne INPUT EM and Magnetic Survey	74G01-0040
Minatco Ltd.	1984	1984	Prospecting, Mapping, Ground Radiometric survey, Petrography.	74B16-0045

Company	Report Year	Work Year(s)	Work Preformed	Assessment Report No.
Uranerz Exploration & Mining Ltd.	1984	1980-1984	Radiometric Prospecting	74G-0013
Uranerz Exploration & Mining Ltd.	1984	1980-1984	Assays and Analysis Certificates	74G-0014
Minatco Ltd.	1984	1983 – 1984	Drilling, Track ETCH, Prospecting, and Mapping	74G01-0032
Minatco Ltd.	1984	1984	Prospecting, Gridline Radiometric Survey, EM, Magnetics	74G01-0033
Uranerz Exploration & Mining Ltd.	1989	1989	Drilling, HLEM, TDEM, Boulder and Outcrop Sampling.	74H-0020
International Uranium Corp.	2005	2005	Diamond Drilling (KS05-01 to KS05-09)	74H04-0103
International Uranium Corp.	2006	2006	Diamond Drilling (KS06-10 to KS06-15)	74G01-0035
International Uranium Corp.	2006	2006	Max-Min, Magnetic	74G01-0039
International Uranium Corp.	2006	2004 – 2006	Airborne EM and Mag (AeroTEM)	74H04-0106
Denison Mines Corp.	2009	2009	Line Cutting and Ground Geophysics	74H04-0119
Denison Mines Corporation	2009	2009	Airborne EM and Mag (VTEM)	74H04-0121
101159623 Saskatchewan Ltd.	2014	2014	HeliFALCON Airborne Gravity Gradiometer Survey	MAW00529
			Geological Mapping Surveys	
			Soil / Lake Sediment Geochemistry Surveys and Soil Radon Surveys	

Table 6-2: Drillhole Locations of the 1978 to 2006 Programs at the Project

Drillhole ID	Easting	Northing	Elevation	Azimuth	Inclination	EOH
CS-78-07	449102.5	6328216	534	115	-45	128.02
CS-78-08	449076.5	6328054	535	85	-45	150.57
CS-78-09	449124.7	6327951	535	85	-45	126.8
CS-78-10	449127.9	6327391	539	85	-61	117.96

Drillhole ID	Easting	Northing	Elevation	Azimuth	Inclination	EOH
CS-78-11	449021.5	6328251	534	115	-45	123.75
CS-78-12	448383.0	6326687	540	130	-55	194.16
CS-78-13	446911.0	6324997	547	150	-60	123.75
CS-78-15	448965.5	6328200	535	115	-60	124.05
CS-79-01	449893.0	6328303	535	80	-45	121.01
CS-79-15	437074.0	6322924	566	270	-50	122.2
CS-79-16	437169.0	6322869	569	270	-50	122.5
CS-79-17	436367.0	6318360	575	290	-50	122.8
CS-79-18	436280.0	6318400	574	290	-50	122.8
CS-79-20	448551.0	6328233	535	85	-60	121.92
CS-79-21	448779.3	6328210	534	85	-55	145.39
CS-79-22	449036.5	6328301	535	115	-50	125.88
CS-79-23	448973.9	6328336	534	115	-50	122.83
CS-79-24	448753.5	6327526	537	85	-50	122.83
CS-79-25	448855.2	6327509	535	85	-50	122.83
CS-79-26	448950.2	6327525	539	85	-50	138.07
CS-79-27	449048.5	6327530	539	85	-50	138.07
CS-79-28	449403.0	6328533	530	137	-55	138.07
CS-79-29	449479.0	6328453	530	137	-55	138.07
CS-79-31	441114.0	6327107	571	310	-50	122.8
CS-79-32	443781.0	6325663	572	310	-50	122.8
CS-79-33	443853.0	6325623	577	310	-50	122.8
CS-81-01	446331.0	6322890	548	85	-45	120.7
CS-81-02	446224.0	6322880	553	85	-45	144.17
CS-81-03	446117.0	6322866	553	85	-45	134.32
CS-81-04	445991.0	6322852	555	85	-45	138.99
CS-81-05	445866.0	6322838	557	85	-45	148.13
DDH-78-S-1	436416.0	6320464	563	265	-60	41.76
DDH-78-S-2	436484.0	6320664	564	257	-60	99.7
DDH-78-S-3	436557.0	6320489	566	266	-60	124.05
DDH-78-S-4	436574.0	6320873	572	269	-60	94.49

Drillhole ID	Easting	Northing	Elevation	Azimuth	Inclination	EOH
DDH-78-S-5	436676.0	6320575	570	0	-90	169.77
KS05-01	444920.0	6324110	557	315	-70	107
KS05-02	445003.0	6324308	555	315	-70	20
KS05-03	445288.0	6324022	556	315	-70	83
KS05-04	445405.0	6324191	556	315	-70	80
KS05-05	446298.0	6324862	553	185	-70	101
KS05-06	446103.0	6324869	545	185	-60	104
KS05-07	445903.0	6324887	546	185	-70	89
KS05-08	446085.0	6324655	555	185	-70	89
KS05-09	445804.0	6323404	551	60	-70	86
KS06-10	436863.0	6320431	570	282	-65	275
KS06-11	437038.0	6321207	570	282	-65	239
KS06-12	437773.0	6324977	560	282	-65	335
KS06-13	438458.0	6324819	575	282	-65	524
KS06-14	438694.0	6327744	555	297	-65	380

6.3 Historical Resource Estimates

No historical resources estimate exist on the Project.

6.4 Past Production

No production has occurred on the Project.

7.0 GEOLOGY AND MINERALIZATION

7.1 Regional Geology

The Project lies just outside the current southeastern margin of the Athabasca Basin, Saskatchewan, Canada (Figure 7-1). The basin hosts the world's largest high-grade uranium deposits (Jefferson et al., 2007). The Athabasca Group consists of the late Paleoproterozoic fluviatile terrestrial quartz sandstone with local conglomerate that are flat-lying and relatively undeformed which is underlain in unconformity with the crystalline basement rocks of Archean to Paleoproterozoic (Card et al., 2007). The Athabasca Group sedimentary sequences are currently persevered in oval shape at surface with approximate dimensions of 450 km east-west by 200 km north-south and reaches a maximum thickness of approximately 1,500 m near the center.

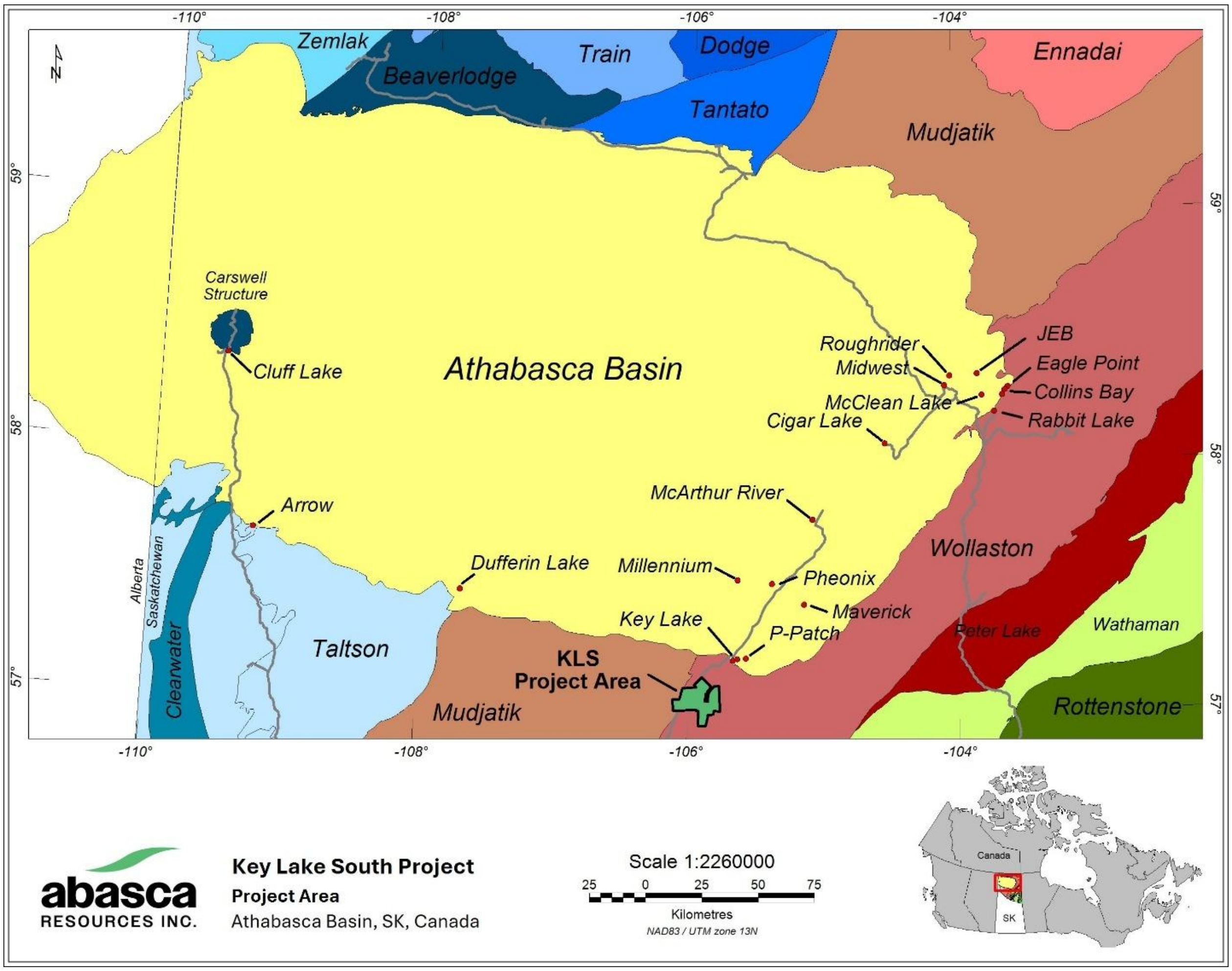


Figure 7-1: Geological Domains in Northern Saskatchewan in the vicinity of the Athabasca Basin and the Project

The basement rocks of the eastern Athabasca Basin and surrounding areas include highly deformed granitoid gneiss, migmatite and intrusions of the Mudjatik Domain which are recognized as mainly Archean aged. These rocks were uncomfortably overlain by the gneisses and migmatites of the Wollaston Domain which are recognized mainly Paleoproterozoic aged.

The Paleoproterozoic gneisses and migmatites were deformed, altered, and metamorphosed along with the adjacent Archean rocks during the Trans-Hudsonian Orogeny (locally 1.75-1.93 Ga, Card et al., 2006, 2018). The Trans-Hudsonian orogeny was followed by a protracted period of erosion, weathering, and the development of a paleo-weathering profile over the deformed and metamorphosed Archean and Paleoproterozoic rocks. The paleo-surface is defined by red hematitic and bleached alteration which transitions down to chlorite alteration and then to fresh basement rocks (Jerfferson et al., 2007). Locally, the gneisses were crosscut by uraniferous pegmatites and mafic intrusions.

7.2 Local and Project Geology

The eastern part of the project area is underlain by the 2.6 Ga Archean Zimmer Lake Granite, which extends northeast to Key Lake mine, where the Gaertner and Deilmann deposits lie on its northwestern flank. The far western part of the project is underlain by Mudjatik granitoid gneiss, presumed to also be of late Archean age. The central part of the project is underlain by metamorphic rocks of the Paleoproterozoic Wollaston Group. Meta-pelites and -arkoses predominate, with minor intrusions. Local hematite and chlorite altered gneiss outcrops were interpreted to be regolith (Ray, 1977). Preservation of the regolith suggests that these rocks were once overlain by Athabasca Group.

The Wollaston metamorphic rocks are associated with a broad magnetic low trending southwesterly from Key Lake across the project. Most of the known uranium deposits in eastern Athabasca Basin are associated with this magnetic low zone, which extends across the basin. Drilling confirms that the EM conductors found in the various ground and airborne surveys are primarily due to graphite. Locally, the metamorphic rocks are cut by uraniferous pegmatites and mafic intrusions.

Uranium exploration at the Project is targeting basement-hosted deposits such as Eagle Point, Millennium, Triple R, and Arrow. These deposits are found in Paleoproterozoic metamorphic rocks associated with reactivated graphitic fault zones. Mineralization occurs between a major lower reverse fault such as the Collins Bay Thrust at Eagle Point or the Mother Fault at Millennium and a narrow upper reverse fault such as the Eagle Point Fault or at Millennium the reverse fault associated with the graphitic marker.

The P-Patch deposit within Key Lake mine area is another example of a basement-hosted deposit associated with faulting and graphitic zones. The mineralization is situated within strongly bleached, argillitized, chloritized, and hematized pegmatoids, and pelitic to semi-pelitic gneisses immediately below a graphitic pelitic gneiss unit.

The Key Lake Mine, situated approximately 15 kilometres north of the project, produced over 200 million pounds of uranium at a grade averaging 2.3% U_3O_8 between 1983 and 1997. Key Lake comprised the Gaertner and Dielmann ore bodies, which were hosted in the both the Athabasca sandstone and the basement gneiss. The two ore bodies were controlled by the intersection of the Key Lake Fault Zone with the unconformity surface.

The qualified person has not verified the information of the adjacent properties, and that the information of the adjacent properties is not necessarily indicative of the mineralization on the project that is the subject of the technical report.

7.3 Mineralization

All significant uranium mineralization in the Athabasca Basin region is spatially related to the unconformity of the Athabasca Group and the underlying crystalline Paleoproterozoic and Archean basement. Within the eastern Athabasca Basin, most of the uranium occurrences are proximal to the boundary between the two litho-structural basement domains, the Archean Mudjatik Domain to the west and Paleoproterozoic Wollaston Domain to the east.

The generally higher grade of metamorphic phases of Archean rocks and their unconformable contact with the overlying Paleoproterozoic rocks disclose the region endured intensive deformation and metamorphism prior to the Trans-Hudson Orogeny. However, the preserved structural futures are mainly the results of the Trans-Hudson Orogeny during 1.93 to 1.75 Ga which severely over printed the previous structural results.

Five sets of structural phases were interpreted around the Key Lake mine, the northeast trending shear zones, the northeast trending folds, the northwest trending folds, the northeast trending brittle faults and the north to northwest trending brittle faults (Harvey and Bethune, 2007). It is assumed the first three phases of northeast trending compression structures were the result of the Trans-Hudson Orogen of locally about 1.93 to 1.75 Ga. The large granite intrusions and pegmatite veins that resulted from this could be the source of uranium mineralization. Following the tectonic conversion of deformation and metamorphism, the region experienced a long uplift and erosion period to form the unconformity base of the Athabasca Basin. Beneath the basal unconformity, red hematitic and bleached clay-altered regolith grades down through chloritic altered to fresh basement gneiss.

The northeast trending brittle faults which locally reactivated the early shear zones are directly related to uranium mineralization in the southeast of the Athabasca Basin and surrounding areas during the relatively stable tectonic setting of the Athabasca Group. The basement graphitic and altered gneisses constitutes weak zones during regional compressive deformation and late reactive extensive deformation. The graphitic and altered rocks in the reactive brittle fault zones can be interpreted as the channels to conduct deep crustal heat upward and to generate convection of hydrothermal fluids for alteration and mineralization. Reduced basement fluids mixed with oxidized basin fluids causing uranium precipitation in the unconformity, egress style with fluids upward, and in basement fractures, ingress style with fluids downward (Jefferson et al., 2007).

At the Project, graphite mineralization is localized along the brittle reactivated structures that are primary targets for uranium exploration. Typically, the graphite mineralization in the region is not greater than 15 m in thickness; however, drilling intersections observed in the Loki Flake Graphite Deposit in the northern part of the project area include wider intervals, up to 60 m.

7.4 Loki Flake Graphite Deposit

One mineral deposit has been discovered at the Project, the Loki Flake Graphite Deposit, which is characterized by a strongly graphitic shear zone. Graphite mineralization within the shear zone is generally bound by silicified metapelitic rocks of the Wollaston Domain and pegmatite, which are situated within the hanging wall over granitic gneiss that is presumed to be Archean. Due to the competent bounding host rocks, the contacts of the graphite mineralization are typically sharp with local gradational contacts, especially toward the lower contact. Local intervals of higher-grade graphite are commonly intersected in along the contacts of the mineralized zone, with the hanging wall being the most common to be higher-grade.

The Loki Deposit is a moderately dipping, southeast-striking graphitic fault zone that is orthogonal to the predominant orientation of northeast regional structures. The deposit, delineated as of April 2026, has an approximate strike-length of 1,700 m and extends from the lower contact of the overburden, approximately 55 m below surface, to 335 m below surface where it remains open.

8.0 DEPOSIT TYPE

8.1 Unconformity-Related Uranium Deposits

The Athabasca Basin hosts numerous high-grade uranium deposits (e.g., McArthur River and Arrow). Uranium deposits in the region have been described as one of two major types: egress-style and ingress-style unconformity-related (Jefferson *et al.*, 2007). The two styles are differentiated based on the location of mineralization (Figure 8-1). Egress-style deposits occur at and/or above the Athabasca-Basement unconformity, where as mineralization of ingress-style deposits occurs within the basement rocks. Due to this association, they are also referred to as sandstone-hosted or basement-hosted deposit.

Both styles feature common characteristics, including, 1) Proximity to unconformity surface between Athabasca Sandstone and basement rocks and 2) presence of re-activated shear- or fault-zones +/- graphite. Mineralization typically occurs as pitchblende and uraninite with lesser uranium-oxide minerals. Alteration around the egress-deposits is characterized by a relatively large halo of clay alteration and may extend several hundreds of metres, where as ingress-deposits have a very narrow halo that generally does not extend more than 50 metres. Both styles also feature a prominent redox boundary with strong hematite alteration distal to the mineralization and chlorite alteration proximal to the deposit.

At the Project, ingress-style deposits are the uranium exploration targets as the Project lies within 15 km south of the current margin of the Athabasca Basin.

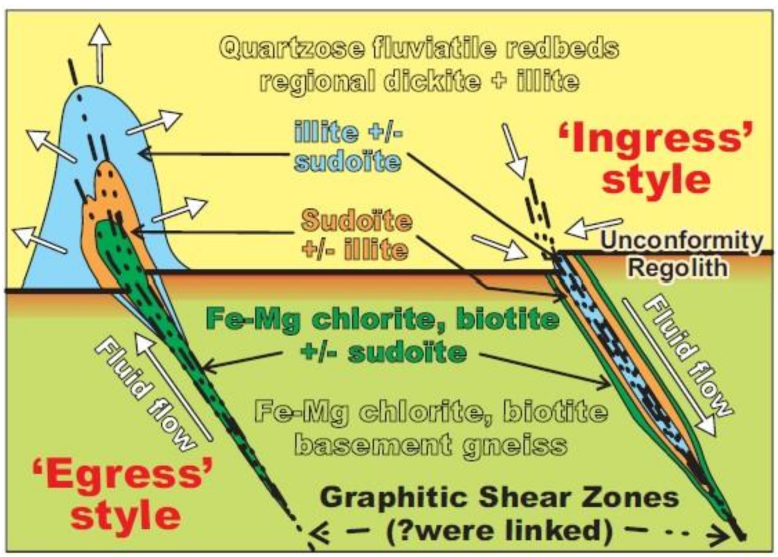


Figure 8-1: Unconformity-related Uranium Mineralization Model (Jefferson et al., 2007)

8.2 Flake Graphite Deposits

Natural flake graphite deposits are typically associated with metasedimentary rocks. The formation of the graphite has generally been described as the transformation of organic materials, accumulated during sedimentation, into graphite during amphibolite to granulite facies metamorphism. Graphite deposits are therefore typically stratabound and hosted in paragneisses and marbles (Harben and Kuzvart, 1996).

Graphite can also be accumulated in shear zones associated with carbon cycling at orogenic boundaries, like those near the Wollaston-Mudjatik geological Domain boundary. At these boundaries, biogenic carbon has been hydrothermally remobilized during the final stages of orogenesis into these shear zones (Toma *et al*., 2024), providing localized concentrations of natural flake graphite. At the Project, the Loki Deposit is interpreted to be such a shear zone, with concentrations of graphite mineralization bound within the zone between the competent basement rocks of the Wollaston Domain.

9.0 EXPLORATION

Between 2014 and 2016, SaskCo compiled historical data and conducted field programs at the Project which included geological mapping, surface sampling, as well as a project-wide HeliFalcon gravity survey, and a targeted MWH ground gravity survey. This work was done with the intent of targeting basement-hosted uranium deposits. Several prospective graphitic corridors were identified, and two target areas were selected for follow-up drill testing: Mustang and Campbell. These targets were drilled in 2016 and resulted in the discovery of what is now referred to as the Loki Flake Graphite Deposit.

Exploration conducted by Abasca at the Project has been exclusively drilling-related. A summary of SaskCo's exploration surveys that led to the discovery of the Loki Deposit from Zhou *et al*. (2018) is described below.

9.1 Airborne Geophysical Surveys

9.1.1 HeliFALCON Airborne Gravity Gradiometer Survey

In April and May 2014, CGG Ltd. conducted a high-sensitivity HeliFALCON® Airborne Gravity Gradiometer (AGG) survey over all 12 claims of the Project (CGG, 2014). The survey was carried out using a Eurocopter AS350-B3. A total of 25 production flights were flown for a combined total of 2,995 line-kilometres of data. Final data processing was completed from the Perth, Australia office of CGG Ltd. in June 2014.

The Project was flown in an east-southeast to west-northwest direction (117°/297°) with a traverse line spacing of 100 metres and a tie line spacing of 2,000 metres at 027°/207°.

During the survey of the Project, the helicopter was maintained at a mean height of 35 metres above the ground with a nominal survey speed of 150 kilometres/hour. Terrain clearance was provided by the radar altimeter at interval of 0.1s.

Terrain clearance for the survey averaged slightly above the nominal clearance of 35 metres, having a mean value of 45.3 metres across the survey area.

The survey specifications are summarized in Table 9-1.

Table 9-1: Airborne Gravity Survey Specification

Key Lake South Uranium Project Survey Specifications	
Total Delivered Kilometres (km)	2995
Clearance Method	Barometric
Nominal Terrain Clearance (m)	35
Traverse Line Direction (deg.)	117/297
Traverse Line Spacing (m)	100
Tie Line Direction (deg.)	027/207
Tie Line Spacing (m)	2000
Real-Time Differential GPS	Novatel OEMV L-band Positioning

Key Lake South Uranium Project Survey Specifications	
GPS Base Station Receiver	Novatel OEM4 L1/L2
Altimeters	King KRA405 Radar Altimeter
	Rosemount 1241M Barometric Pressure Sensor
Laser Scanner	Riegl LMS-Q140I-80

Terrain corrections were derived from the digital terrain model grid for every data point in the survey. A terrain density of 1.00 g/cm^3 was used to compute the terrain correction channels, which were then multiplied by the chosen correction density before being subtracted from the data.

9.1.2 Airborne Gravity Results

Because the Project is located outside of the Athabasca Basin, a standard density of 2.67 g/cm^3 was applied (a correction density of 2.00 g/cm^3 is typically selected for the Athabasca Basin's sandstone cover). The transformation of the results into Vertical Gravity (gD) and Vertical Gravity Gradient (gDD) was accomplished using two methods: Fourier domain transformation and the Method of Equivalent Sources.

After reviewing the results, the best representation of the gD and gDD data for the Project was selected as the Fourier result (density 2.67 g/cm^3), illustrated in Figure 9-1 (gD) and Figure 9-2 (gDD) with a structural interpretation overlay. The 2014 helicopter-borne high-sensitivity HeliFALCON® Gravity Gradiometer survey provided a new layer of project-wide data and allowed for an initial interpretation of major basement lithologies, structures, and possible zones of hydrothermal alteration.

Data interpretation by CGG Airborne (2015) identified small gravity highs and lows in the vicinity of EM conductors and proposed them as possible zones of hydrothermal alteration. The Gzz (and shallow depth slice of Gz) response was used to identify the small gravity anomalies thought to be derived from the small density changes due to alteration. CGG considers the high number of small Gzz anomalies to be a result of density variations within the pelite and psammite units, which may mask the subtle ±0.2 g/cc variations expected from silicification and de-silicification alteration.

The large gravity high seen in the vertical gravity results, as per Figure 9-1, and is interpreted as a dense rock mass, less amenable to the development of structures required to control uranium-rich hydrothermal fluids. The conclusion decreases the prospectivity of conductors present within the northern portion of claim S-112435 until further analysis suggests otherwise. On the other hand, the potential of the northeast-trending conductors occurring along the southern boundary of the gravity high is heightened by this interpretation, because a structure marking a competency contrast may provide a favorable conduit for hydrothermal fluids.

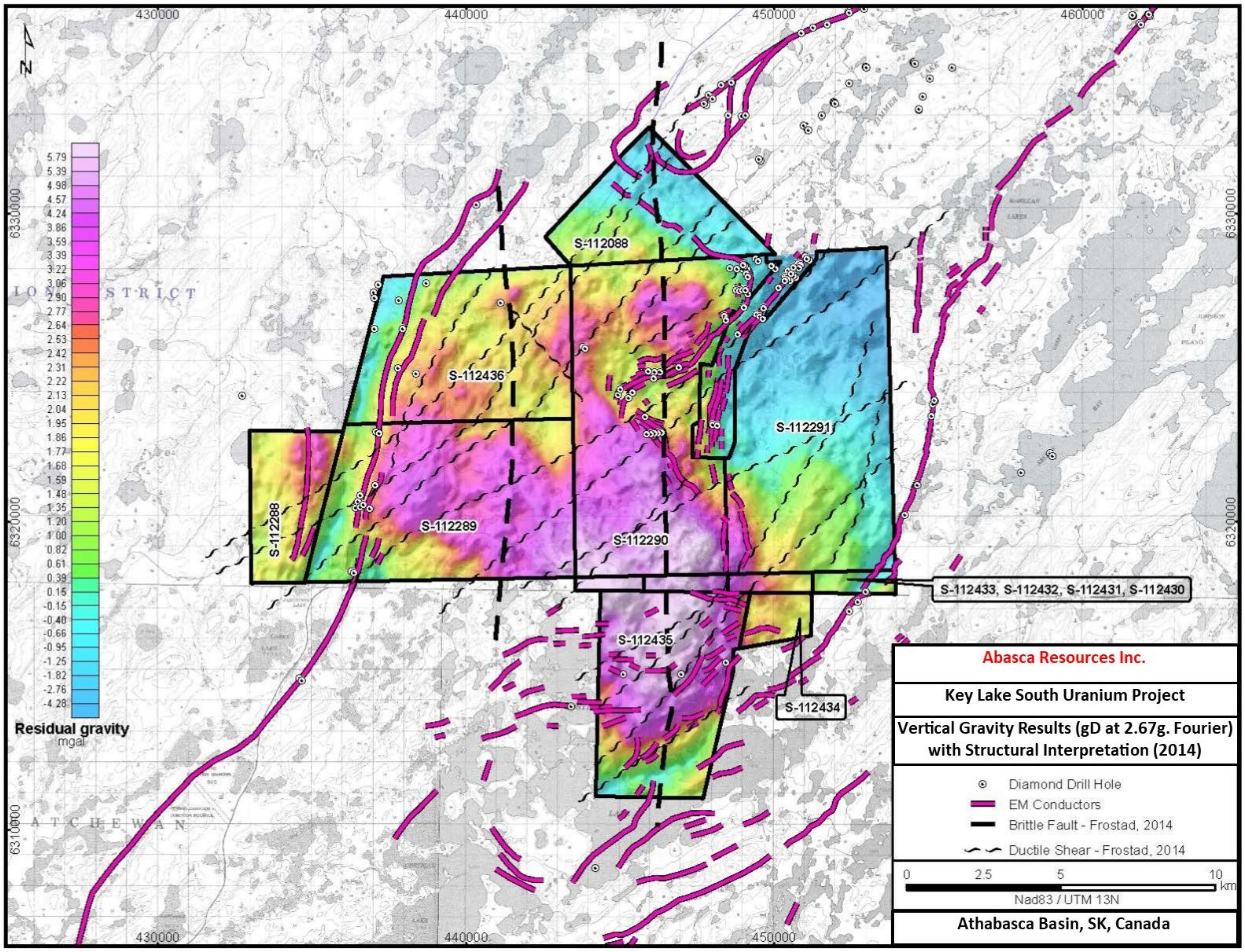


Figure 9-1: 2014 Vertical Gravity Results (gD) with Structural Interpretation (Zhou et al. 2018)

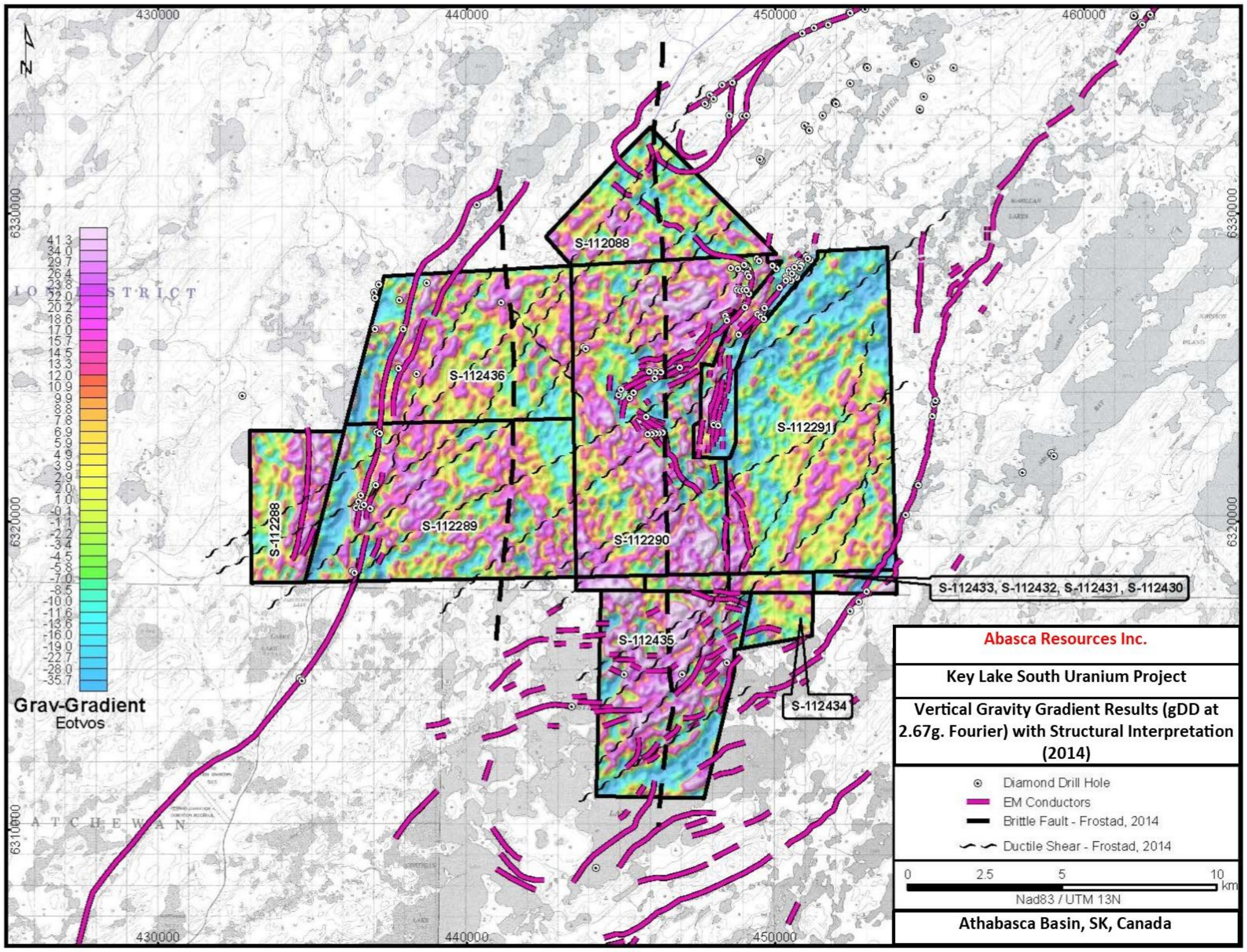


Figure 9-2: 2014 Vertical Gravity Results (gDD) with Structural Interpretation (Zhou et al 2018)

The gravity survey has also outlined gravimetric lows along areas of EM conductors not previously tested by diamond drilling. The gravity lows may be due to hydrothermal alteration, and they are highly prospective for uranium mineralization.

9.2 Ground Geophysical Surveys

9.2.1 2016 Ground Gravity Survey

In March 2016, MWH Geo-Surveys was contracted to carry out a ground gravity survey on three grids spanning the Mustang trend: Mustang South, Seager Central, and Seager North (Figure 9-3). A total of 1,027 stations at approximately 100 metres spacing were surveyed.

The following field procedures are taken from the logistics report (MWH, 2016):

LaCoste & Romberg gravity meters serial numbers 371, 697& 792 were used on this project. These meters are electronically nulled, are equipped with highly accurate electronic levels and feature one micro-gal resolution. Data is sent via a Bluetooth wireless link to a hand-held field PC running proprietary GControl gravity data logging software. GControl collects a gravity reading sample every 2 seconds and subsequently averages the collected samples to mitigate the effects of high frequency noise caused by wind and ice motion. All gravity readings were taken within loops to and from a gravity base established at the Abasca campsite. To determine the absolute gravity value of the new

Abasca base, multiple ties were made between the new base and through a series of intermediate sites, the Canadian Gravity Standard Network base in LaRonge (base# 9547-1977; value: 981,380.330).

A total of 971 unique stations and 56 repeats (not including base ties) were collected on 3 grids; Mustang, Seager Central and Seager North during 9 survey production days. Access to gravity sites was by snowmobile and on foot.

The Bouguer slab gravity at a density of 2.5 was used for this project.

In 2016, Bingham Geoscience conducted an interpretation of the 2016 ground gravity survey (Figure 9-4, Figure 9-5, Figure 9-6) that was conducted in three areas of the Mustang trend. Residual gravity lows were detected in the Mustang South, Seager Central, and Seager North sectors. These gravity low anomalies may be indicative of zones of alteration, clay development, and potential uranium mineralization along electromagnetic (EM) conductors. Anomalies MS1, SC3, and SN1 have coincident (or nearby) well-defined reinterpreted MaxMin conductors and are ranked as priority targets.

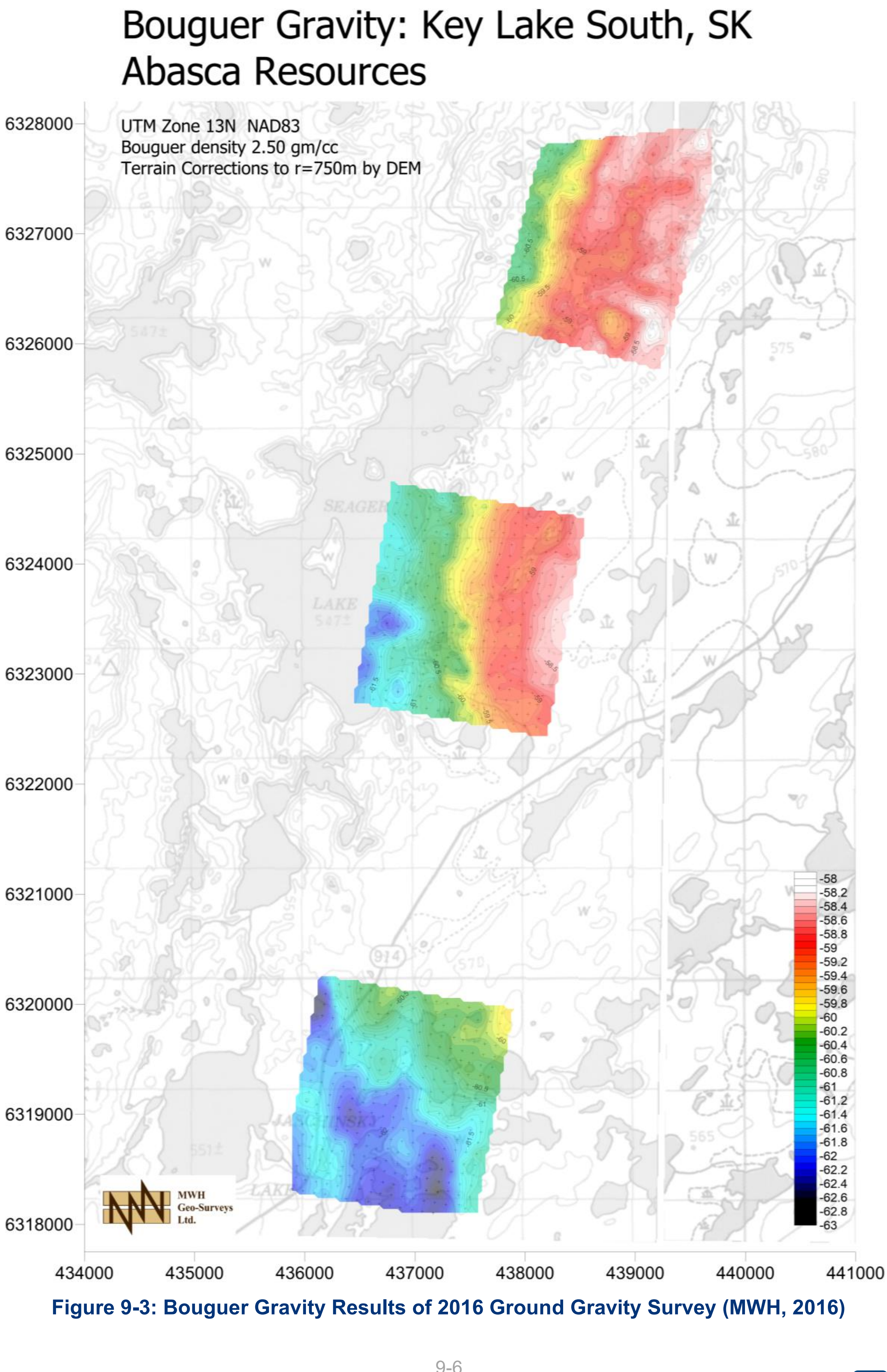


Figure 9-3: Bouguer Gravity Results of 2016 Ground Gravity Survey (MWH, 2016)

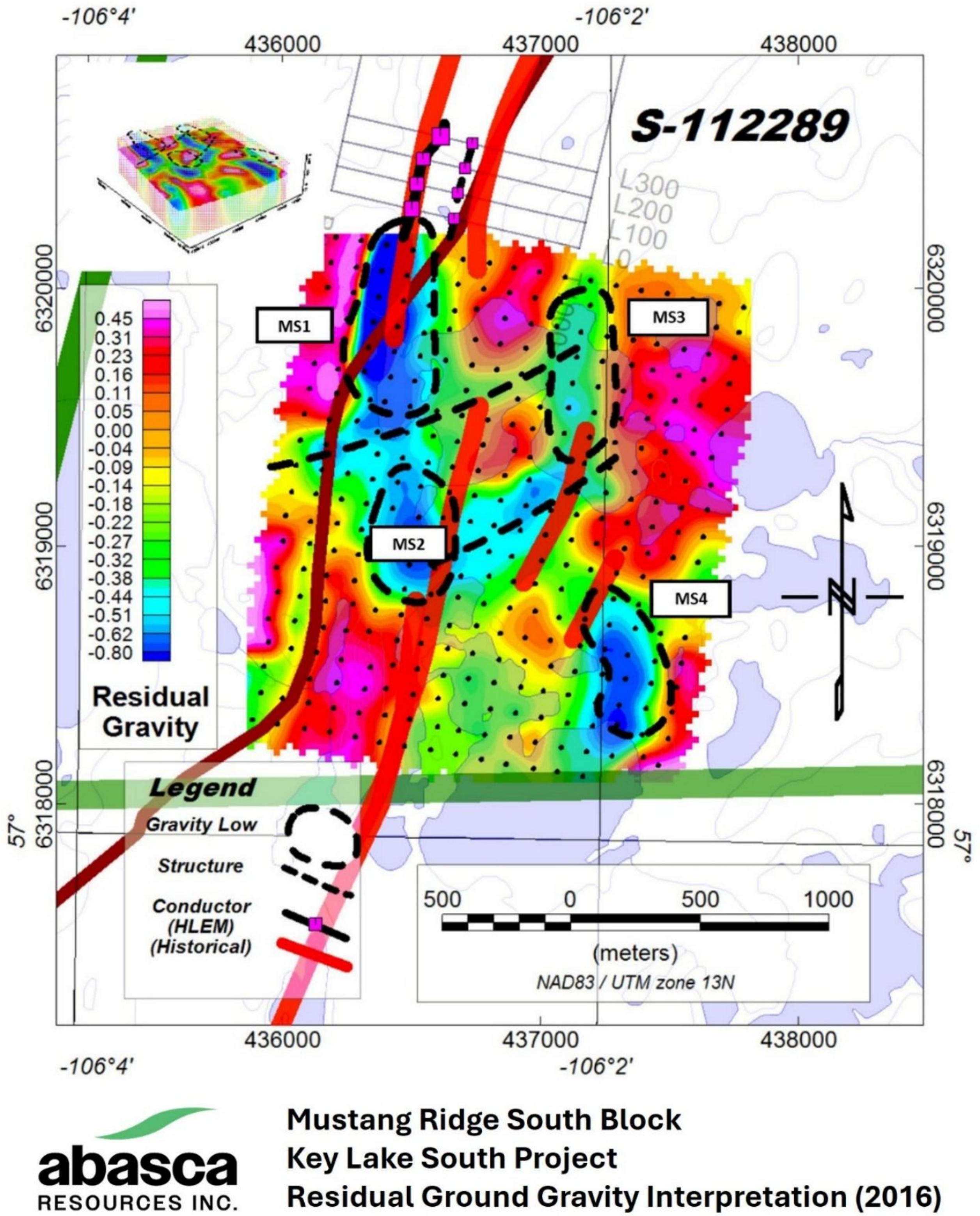


Figure 9-4: Residual Gravity Interpretation, Mustang Ridge South Block of the Project (Modified from Zhou *et al.,* 2018)

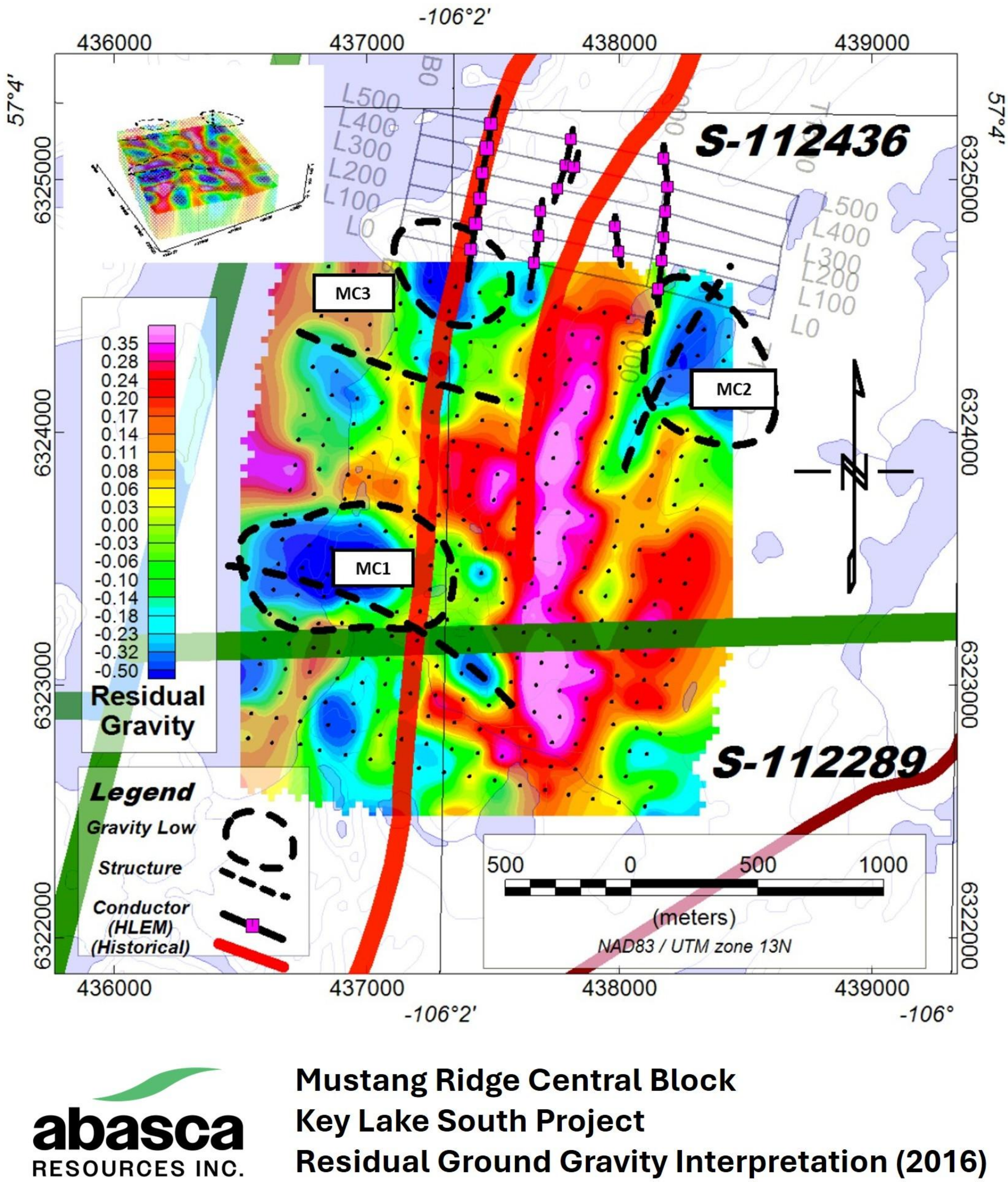


Figure 9-5: Residual Gravity Interpretation, Mustang Ridge Central Block of the Project (Modified from Zhou et al., 2018)

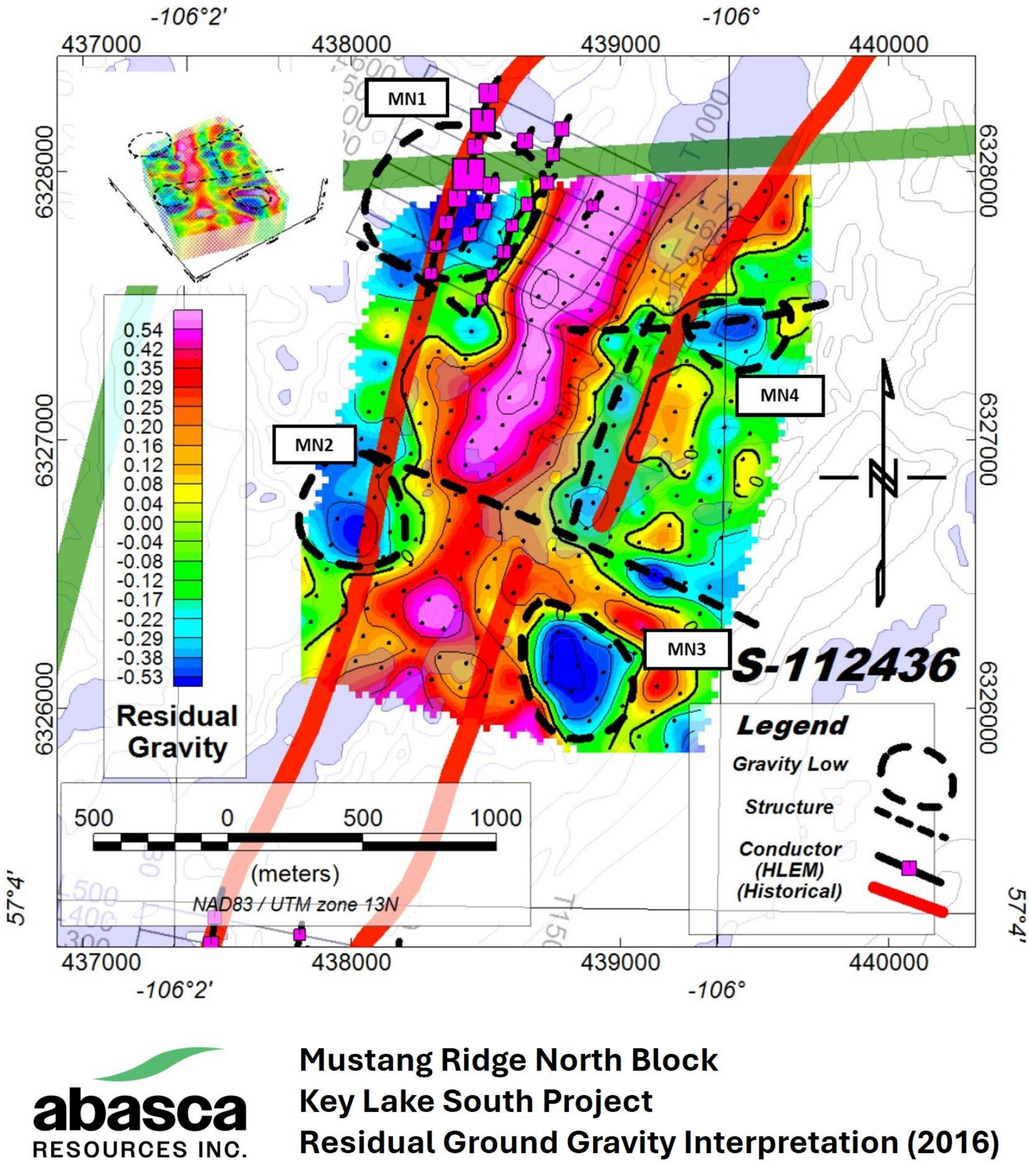


Figure 9-6: Residual Gravity Interpretation, Mustang Ridge North Block of the Project (Modified from Zhou *et al*., 2018)

9.3 Geochemical Surveys

9.3.1 2014 Soil Sampling Program

A soil survey was conducted in 2014; a total of 804 samples were collected in two separate areas of the project:

- A grid at the north end, subdivided into Area A (on claim S-112088) and Area B (on claims S-112088 and S-112290), covered a zone where an EM conductor, untested by diamond drilling, coincides with an airborne gravity low and a magnetic low (Figure 9-7 and Figure 9-8).
- A grid in the southwest corner of the project, Area C, is located on claim S-112289 (Figure 9-9 and Figure 9-10). This grid covered a structurally complex area, as evidenced from geophysics, where two parallel north-south trending EM conductors display a break/offset of 400 to 700 metres.

Sampling grids were designed to cover portions of a previously established cut and picketed grid having a line-spacing of 200 m and stations marked every 50 m. Sample location coordinates were uploaded into the GPS units that were used to navigate to the sample sites. The samplers chose a suitable sample location close to the GPS coordinates for that sample site and collected the black A1 organic soil layer by hand or with a spade. The samples were collected in Kraft paper sample bags and labelled with the predetermined sample ID.

The A1 horizon was occasionally just below the plant litter and could be easily scraped up. Elsewhere, the A1 horizon was most easily accessed by pulling up the surface vegetation by hand and collecting the black soil at the root base. Where identified, the A1 horizon varied in thickness from 1 to 6 cm. At lower elevations where a distinct A1 soil horizon could not be identified, peat samples were taken in lieu of soil. All samples were described for colour, sand content, and the percentage of peat.

All samples were sent to Saskatchewan Research Council (SRC) in Saskatoon, SK for ICP-MS analysis. Samples were air dried, mortared then sieved to 180 microns. Initial samples were analyzed after both partial and total digestion. Partial digestion was suggested as a means of avoiding interference that arise from conducting ICP-MS on totally digested samples. For partial digestion, a 0.250 g pulp was digested with 2.25 ml of 8:1 ultrapure HNO_3:HCl for 1 hour at 95 C. For total digestion, a 0.125 g pulp was gently heated in a mixture of ultrapure HF/HNO_3/$HClO_4$ until dry and the residue dissolved in dilute ultrapure HNO_3.

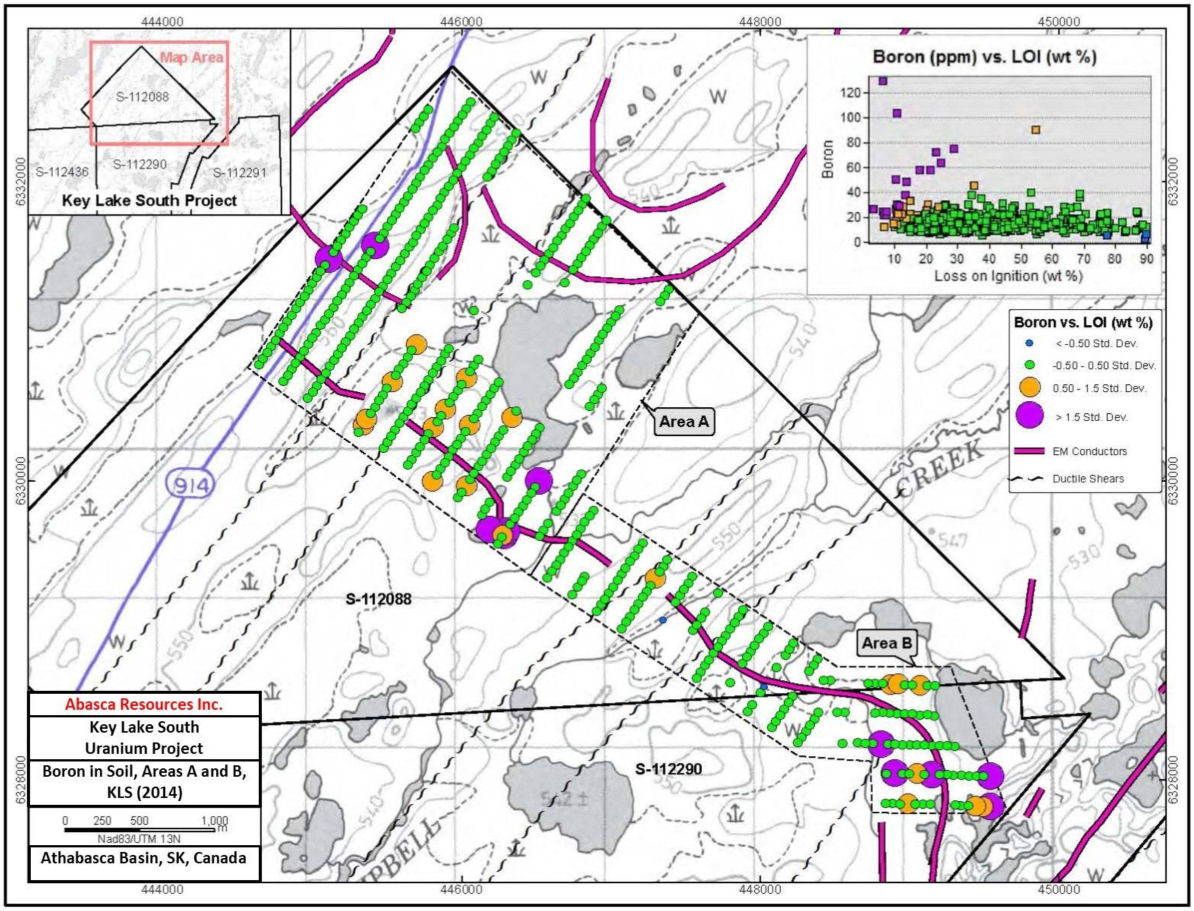


Figure 9-7: Areas A and B: Boron in Soils (Zhou *et al*., 2018)

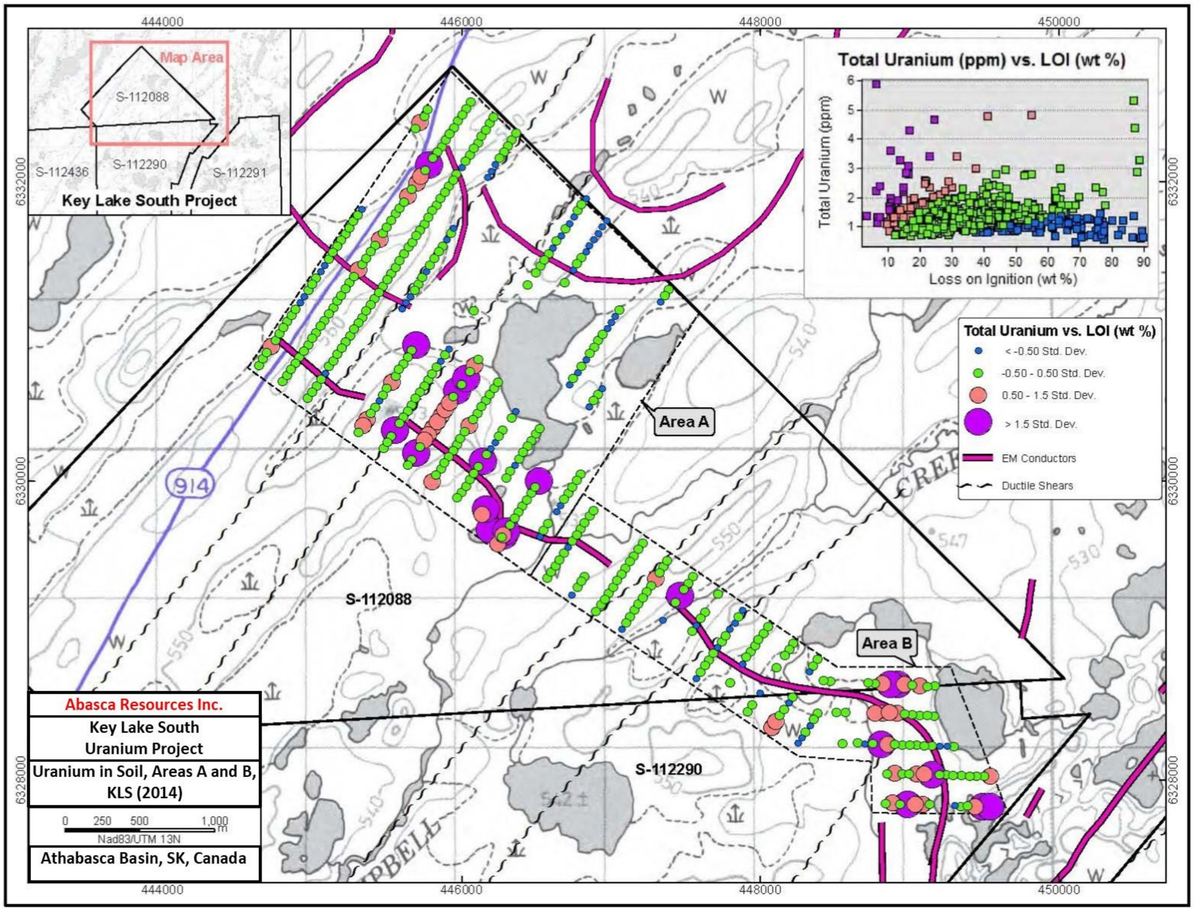


Figure 9-8: Areas A and B: Uranium in Soils (Zhou *et al.*, 2018)

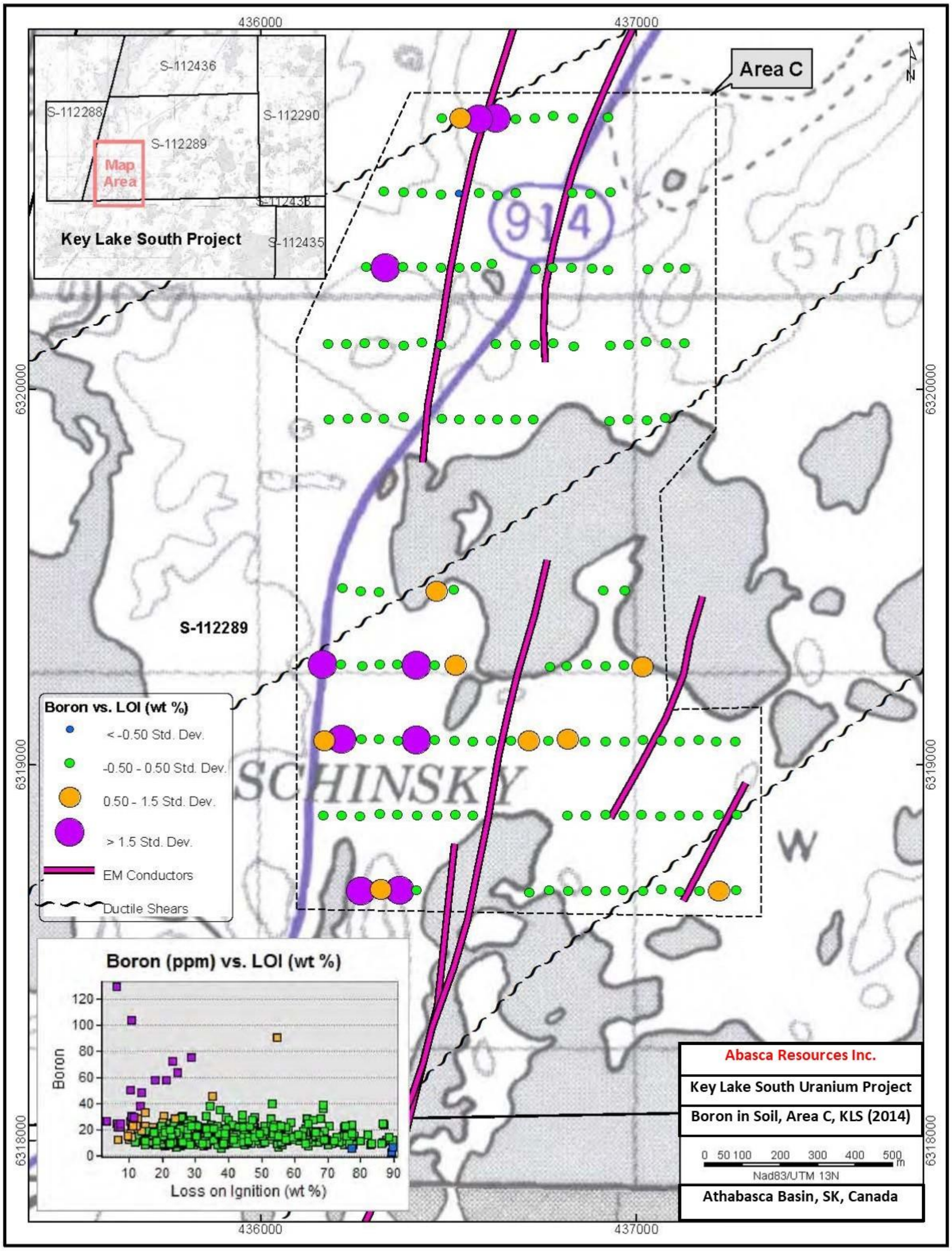


Figure 9-9: Area C: Boron in Soils (Zhou *et al*., 2018)

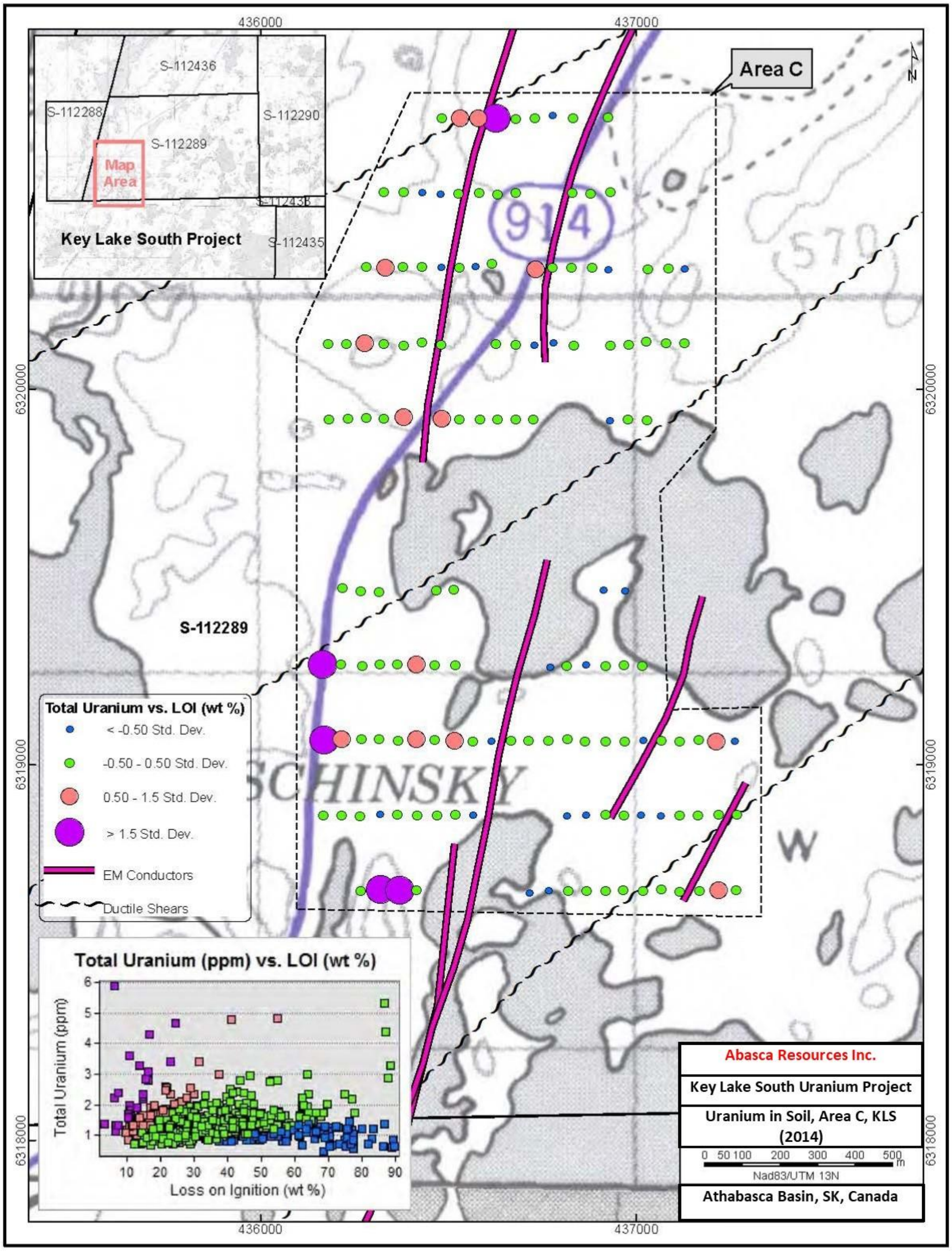


Figure 9-10: Area C: Uranium in Soils (Zhou *et al*., 2018)

9.3.2 Soil Survey Results

The samples containing peat have high values for Loss on Ignition (LOI). Samples with high LOI values (therefore high in organics) generally have higher values for most elements, except for boron, cobalt, and uranium; those elements weren't as affected by high LOI values. The influence of LOI was not affected by the type of digestion, whether using partial or total digestion. Figure 9-11 shows plots of Nickel, Uranium, Vanadium, and Zinc Assays vs. Loss on Ignition (LOI).

Based on a review of the plots for each element, it was decided to treat the geochemistry results as two groups:

- Samples with < 85% LOI were classified as soils
- Samples with ≥ 85% LOI were classified as peat, except for Area C where only three samples had ≥ 85% LOI

To remove the LOI influence, each assay was normalized by the corresponding LOI result, the standard deviation was calculated, then the data was sorted based on the third standard deviation percentile.

The treatment of the samples is considered appropriate since the survey procedure could have distinguished soil samples from vegetation (peat) samples using the field notes. However, initially plotting all the samples with LOI allowed for a quantitative separation of soil and vegetation samples.

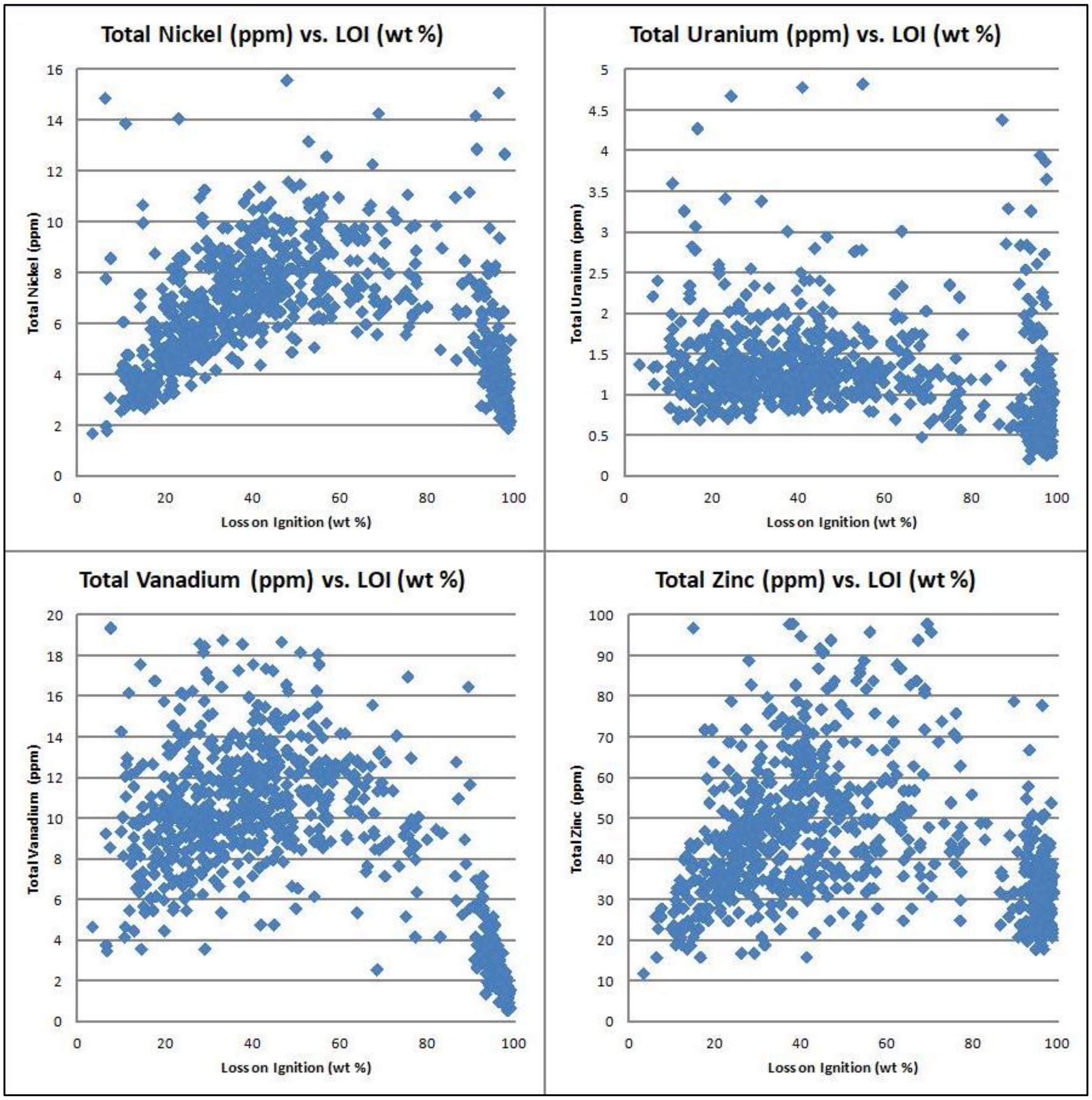


Figure 9-11: Nickel, Uranium, Vanadium, and Zinc Assays vs. Loss on Ignition (LOI) (Zhou *et al*., 2018)

9.3.3 Interpretation of Soil Survey Results

The geochemical interpretation considered the following as elements of interest: As, B, Co, Cu, Pb, Mo, Ni, U, V, and Zn.

In Areas A and B, a compilation of the soil geochemical results returned four prominent multi-element anomalies, S1 to S4, which are shown in Figure 9-12. Anomalies S1 and S2 (Campbell target) are considered strong exploration targets due to the coincidence of high metal content with favourable geophysics and interpreted structures. Four anomalies in what were classified as peat samples are also outlined but are not considered reliable.

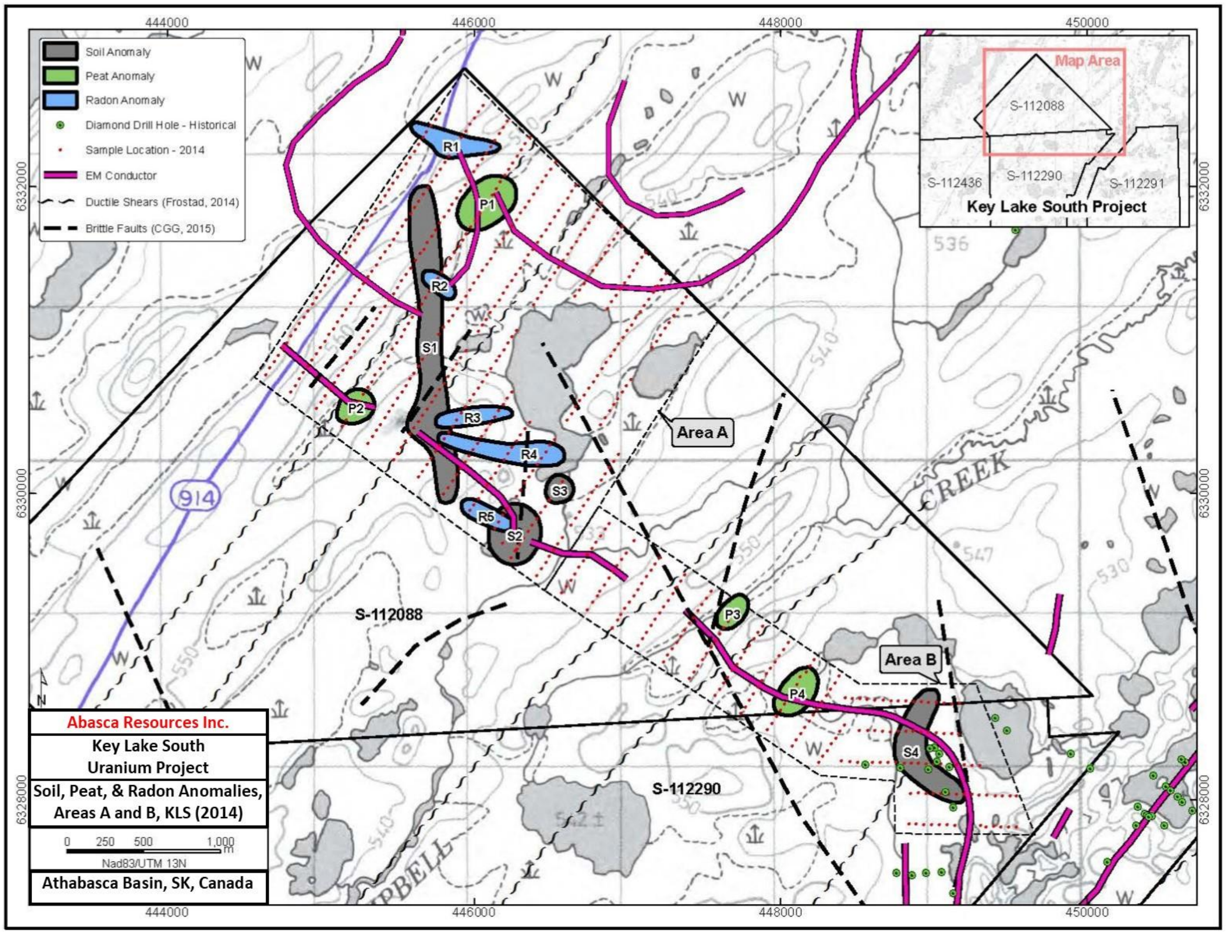


Figure 9-12: Compilation Plan Map of Areas A and B with Soil, Peat, and Radon Anomalies Overlaid with Interpreted Structure and EM Conductors (Frostad., 2018)

In Area C (Mustang target), the soil geochemical results returned five prominent multi-element anomalies, S5 to S9, which are shown in Figure 9-13. Anomaly S7 constitutes a high-priority target due to its coincidence with geophysics and a nearby lake sediment anomaly.

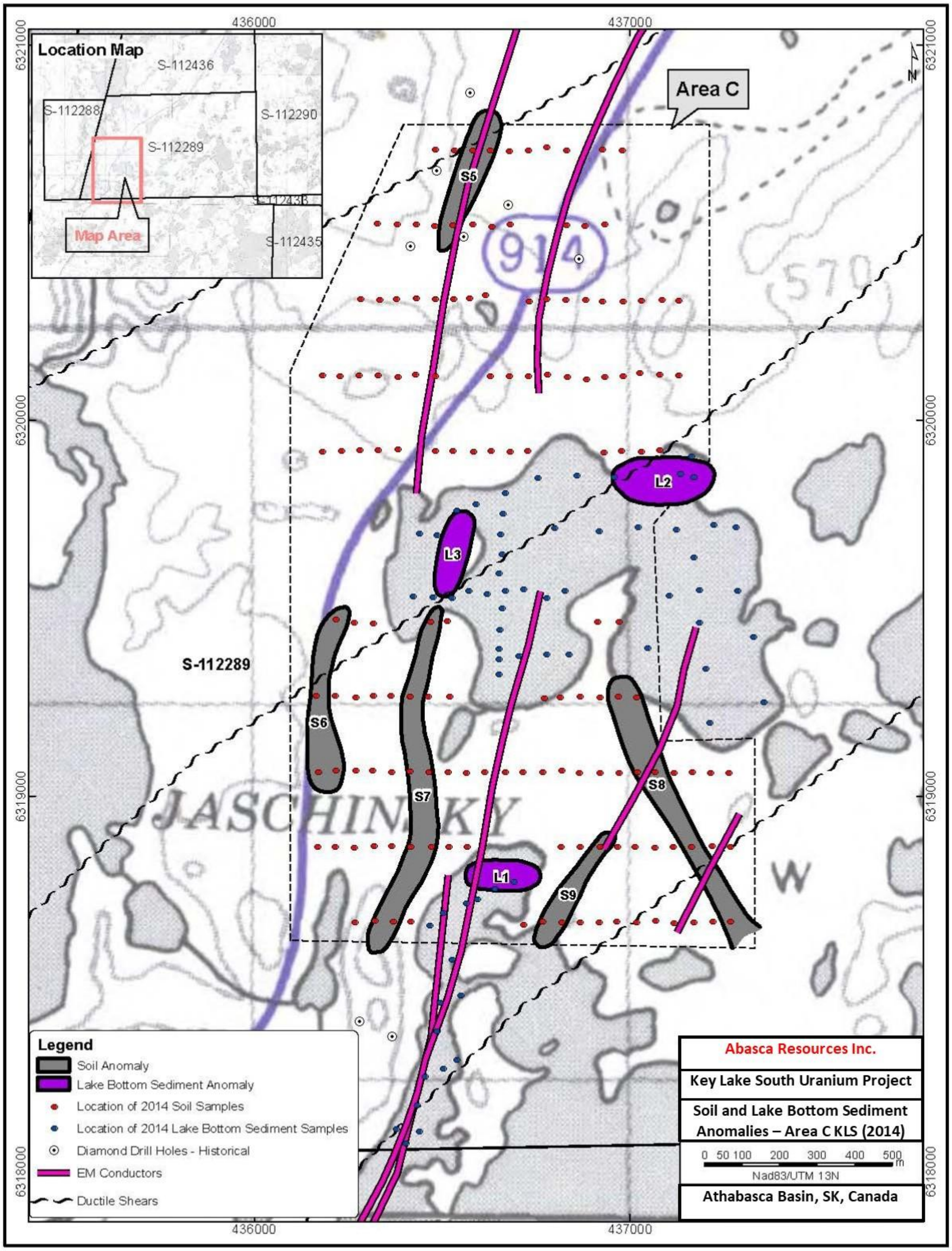


Figure 9-13: Compilation Plan Map of Area C with Soil, Peat, and Radon Anomalies Overlaid with Interpreted Structure and EM Conductors (Frostad, 2018)

Table 9-2 shows the element associations for each interpretated anomaly by Frostad (2015) and Table 9-3 summarizes their characteristics. The geochemical anomalies were selected by noting groups of samples with higher concentrations of numerous elements above what was considered background for each area. A definition of anomalous concentration for each element was not made and a ranking of the soil/vegetation anomalies based on concentrations was not attempted.

Table 9-2: Element Association Defining Soil and Peat Anomalies (Frostad, 2015)

Soil/Peat Anomaly ID	Area	Anomalous Element Concentration									
		As	B	Co	Cu	Pb	Mo	Ni	U	V	Zn
S1 (North)	A										
S1 (South)	A										
S2	A										
S3	A										
P1	A										
P2	A										
S4	B										
P3	B										
P4	B										
S5	C										
S6	C										
S7	C										
S8	C										
S9	C										

Table 9-3: Summary of Soil Geochemical Anomalies (Frostad, 2015)

Area	Name	Size	Orientation	Intensity	Geophysical Correlation	Geological Correlation	Geochemical Signature
A	S1	2 km × 75 m	N-S	Strong	Strong airborne magnetic and gravity structure		North: B, Mo, Ni, V South: Cu, Mo, Ni, Pb, V, Zn
	S2	400 m	Round		Significant bend and break in the EM conductor that lies on the south side of a magnetic high.	Magnetic high may represent granite	Highest U values of the survey: up to 104 ppm U (project average= 1.5 ppm U. All elements of interest are elevated except Pb.
	S3	150m	Round		None	Anomaly may be associated with a N-S fault or may be due to topographic low capturing metals from S2 anomaly.	All elements of interest are elevated with the exception of copper and zinc. Strongly anomalous at 18.6 ppm U.
	P1 and P2	Soil samples w >85% LOI, anomalies not considered reliable as may be distal from source					
B	S4	700 m × 200 m	Arcuate	Strong	coincident large-scale bend in an EM conductor, a magnetic low, and a gravity low.	Near hole CS-78-11 that intersected 167 ppm U over 3.7 metres in a clay-altered fault zone.	All elements of interest are elevated.
	P3 and P4	Soil samples w >85% LOI, anomalies not considered reliable as may be distal from source					
C	S5	400 m × 50 m × 100 m	N-S	Strong	Along an EM conductor and coincident with a strong magnetic low and a moderate gravity low response.		Elevated in all elements of interest with the exception of lead and zinc.
	S6	600 m	N-S	Strong	Same orientation as nearby EM conductor. Located along the eastern edge of a magnetic high as well as the eastern edge of a small gravity high response.	Coincides with an interpreted NE-trending fault.	Elevated in all elements of interest. Coincides with highway so may be due to contamination.
	S7	1 km × 100 m	N-S		Parallel to EM conductors. Within a magnetic low that sits between the major break of the EM conductors.	May be showing where hydrothermal alteration of graphite has suppressed an EM response.	All elements of interest have elevated concentrations except As.
	S8	~ 800 m × 50 m	N-NW		Cross-cuts two EM conductors. At the western edge of a magnetic high and a gravity high.	Interpreted to be associated with a fault.	
	S9	400 m × 50 m × 100 m		Weak	At the southern terminus of an EM conductor		Anomalous in Co, Mo, Ni, U and V.

9.3.4 2014 Lake Sediment Survey

A lake sediment sampling survey was conducted in 2014 in Area C, on claim S-112289, proximal to the 2014 soil sampling grid. A total 65 samples were collected (Figure 9-15).

A lake bottom sampling grid was designed to cover areas hosting known EM conductors. The average distance between samples over the conductors was 50 metres and additional infill samples to cover the lake were taken between 100 and 150 metres apart. Sample coordinates were uploaded into GPS units prior to the field collection. The GPS units were then used to guide the sampling teams to each preselected and pre-named sample site on the lake. An anchor was lowered, and the boat was allowed to steady its position, allowing for two samples to be collected from the same spot if the sample size from the initial sample was not sufficient.

An Ekman Bottom Grab sampler was utilized for collecting the sediment samples from the soft sediment at the bottom of the lakes. As the sampler is lowered, two hinged upper lids swing open to let water pass through and close upon retrieval, preventing sample washout. When the sampler reaches the bottom, a messenger is sent down the line, tripping the overlapping spring-loaded scoops. The samples were then placed with 12 inches × 18 inches sediment sample bags that allowed for water to drain.

9.3.5 Lake Sediment Survey Results

The concentrations of the two elements of interest, uranium and nickel, were plotted against LOI for all lake sediment samples (Figure 9-14). Nickel and uranium values were higher in samples with higher percentages of LOI. Based on a review of these two plots, the anomalous values for uranium and nickel were divided into three groups: i) anomalous, ii) weakly anomalous and iii) background. Results for nickel and uranium are illustrated in Figure 9-14.

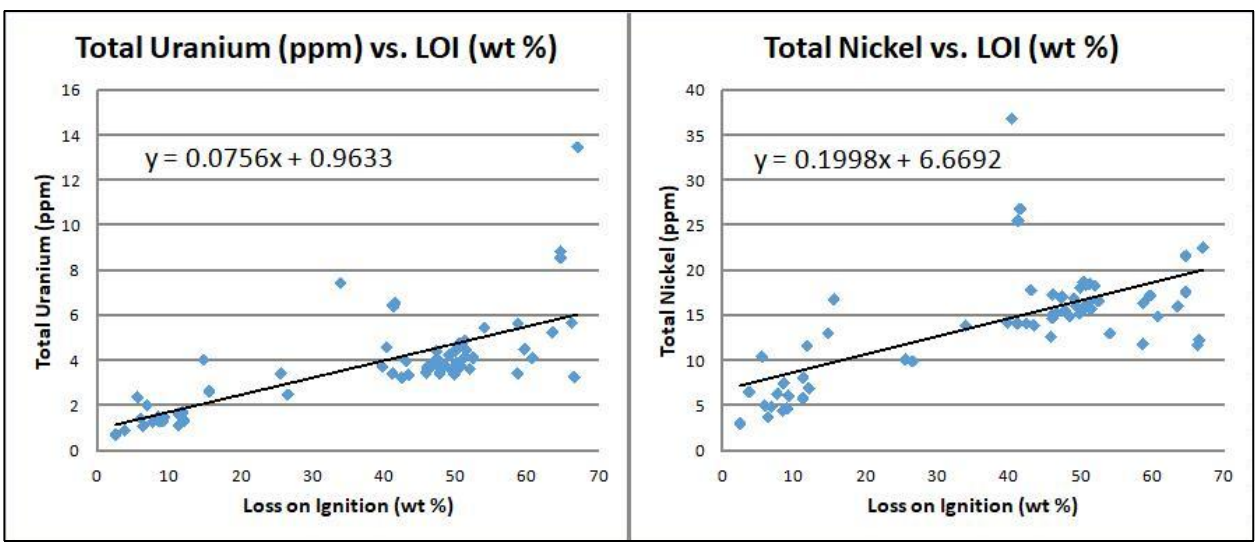


Figure 9-14: Uranium and Nickel vs. LOI in Lake Sediment Samples (Frostad, 2015)

Three lake sediment anomalies were identified (L1 to L3) and are shown in purple on the compilation map in Figure 9-13. Characteristics of these anomalies are summarized in Table 9-4.

Three lake sediment anomalies were identified (L1 to L3) and are shown in purple on the compilation map in Figure 9-13. Characteristics of these anomalies are summarized in Table 9-4.

Table 9-4: Summary of Lake Sediment Anomalies

Area	Anomaly	Intensity	Geophysical Correlation	Other Correlations	Geochemical Signature
C	L1	Strong U; moderate Ni	Near the terminus of an EM conductor; coincident with a magnetic low and a gravity low response	Between soil anomalies S7 and S9.	From 8.6 to 13.5 ppm U over a distance of approximately 100 metres; one of these samples also had the highest nickel concentration at 36.9 ppm Ni.
	L2	Strong U and Ni	No EM correlation but is located on the western edge of a coincident magnetic high and a gravity high response.	May relate to a lithological contact	> 25 ppm Ni and > 6.4 ppm U
	L3	Weak U and Ni	Coincides with a moderate magnetic high and a moderate gravity high.	At northern extension of the S7 soil anomaly.	> 11 ppm Ni and > 4 ppm U

Anomaly L1 is considered the most encouraging lake sediment anomaly, as its high concentrations of uranium and nickel are coincident with favourable geophysics and interpreted structures.

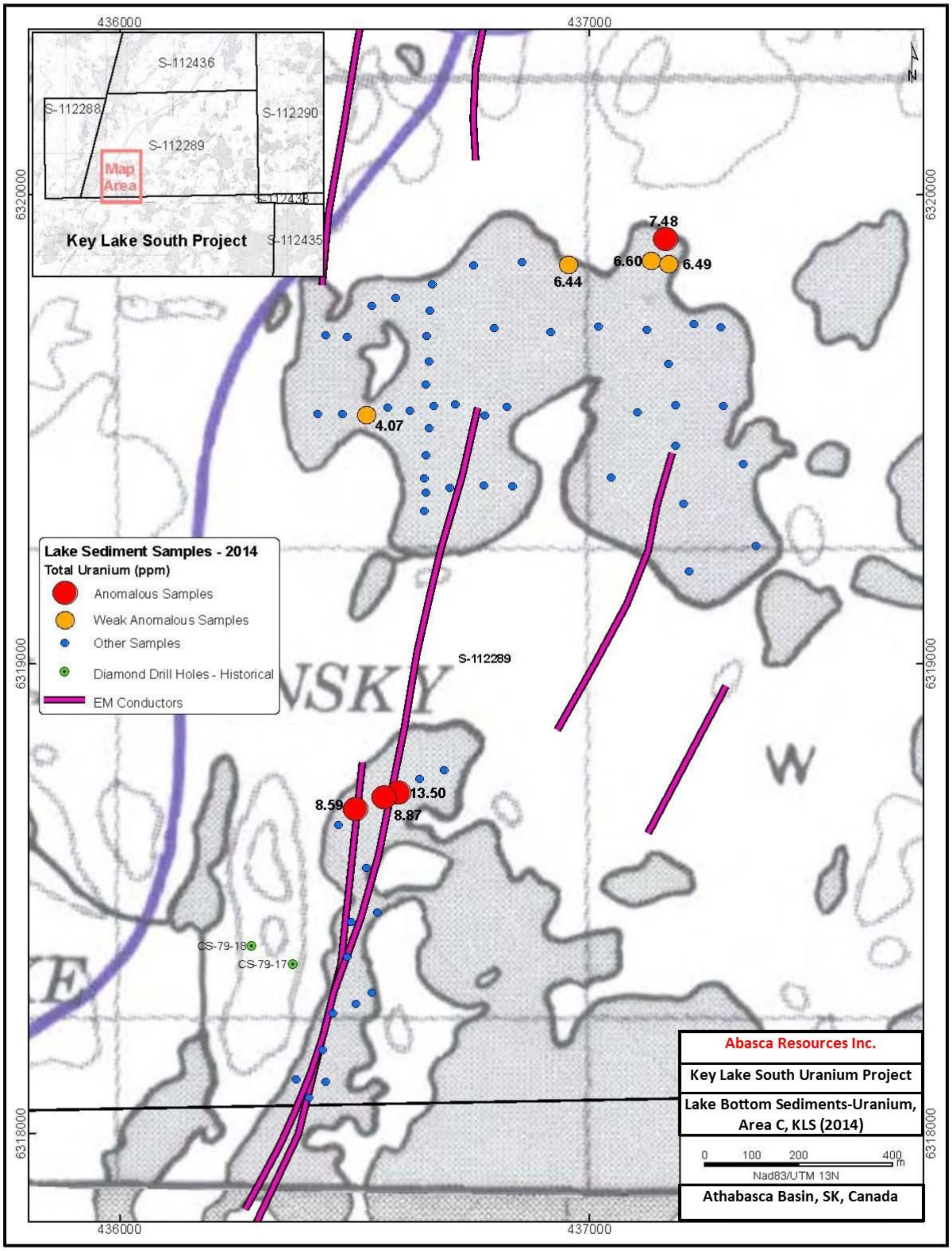


Figure 9-15: Area C Uranium in Lake Sediments (Frostad, 2015)

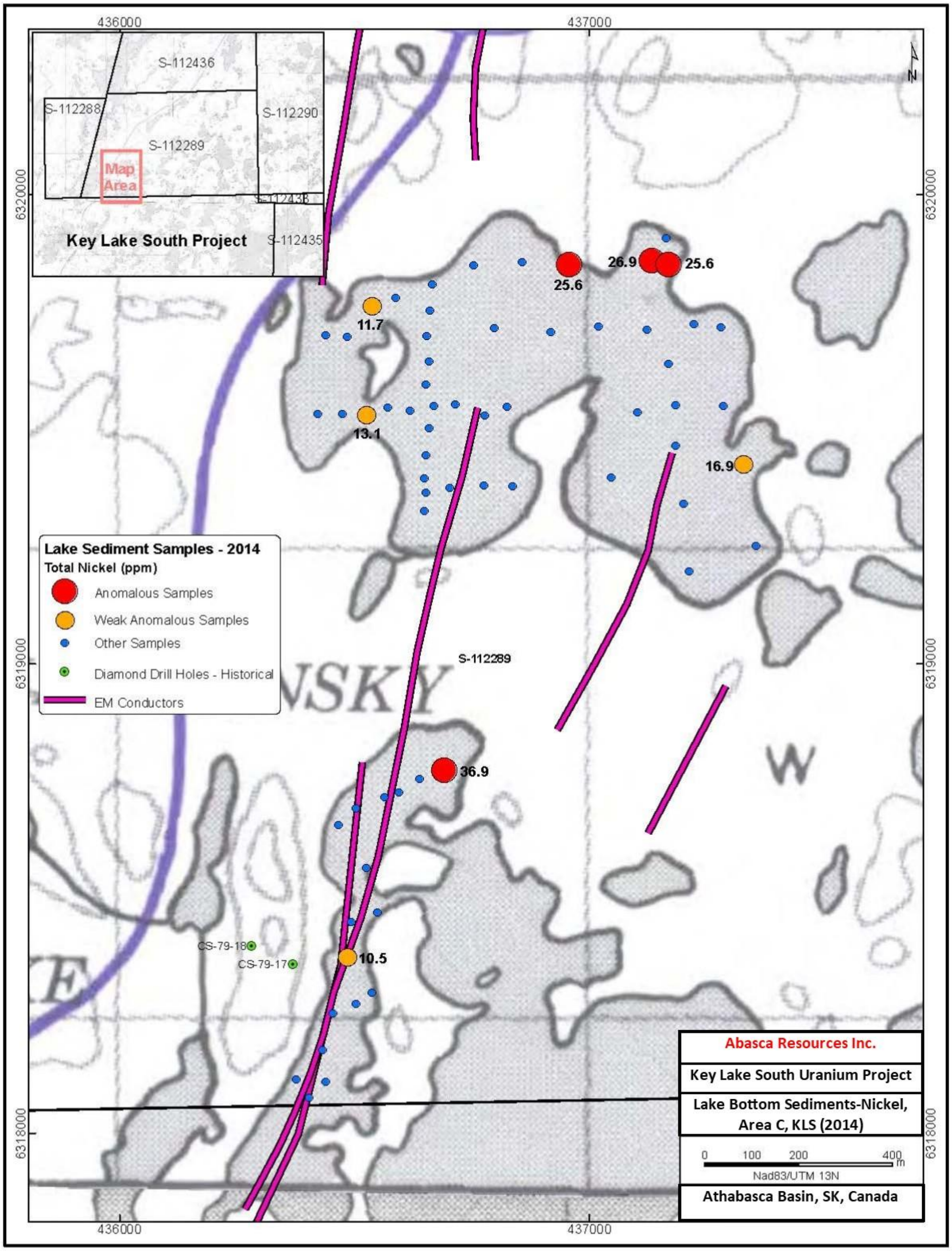


Figure 9-16: Area C Nickel in Lake Sediments (Frostad, 2015)

9.3.6 2014 Radon-in-Soil Survey

In 2014, a radon sampling survey over Area A (disposition S-112339) was conducted in conjunction with the previously described soil sampling survey. The radon sampling survey involved the collection of 282 readings using a Pylon AB6A Monitor with Lucas-style scintillation cells. The Pylon AB6A Monitor measures radon and thoron by detecting the alpha particles from the gases as they decay. Each reading location recorded the background radon, the soil radon reading, and a scintillometer measurement of total gamma recorded.

A sampling grid was designed to utilize the historical cut grid having a line-spacing of 200 metres and picketed stations every 50 metres. Sample coordinates were uploaded to GPS units which were then used to navigate to the predetermined sample site. After choosing a suitable sample location close to the GPS sample coordinate, a battery powered drill with a ¾-inch ship auger bit was used to drill a hole 18 inches deep. The radon probe was then placed in the hole and connected to the scintillation cell with a vacuum hose. A 15 inHg vacuum is created in the cell using a handheld vacuum pump with the valve on the vacuum hose closed. The AB6A is turned on and the date, location, and the hand scintillometer measurement is recorded before the measurements begin. The first three intervals are recorded with the cell under vacuum. When the third interval is complete, the flow valve is opened until the vacuum pressure in the cell reaches zero inHg. This draws the gases from the soil into the cell. Intervals four, five and six then measure the decay of the gas in the cell. The measurements are saved internally in the AB6A. When the measurements are complete, the cell is then flushed for 25 seconds using the vacuum pump.

9.3.7 Radon Results

The radon samples were collected sites that were also sampled for soils. To ensure that peaty terrain was not influencing the radon-in-soil readings, LOI from the soil sampling results were plotted against the measured radon concentrations (Figure 9-17). The plot suggests that the peat-rich content (i.e., with LOI > 90%) does not influence the radon measurements. The radon results are shown in Figure 9-18.

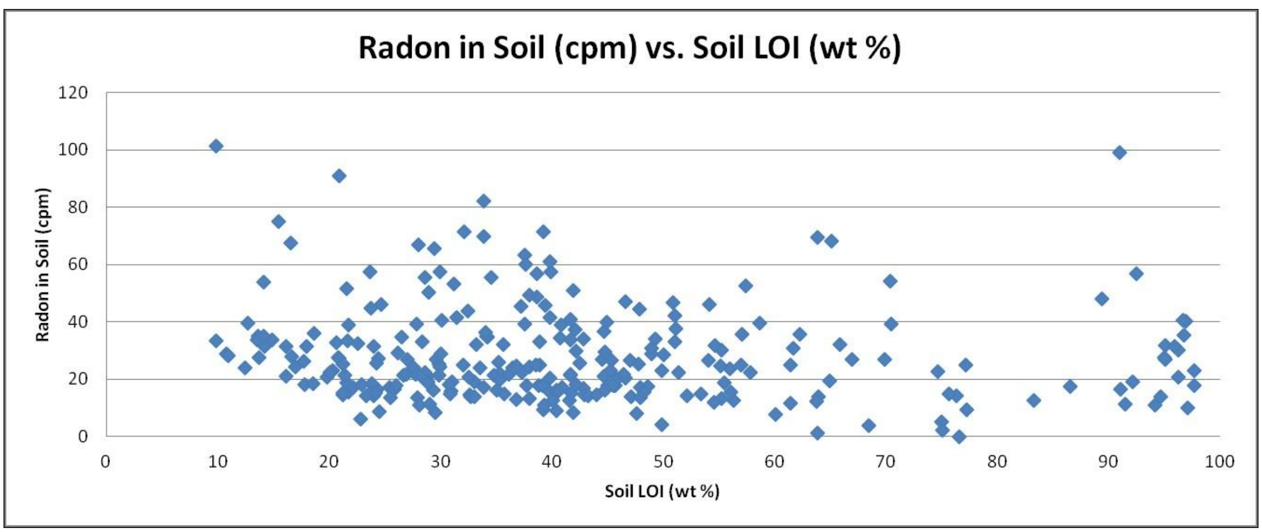


Figure 9-17: Radon in soil (cpm) vs. soil LOI (wt %) (Zhou *et al.*, 2018)

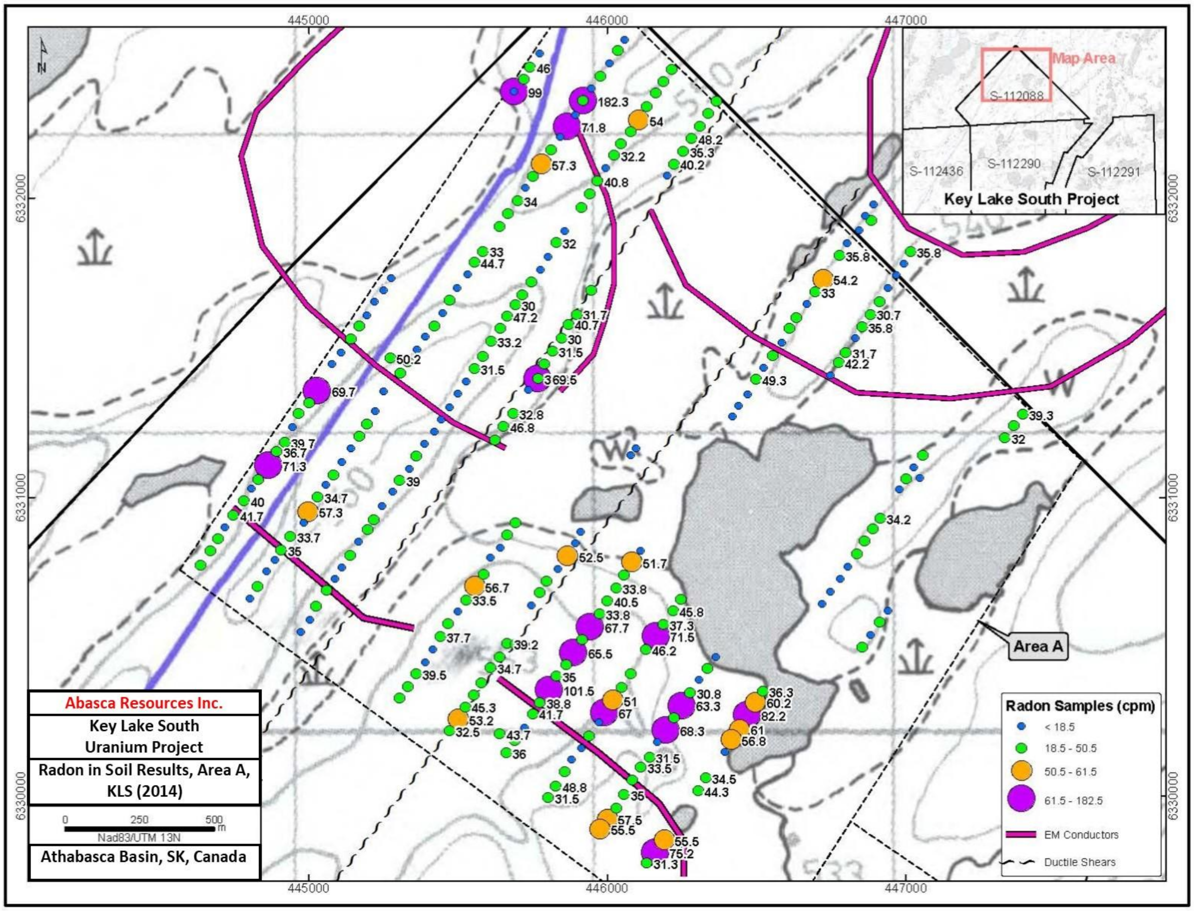


Figure 9-18: Radon Survey Results Area A (Zhou *et al*., 2018)

Five radon-in-soil anomalies were identified, R1 to R5, and are plotted on the geochemical compilation map for areas A and B (Figure 9-12). High radon concentrations were measured proximal to strong soil anomalies within the Campbell target area. Although the radon-in-soil anomalies appear discordant to the underlying geology, the anomalies are interpreted to be migrating downhill from their source. The anomalies are summarized as follows:

- Anomalies R1 and R2 appear related as they occur at opposite ends of an arcuate EM conductor located in the northern corner of the sampling grid.
- Anomaly R2 is coincident with the northern portion of the S1 soil anomaly.
- The R3 and R4 radon-in-soil anomalies are associated with the S1 soil anomaly but trend east-west. The R3 and R4 anomalies are interpreted to share a similar uranium source, located where the S1 soil anomaly crosses a break the EM conductor. The anomalies are interpreted to disperse downhill from this topographic high towards the central lake.
- The R5 radon-in-soil appears to be associated with the S2 soil anomaly, the strongest soil anomaly in the program. Again, the elongate east-west trending anomaly may be suggesting migration away from a radon source near or beneath the S2 soil anomaly.

9.4 Geological Surveys

9.4.1 2014 Geological Mapping

Between July 9 and 15, 2014, a geological mapping program was conducted by four SaskCo geologists. The fieldwork was conducted from a camp constructed at kilometer 185 on Highway 914. Although the Graham Lake skidder trail provided ATV access into the central area of the project, most locations were only accessible by foot due to dense tree cover, muskeg, and steep topography. The mapping program was successful in locating most of the key outcrops as well as discovering additional outcrops. However, due to limited access to the project, not all historical outcrops were visited.

The purpose of the mapping program was to validate historical mapping, to improve the density of mapping and to create an updated lithological map by compiling the field results with recent airborne magnetic and gravity surveys. The resultant project geology map is presented in Figure 9-19.

The 2014 mapping program identified seven lithologies. The nomenclature used in historical maps was considered and integrated into the geological compilation map. The metasedimentary sequence was dominant in the mapped area. The electromagnetic (EM) conductors detected by ground and airborne surveys correspond with the pelites and psammo-pelites and foliation measurements closely related to magnetic fabrics. Large granitic bodies were mapped on the eastern and western margins of the project.

Samples of each rock type were collected and used for specific gravity measurements to aid in interpreting the results of the airborne gravity survey. Calc-silicates were the densest unit averaging 2.76 g/cm^3 while granite was the least dense averaging 2.61 g/cm^3. The specific gravity of the metasediments was in between these two rock units with an average density of 2.67 g/cm^3.

Table 9-5 summarizes these results.

Table 9-5: Summary of Field-Specific Gravity Survey

Rock Type	No. Samples	Specific Gravity (g/cm^3)			
		Mean	Median	Min	Max
Granite	2	2.60	2.60	2.63	2.57
Archean Granite	4	2.61	2.60	2.65	2.58
Pelite	43	2.66	2.69	2.96	1.30
Pegmatite	7	2.67	2.59	3.08	2.49
Psammite	14	2.68	2.66	3.01	1.59
Calc-Silicate	11	2.76	2.77	2.93	1.56
Total	**81**				

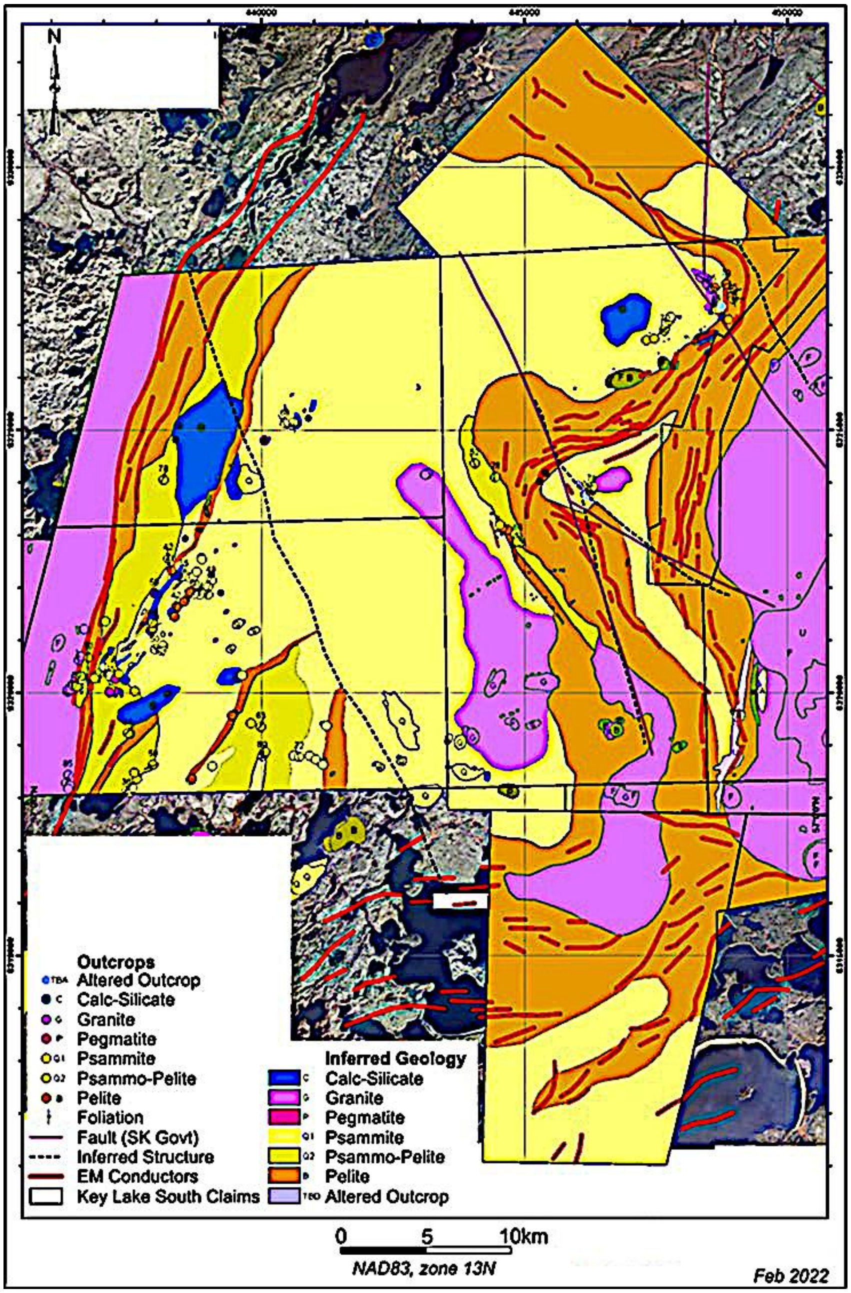


Figure 9-19: Geology Map for the Key Lake South Uranium Project (SRK, 2022)

9.4.2 2015 Exploration Targeting

Frostad (2015) defined 14 exploration targets and described them in detail with accompanying geophysical maps. Subsequent compilations that integrated results of later surveys further confirmed and refined these targets.

A structural interpretation and targeting exercise conducted by SRK (2015) generally agreed with Frostad's assessment and concluded with a priority ranking of the targets. The rankings as well as the characterization of the various parameters are relative to other targets within the dataset and are not to be interpreted as absolute values. The description and ranking of these targets are summarized in Table 9-6 and illustrated in Figure 9-20.

Table 9-6: Exploration Targets

Target/Priority Ranking	Significant Historical Exploration	Anomalous Radiometric Signature	Geochemical Signature	Geology	EM Signature	Structure	Geophysical Indicators of Alteration	Conclusions
Loki LOW to HIGH	Extension of the Key Lake Mine conductor?	Strong	U, radon	Conductors are interpreted as graphitic	Conductor in hinge of F_3 fold; corresponds to broad mag low	Conductors are folded and faulted.	Gravity low	High potential due to possible correlation with Key Lake Mine conductor.
Campbell HIGH		Moderate	U, radon		5 kilometres long narrow low-mag conductor forms boundary with radiometric high	Conductor is folded, faulted (brittle D_5), and parallel to a D_6 fold.	Gravity low coincident with conductor	
Hart HIGH	Considerable historical drilling for U: graphitic pelites, kaolinite	Strong	Strong U, moderate radon,		Very strong conductor coincident with the boundary of a gravity low	Conductor in F_4 hinge intersected by NE D_5 brittle fault.		
Zimmer HIGH	Sparse drilling, best targets not tested.	Strong			Semi-continuous relatively linear low-mag sub-parallel conductors	Conductor parallel to NE D_5 fault.	Linear gravity low, contiguous with conductor	Graphitic pelites with complex geometries, ductile deformation with evidence of later brittle reactivation, and anomalous geochemical, radon, radiometric and/or gravity surveys
Twin Lake HIGH	Historical drilling w few minor U anomalies.	Strong	U, moderate radon	Northern conductor: graphitic. Southern conductor: massive sulphide.	Series of semi-continuous relatively linear conductors	Conductors and subparallel D1-D3 shear zones are folded by F_4 fold. D_5 and D_6 faults bisect the core of the fold.	Gravity low	
Mustang HIGH	Previous drilling but did not test best targets.	Moderate		Graphitic pelites overlying a granite	Semi-continuous linear conductors (graphitic)	Conductors = D_1-D_3 shear zones, crosscut by NE D_5 faults and NW D_6 faults.	Broad gravity low in south	
Reed MODERATE		Moderate	Moderate radon		3 kilometres continuous linear high mag conductor	Conductor is parallel to D_1-D_3 shear zones and a D_6 brittle fault and is cut by a D_5 brittle fault.	Gravity and mag lows	
Graham Lake MODERATE	Historical lake sediment survey	Strong	Moderate radon		Multiple semi-continuous linear conductors, potentially truncated by E-W conductors in the south.		Gravity low	
Sky MODERATE		Moderate	Moderate U, moderate radon		Curvilinear conductor straddling the contact with Zimmer Lake granite	Conductor is parallel to D_1-D_3 shear zones and is folded or faulted.	Gravity low	Structural and stratigraphic settings of these target areas may be favourable but more detailed work is needed.
Seager MODERATE	Drilling intersected graphite, graphitic pelites, and uraniferous pegmatites. Granite/metased contact is between two conductors. Radioactive boulders.	Strong	Moderate U, moderate radon	Faults and conductors mark the marking the granite/metased contact.	Continuation of Mustang conductors	Conductors are along D_1-D_3 faults.		
Seager North MODERATE		Moderate	Moderate radon				Gravity low	
Costigan North LOW								
Costigan South LOW			Moderate radon		Shorter and disjointed EM conductors	Conductors are parallel to D_1-D_3 shear zones which are tightly folded (F_4). Cut by D_5 fault and multiple D_6 faults.	Gravity low	EM conductors that may represent thin packages of metasedimentary rocks overlying granite bodies
Jaschinsky LOW		Strong						

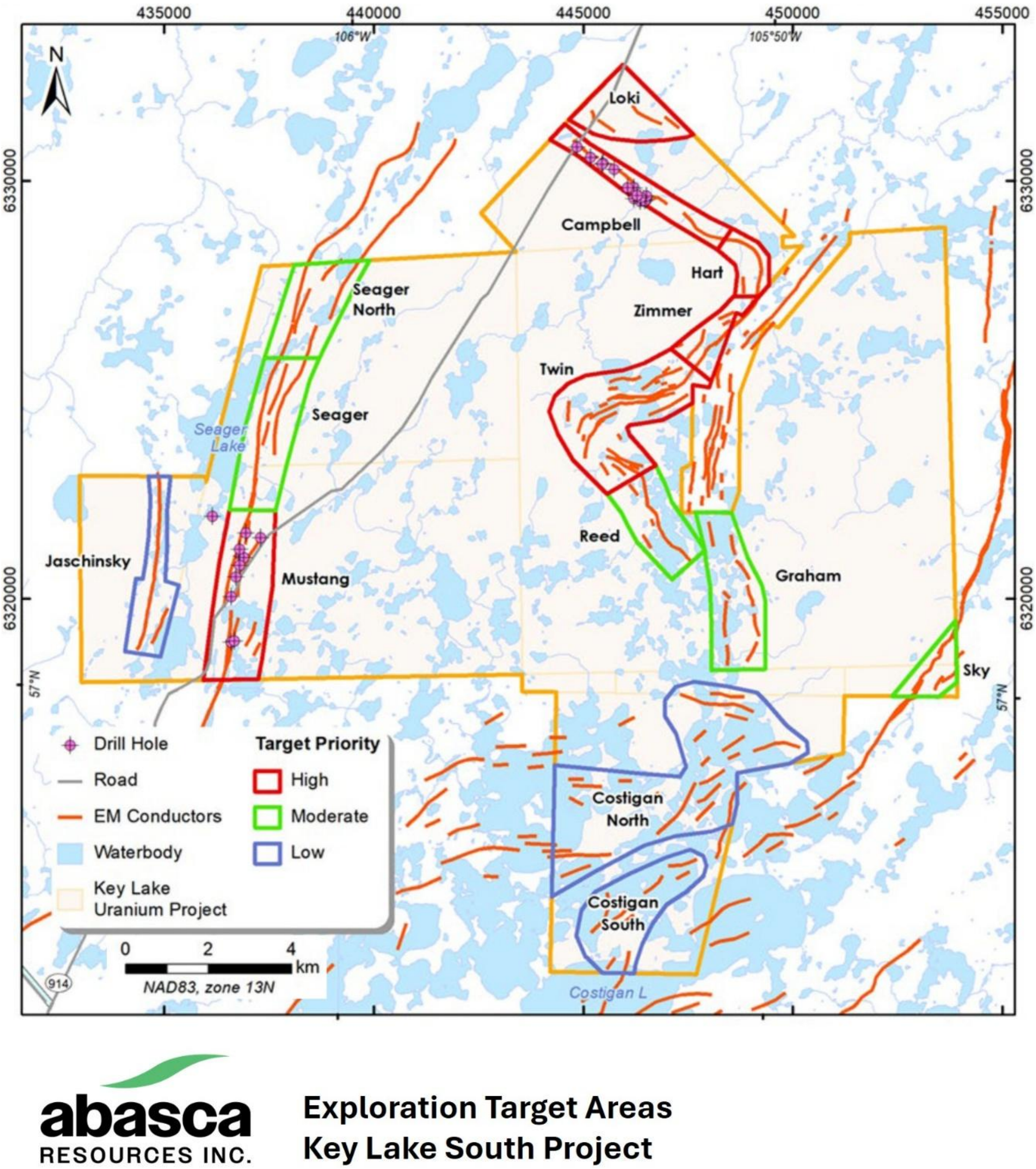


Figure 9-20: Exploration Targets (Modified from SRK, 2015)

9.5 QP Comment

In the Author's opinion, the sampling methods and sample quality meet or exceed industry standards, the samples are representative, and there are no known factors that may have resulted in sample biases.

10.0 DRILLING

10.1 Summary

In 2016, the previous operator, SaskCo, completed a 26-hole diamond drilling program designed to test geophysical anomalies associated with conductive features interpreted to represent graphitic horizons prospective for uranium mineralization. The program resulted in the discovery of the Loki Flake Graphite Deposit in the northern portion of the Project and intersected localized weak uranium mineralization within the Mustang target area.

Since acquisition of the Project, Abasca has completed six drill programs on the Project during the winter 2023, summer 2023, summer 2024, winter 2025, summer 2025, and winter 2026 field seasons. The 2023 programs were primarily focused on uranium exploration at the Project, whereas the summer 2024 program focused principally on the initial delineation of the Loki Flake Graphite Deposit, with limited uranium exploration conducted at additional targets across the Project.

In the fall of 2023, historical drill core from the 2016 Campbell program was re-sampled and re-analyzed for graphite content. Based on the results of this work, the Company initiated follow-up drilling to evaluate the graphite potential of the area. A notable intercept from hole KS-CC16-12 returned 22.2% Cg over 0.5 m.

The 2024 drilling campaign focused on the initial delineation of the Loki Deposit at inferred-resource drill spacing (100 m x 100 m). A total of 20 diamond drill holes, comprising 5,499 m, were completed over a 600 m strike length of the Loki Flake Graphite Deposit. All drill holes intersected significant graphitic intervals comparable to those identified during the 2016 drilling campaign. Highlighted intercepts include hole KLS-24-031, which returned 7.18% Cg over 52 m.

An additional two drill holes totaling 912 m were completed on a parallel conductor north of the Loki Zone, resulting in the discovery of a new graphite-bearing zone (Thor Zone) up to 50 m in width. Highlighted results from this area include hole KLS-24-046, which returned 8.91% Cg over 27 m.

A further five drill holes totaling 2,681 m were completed along the Mustang–Seager Lake trend. Drilling intersected pervasive silicification and localized graphitic fault zones.

The 2025 drilling programs continued deposit expansion at inferred-resource spacing as well as infill drilling intended to support conversion of inferred Mineral Resources to the indicated category. The Company completed two drill programs at the Loki Flake Graphite Deposit comprising 51 diamond drill holes totaling 11,950.6 m. One drill hole was abandoned at 72 m due to poor ground conditions.

The 2025 programs extended inferred-spaced drilling by approximately 300 m to the northwest and 400 m to the southeast along strike. Significant intercepts include hole KLS-25-101, which returned 10.11% Cg over 49.7 m, and hole KLS-25-097, which returned 9.49% Cg over 56 m. Geotechnical data was also collected in the summer drill program at the Loki Deposit.

In addition, six sonic drill holes totaling 235.2 m were completed to characterize overburden conditions and evaluate the unconformity for mineralization potential. Three holes were instrumented with vibrating wire piezometers, and three additional holes were instrumented with standpipe piezometers.

One additional drill hole, KLS-25-072, totaling 429 m, was completed at the Thor Zone approximately 400 m up-dip of KLS-24-046. The drill hole intersected three graphitic intervals totaling 72 m in cumulative thickness. Significant intercepts include 7.36% Cg over 8 m, 9.78% Cg over 32 m, and 6.39% Cg over 32 m.

As of the effective date of this report, the Company completed a winter 2026 drill program at the Loki Deposit comprising 13 diamond drill holes totaling 2,961.3 m. One hole was abandoned at 36 m due to poor ground conditions. The program successfully extended inferred-spaced drilling approximately 200 m along strike toward the southeast. Geotechnical logging was also completed.

In addition, 12 sonic drill holes totaling 404.2 m were completed to characterize overburden conditions and test the unconformity for mineralization. A further five diamond drill holes totaling 1,556 m were completed at the Thor Zone, of which one hole was abandoned at 33 m due to poor ground conditions. Highlighted intercepts include KLS-26-112 (7.78 % Cg over 18.75 m, 3.23 % Cg over 21 m, and 7.8 % Cg over 14 m) and KLS-26-115 (4.7 % Cg over 6 m, 3.36 % Cg over 10 m, 3.48 % Cg over 68 m, and 7.42 % Cg over 14 m).

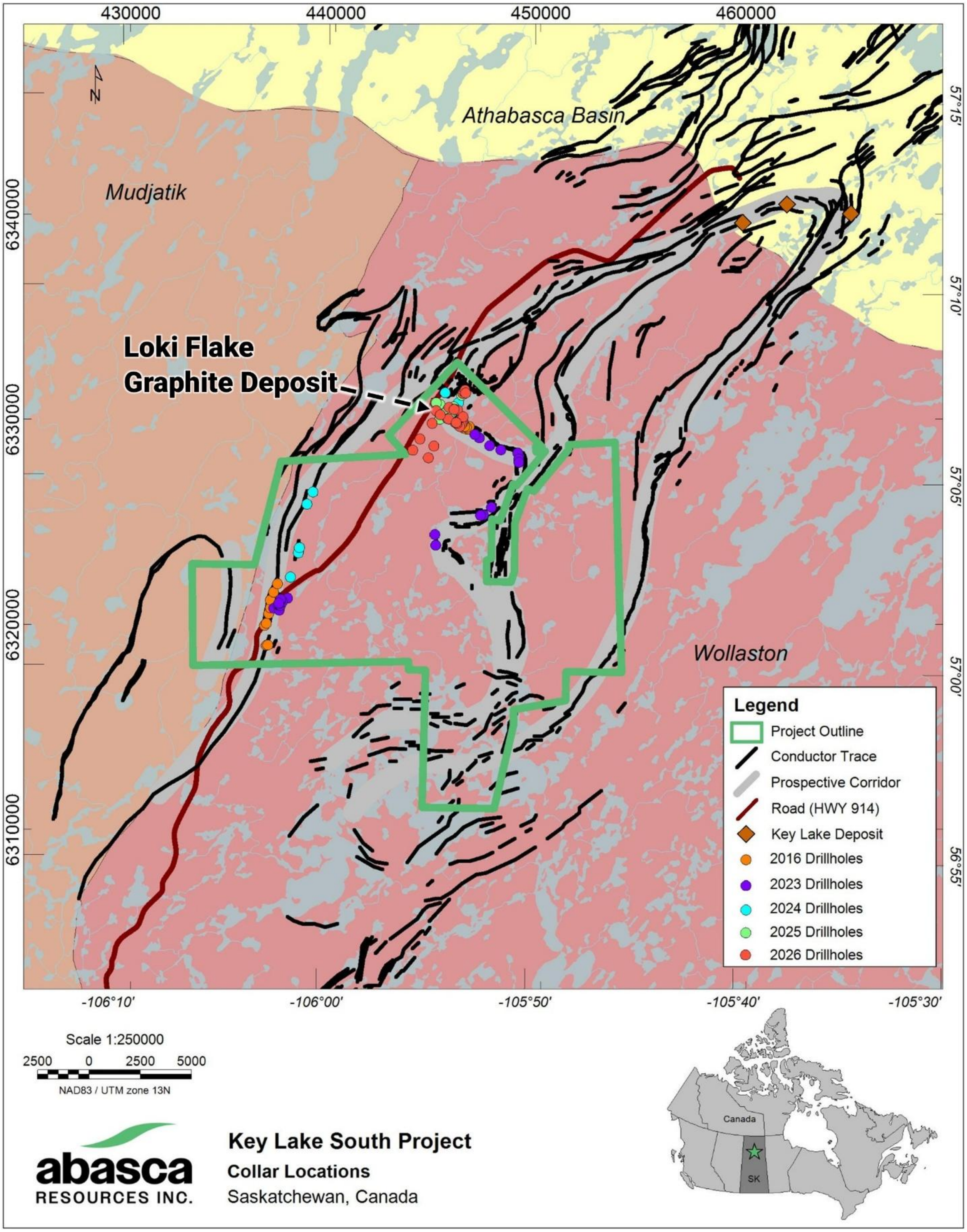


Figure 10-1: Map of the Project Area and the Collar Locations of the 2016 to 2026 Drill Programs

10.2 Uranium Exploration Drilling Results

Two target areas, Mustang and Campbell, were drilled in 2016 by SaskCo for a total of 4,553 metres. The 26 drillhole program intersected anomalous uranium (> 100 ppm U) in both target areas. The best intersection of the Mustang drilling was 256 ppm U over 0.4 m from 48.8 to 49.2 m in hole KS-MS16-07, and the best intersection in the Campbell area was 1950 ppm U over 0.3 m from 73.5 to 73.8 m in KS-CC16-06. The relationship between the sample length and the true thickness of the uranium mineralization is unknown, and the orientation of the system is also unknown.

The Winter and summer drill programs conducted by Abasca in 2023 totalled 10,135 metres. The programs successfully tested prospective conductor corridors throughout the project and intersected localized alteration proximal to re-activated graphic fault zones. The best intersection was 1,260 ppm over 10 cm from 310.5 to 310.6 m (true thickness and orientation is unknown) in KLS-23-004 in the Mustang target area.

In the summer of 2024, regional exploration comprised a total of 5 drill holes totalling 2,681 m were completed along the Mustang-Seager Trend. Strong silicification along with local clay alteration, including illite, and oxide staining near fault zones were observed along the trend. The corridor remains largely untested with many prospective targets. No anomalous uranium (> 100 ppm U) was intersected.

Following the 2024 drill programs, exploration activities on the Project transitioned from uranium exploration to delineation and expansion of the Loki Flake Graphite Deposit. As a result, no additional uranium-targeted drilling has been completed on the Project subsequent to the 2024 field season.

10.3 Graphite Exploration Drilling Results

Previous drilling in 2016 by SaskCo in the area that is now the Loki Deposit focused on uranium exploration given its proximity to the Key Lake deposits as well as EM anomalies. Although the uranium anomalies intersected here were limited, the drilling demonstrated that the graphitic shear zone was at least 2 km in strike-length, approximately 35 m in thickness, and extended up to the base of the overburden. At the time, only representative samples from the graphitic shear zone were collected for whole-rock analyses at SRC Geoanalytical Laboratories (SRC) in Saskatoon for the purpose of uranium exploration in the area.

At the end of 2023, Abasca evaluated the Campbell area for its graphite potential. Representative samples from the 2016 drill program stored at SRC were tested for graphite content. The best assay returned 22.2 % Cg from KS-CC16-12 between 88.0 and 88.5 m. Additionally, 10 samples were selected for flake size evaluation by QEMSCAN of which, the best result returned a median diameter passing percentage of 214 µm. These initial results were sufficient for Abasca to conduct further investigations that led to the 2024 summer drill program, which comprised delineation of the Loki Flake Graphite Deposit as well as the drilling of two holes along parallel conductors to the north. Additionally, the core from the 2016 drilling was re-sampled, comprising 656 samples, and analysed for graphite content. Results from the resampling within the Loki Deposit were comparable with results from the 2024 drilling, with the best result from drillhole KS-CC16-13 from 116.0 to 116.5 m at 17.6 % Cg.

Drilling at the Loki Flake Graphite Deposit in the summer of 2024 included 20 drillholes, totaling 5,499 m. The drilling, which was conducted on a 100 m x 100 m grid, focused on testing the continuity along strike and at depth within the central area of the known conductor. All holes successfully intersected graphite mineralization. Highlighted intercept from this program include 60 m at 9.09 % Cg in KLS-24-043. Representative plan view and cross sections of assay graphite grades of the Loki Deposit are available in section 14.

The two holes drilled along conductors to the north of the Loki Deposit (Thor Zone) both intersected graphite mineralization. Graphite mineralization here was visually similar to the Loki Deposit with the best intersection being 27 m in length at 8.91 % Cg in KLS-24-046.

Further delineation of the Loki Deposit was completed during the 2025 winter and summer drill programs. A total of 51 diamond drill holes totaling 11,950.6 m were completed at the Loki Flake Graphite Deposit, with one hole abandoned at 72 m due to poor ground conditions. Drilling focused on extending the deposit approximately 300 m to the northwest and 400 m to the southeast, while also completing infill drilling to support the conversion of inferred Mineral Resources to the indicated category. All holes successfully intersected graphite mineralization. Significant assay results include 10.11 % Cg over 49.7 m in hole KLS-25-101 and 9.49 % Cg over 56 m in hole KLS-25-097. In addition, hole KLS-25-072, drilled approximately 400 m up-dip of the graphite-bearing zone identified north of Loki in 2024, intersected three graphitic intervals totaling 72 m, including 7.36 % Cg over 8 m, 9.78 % Cg over 32 m, and 6.39 % Cg over 32 m. The graphite mineralization intersected in KLS-25-072 confirmed the continuity of the northern parallel conductor, now referred to as the Thor Zone.

In addition to the diamond drilling completed during the 2025 winter and summer programs, six sonic drill holes totaling 235.2 m were completed across the Loki Deposit area to characterize overburden conditions and evaluate the unconformity for mineralization potential. Three holes were instrumented with vibrating wire piezometers, while three additional holes were instrumented with standpipe piezometers.

Further drilling was completed during the 2026 winter drill program. At the Loki Deposit, 13 diamond drill holes totaling 2,961.3 m were completed, with one hole abandoned at 36 m due to poor ground conditions. In addition, 12 sonic drill holes totaling 459 m were completed to characterize overburden conditions and test the unconformity for mineralization. Drilling extended inferred-spaced drilling approximately 200 m toward the southeast along strike of the deposit. In addition, five diamond drill holes totaling 1,556 m were completed at the Thor Zone, with one hole abandoned at 33 m due to poor ground conditions. Highlighted intercepts include KLS-26-112 (7.78 % Cg over 18.75 m, 3.23 % Cg over 21 m, and 7.8 % Cg over 14 m) and KLS-26-115 (4.7 % Cg over 6 m, 3.36 % Cg over 10 m, 3.48 % Cg over 68 m, and 7.42 % Cg over 14 m).

10.4 Drill Hole Surveying

Base Diamond Drilling was contracted for the drill programs in 2023, 2024, 2025 and 2026 used a Zinex A5 diamond drilling rig. Drillholes were spotted using a handheld GPS and confirmed using a Trimble TDC650 GPS with accuracy less than 1 m. The drill was oriented using a Stockholm Precision Tools Rig Aligner and downhole orientation was surveyed using a Stockholm Precision Tools GyroMaster, both tools being True-North non-magnetic gyrocompasses. Axis Mining core orientation tools were used to orient the drill core to obtain structural measurements. Drillholes were probed using either a Mount Sopris 32GR gamma probe or Stockholm Precision Gamma Probe.

Forged Drilling Co. was contracted for the Sonic drilling conducted in 2025 and 2026 using a SDC-450 sonic drill. Drillholes were spotted using a handheld GPS and confirmed using a Trimble TDC650 GPS with accuracy less than 1 m.

The 2016 drilling by SaskCo was conducted by Team Drilling. Downhole surveys and core orientation were conducted using a Reflex equipment. Drillholes were probed using a Mount Sopris HLP-2375 gamma probe.

10.5 Drill Core Handling and Logging Procedures

Drill core is logged at the temporary work camp and the data was entered into a custom Access Database. Core is oriented and recovery and Rock Quality Designation (RQD) is measured between each 3 m block marker and written on each box. Radioactivity is measured using a handheld scintillometer. Geological observations (lithology, alteration, major and point structures) are recorded in the database.

Core samples are taken systematically, every 10 m, and around areas of elevated radioactivity and graphitic zones. PIMA samples are taken every 10 m in clay-rich zones, or where clay alteration is observed.

QAQC samples are taken/inserted throughout the sampling process and include:

- Pulp-duplicate sample: every 100 samples
- Crush-duplicate sample: every 100 samples
- Field-duplicate sample: every 40 samples
- Blank reference material: every 40 samples
- Certified reference material: every 40 samples within mineralized zones
  - Uranium-specific or graphite-specific materials are used, depending on the exploration type.

Core is photographed, both wet and dry, after logging and sample selection has been completed.

In reviewing previous documentation on the project, the QP is of the opinion that core handling and logging procedures in 2016 was completed in a similar manner to that of the Abasca procedure outlined above.

10.6 QP Comment

In the Author's opinion, the drilling, core handling, logging, and sampling procedures meet or exceed industry standards and are adequate for the purpose of this Technical Report. The Author is not aware of any drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results.

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Historical Sampling Procedures

The reviewed historical results from 1969 to 2010 have not been verified by the author and there is a risk that any future confirmation work and exploration may produce results that substantially differ from the historical results. These results, which are only discussed in section 6, are considered relevant to assess the mineralization and economic potential of the project, but not to the Mineral Resource disclosed in this document. Detailed information regarding sample preparation, quality control measures, analyses or security in the private and publicly available reports documenting grab, chip, channel, or drill core sampling, for those exploration programs were not available. The QP is not aware of the laboratory used, the laboratory's relationship to the issuer, or if the laboratory was certified by any standards association for these samples.

11.2 Sample Preparation

Core samples were split using a hydraulic splitter. Prior to each sample being split, the splitter and work area was cleaned. Core samples were split perpendicular to the main foliation to ensure representative samples were being collected. Each core sample comprised half the core with exception to field-duplicate samples which were quartered. Samples were stored in separate bags that were labelled and included a barcode label inside the bag. Each sample bag was sealed and packaged in sample pails and dispatched to SRC. Sample dispatches and analytical requests are documented and stored in the Company's database. Sample preparation by SaskCo during the 2016 drill programs was conducted in a similar manner, but a core saw was used instead of a hydraulic splitter.

11.3 Analyses

SRC operates in accordance with ISO/IEC 17025:2005 (CAN-P-4E), General Requirements for the Competence of Mineral Testing and Calibration laboratories. All processes performed at the laboratory are subject to a strict audit program, which is performed by approved trained professionals.

SRC is independent of Abasca.

11.3.1 Drill Core Geochemical Analyses and Assay

All samples were submitted for SRC's ICP-MS2 Basement Exploration Package. The package includes ICP-MS analysis on the partial digestion as well as trace elements on the total digestion. ICP-OES is used for major and minor elements on the total digestion. All samples are also analysed by ICP-OES for Boron by fusion. Samples within the graphite mineralized zones were analysed for total Graphitic Carbon and total Sulfur by induction furnace. Samples in areas of elevated radioactivity were also submitted for SRC's U_3O_8 Assay package by ICP-OES. Analysis of Rare Earth Elements (REEs) was requested for some samples and was done at SRC by fusion and ICP-MS.

11.3.2 Drill Core Bulk Density Analyses

In-house field density analysis was conducted by weight difference between the sample dry and wet. Analysis was done approximately every 100 samples. Half-core samples were also selected for density analysis at SRC using their SG3 package. Samples are weighed, coated in wax, and weight wet to determine their density.

11.4 Quality Assurance and Quality Control

11.4.1 Protocols

Quality assurance/quality control (QA/QC) programs validate the accuracy of analytical results and are essential for verifying the results of the resampling program. Abasca's QA/QC protocols for drilling programs includes:

- Certified reference materials (CRM) – Determination of accuracy
- Blank samples – Screen cross-contamination between samples during preparation and analyses.
- Duplicate samples – Determination of precision/repeatability

CRMs are inserted into the sample stream at a frequency of approximately 1 CRM per 40 samples. CRMs are also inserted at the beginning of each mineralized zone as well as randomly where the mineralization intensity appears to change. Within graphite-mineralized zones, Abasca used OREAS 722, 723, 724, and 725 CRMs. The OREAS 722-725 CRMs are a Graphite ore standard purchased from Analytichem in Baie D'Urfé, Quebec, Canada. The CRMs are prepared by blending graphite ore from the Queens Graphite Mine in the Matale/Kurunegala Project area in central Sri Lanka with barren I-type granodiorite from the Lysterfield complex located in eastern Melbourne, Australia. Within elevated radioactive zones, Abasca uses BL2A, BL4, and BL5 uranium CRMs prepared by CANMET and sourced from the Beaverlodge Domain uranium deposits. The CRMs used at the Project is summarized in Table 11-1.

Table 11-1: Certified Reference Material Details

CRM Code	Unit	Certified Values	Certified Standard Deviation
OREAS 722	Cg wt %	2.03	0.093
OREAS 723	Cg wt %	5.87	0.169
OREAS 724	Cg wt %	12.06	0.311
OREAS 725	Cg wt %	24.52	0.728
BL2A	U ppm	4264	32.5
BL4A	U ppm	1260	20
BL5	U ppm	71200	350

11.4.2 QA/QC Results

Results of the QA/QC program have been well documented by Abasca and SaskCo. The QP reviewed the documentation provided by Abasca in addition to reviewing the QA/QC data.

Results from the QA/QC samples are continually tracked by Abasca as certificates for each sample batch are received, checking for batches that exceed the failure criteria. Standard reference materials fail when results are more than three standard deviations from the expected value. Blank samples fail when results are greater than 20 times the lower detection limit. If QA/QC samples of a sample batch pass within acceptable limits, the results of the sample batch are imported into the master database. If the QA/QC sample fails, the entire batch is to be reanalyzed.

There were four blank sample failures at 0.26%, 2 at 0.30%, and 0.41% Cg, which were ultimately accepted given the CRMs of the batch passed (Figure 11-1). Of the 463 graphite CRMs measured, twenty-six standards fall outside of the acceptable range of mean plus three standard deviations (Figure 11-2). The field and laboratory duplicates demonstrated an acceptable level of precision and repeatability. Abasca and the QP consider these results acceptable given the failures are just outside the failure limits.

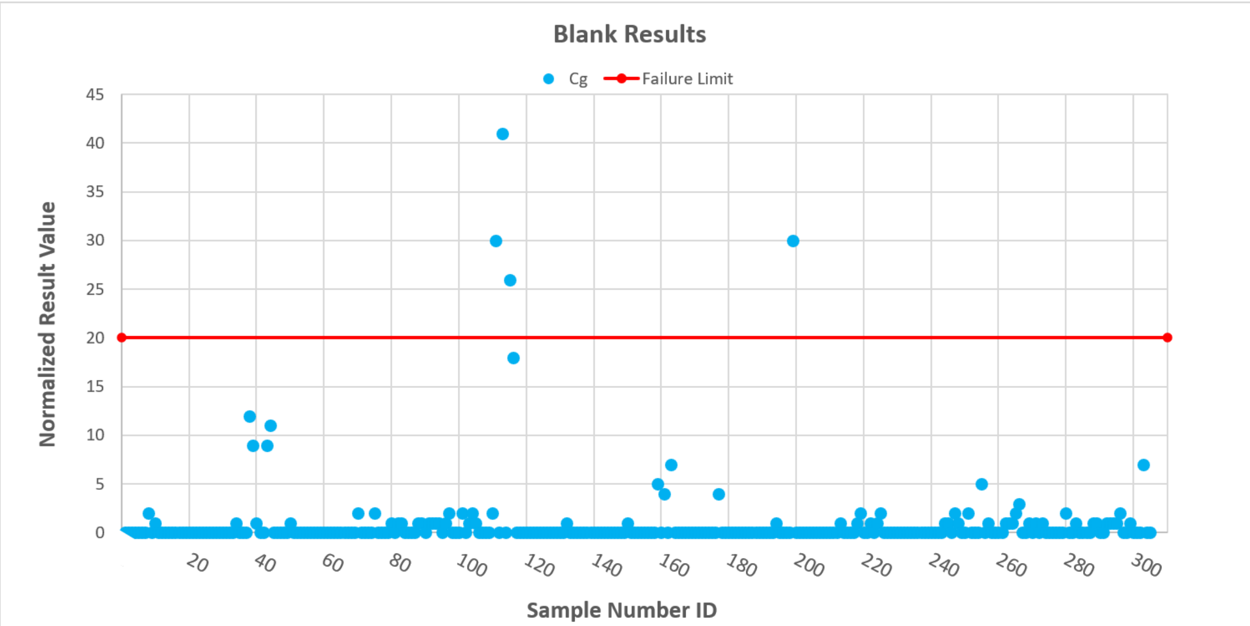


Figure 11-1: Blank Results

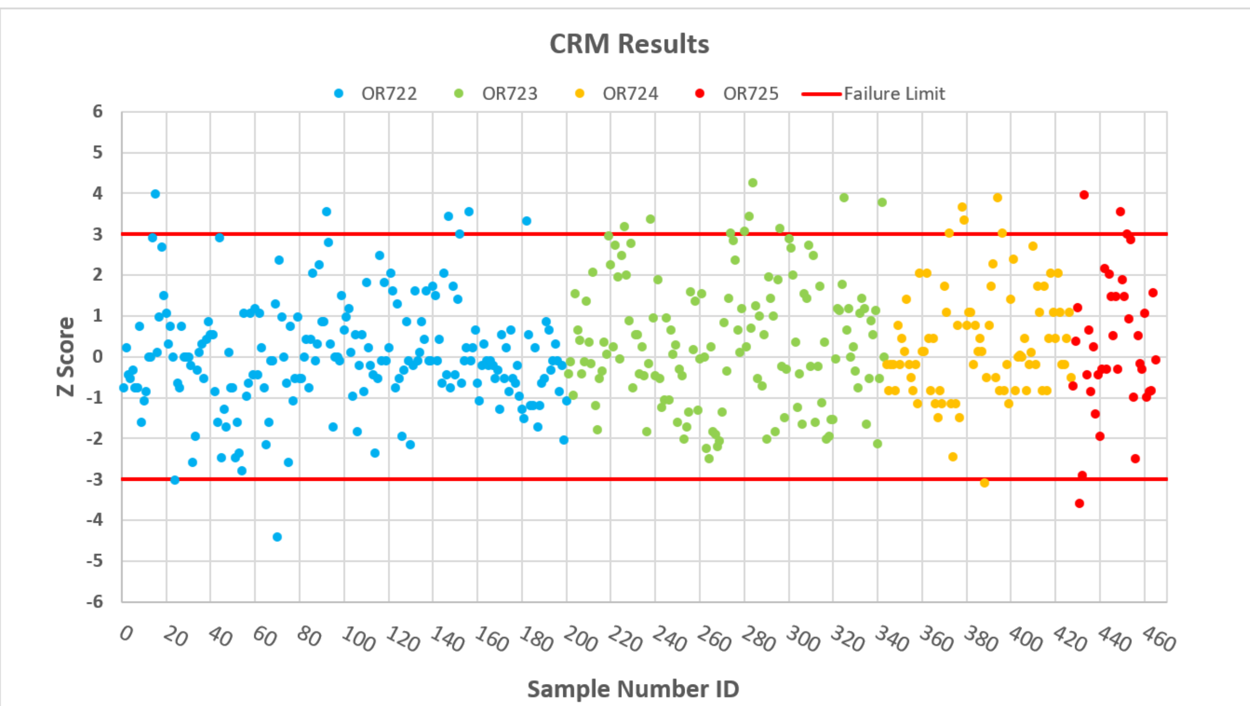


Figure 11-2: CRM Results

11.4.3 SRC Internal QA/QC Program

Quality control was maintained for all analytical apparatus at SRC with certified reference material used to track analytical drift, and data accuracy and precision. Independently of Abasca's QA/QC samples, standards were inserted into sample batches at regular intervals by SRC. Any certificates that include a standard sample that fall outside of 3 standard deviations (SD) is returned for re-analysis. As well, any certificate that includes two successive samples that fall outside 2 SD are also returned for re-analysis. All laboratory control samples fall within control limits.

11.5 Security

As each hole was drilled, drilling contractor personnel placed the core in boxes at the drill site and secured core boxes with lids secured to the box. Core was then delivered to the core processing facility twice a day. All core was logged, sampled and stored at the camp's logging facilities. On site sample preparation consists of core splitting by geological technicians under the supervision of geologists. The bags containing the split samples are then placed in buckets with lids for transport.

All samples are driven by Abasca personnel to SRC. If appropriate, the samples were accompanied by Transport of Dangerous Goods (TDG) documentation completed by qualified personnel. A request for analysis form was prepared prior to shipment, detailing each batch of samples, sample types, preparation codes, and analysis codes.

Samples were received at SRC either as dangerous goods requiring appropriate TDG documentation or as exclusive-use samples (with no radioactivity documentation attached). Upon arrival, all information pertaining to a received shipment of samples is verified by SRC personnel, including sample numbers, number of pails, sample type/matrix, condition of samples, and request for analysis. After the completion of analyses, data are sent securely via electronic transmission to Abasca. These results are provided as a series of PDFs and an Excel spreadsheet.

SRC places a large emphasis on confidentiality and data security. Appropriate steps are taken to protect the integrity of samples at all processing stages. Access to the SRC premises is restricted and monitored.

In reviewing previous documentation on the project, the Author is of the opinion that sample handling, shipment, and security for samples collected in 2016 was completed in a similar manner to that of the Abasca procedure outlined above.

11.6 QP Comment

There were no significant issues identified concerning sample preparation, shipments or sample security for the 2016 to 2024 drill programs, which constitute the basis for the Mineral Resource Estimate enclosed in this report.

The QP has reviewed the 2016 to 2026 data and is of the opinion that the procedures and systems employed to collect and manage this information meet or exceed industry standard practice, and that the QA/QC results demonstrate acceptable levels of accuracy and precision at the laboratories. In the Author's opinion the results indicate that the database is suitable for the purposes of this Technical Report.

12.0 DATA VERIFICATION

12.1 Site Visits and Core Review

Mining and Infrastructure QP Site Visit

Dr. Sabry Hafez, P.Eng., PhD, of Tetra Tech Canada Inc., conducted a personal inspection of the Key Lake South Project, including the Loki Flake Graphite Deposit, on June 18, 2026.

The purpose of the inspection was to obtain familiarity with the Project and evaluate site conditions relevant to the conceptual open-pit mine plan, proposed site infrastructure, capital and operating cost estimates, and economic analysis presented in this Technical Report.

During the personal inspection, Dr. Hafez:

- inspected the general Project area;
- reviewed access to the Project from Provincial Highway 914;
- inspected the general locations of the proposed open pit, processing plant, waste-rock and tailings facility, and supporting infrastructure;
- observed the local topography, surface-water features, vegetation, overburden conditions and other physical constraints relevant to the proposed Project layout;
- reviewed the potential locations and access requirements for mine haul roads and other Project roads;
- considered the proximity of the Project to existing regional infrastructure, including the Key Lake mill area and provincial power infrastructure; and
- discussed the Project's exploration history, proposed mine development, infrastructure requirements, environmental studies and future work programs with representatives of Abasca Resources Inc.

Based on the personal inspection and verification activities described above, Dr. Hafez considers the information supporting the mining, mine planning and economic-analysis portions of the PEA to be adequate for the purposes of this Technical Report.

Geology QP Site Visit and Core Review

A site visit to the Project was carried out January 28 to 29, 2025, by UMR's QP, Matt Batty, MSc, P. Geo. The two-day site visit included:

- Review of drill core from an ongoing drillhole,
- Review of mineralized drill core from two drillholes completed in 2024,
- Confirmation of three drillhole collar locations,
- Review and verification of the geological setting / environment of the Project,
- Review of drilling, logging, sampling, analytical and QA/QC procedures, and
- Review of overall site facilities.

Mr. Batty reviewed the entirety of available core from KLS-25-056 (0 to 233.0 m), which was being drilled at the time of the visit, and mineralized intervals from drillholes KLS-24-037 (79.7 to 125.9 m) and KLS-24-039 (156.5 to 204.9 m) (Figure 12-1). The selected drillholes provided examples of low- and high-grade graphite mineralization, an overall sense of the Project's geology, and spatial representation. A comparison of the drill logs and assay results with the drill core showed that the information recorded in the drill database matched well with the drill core. As part of the review, the QP verified the occurrences of mineralization visually (Figure 12-2).

Figure 12-1: Loki Core Review

Figure 12-2: Visual Inspection of Mineralization from KLS-24-037

The locations of three drillhole collars were confirmed visually and with a handheld Garmin GPS, inclusive to KLS-25-053, KLS-25-056, and KLS-25-057. The database records were within 3 metres of the less accurate handheld measurements; and therefore, were deemed acceptable. The collar locations for the KLS-25-053 and KLS-25-056, were demarked with tree branches or inserted into the ground near the drill collar (Figure 12-3). KLS-25-057 was actively being drilled during the site visit and the collar location was recorded at the drill head.

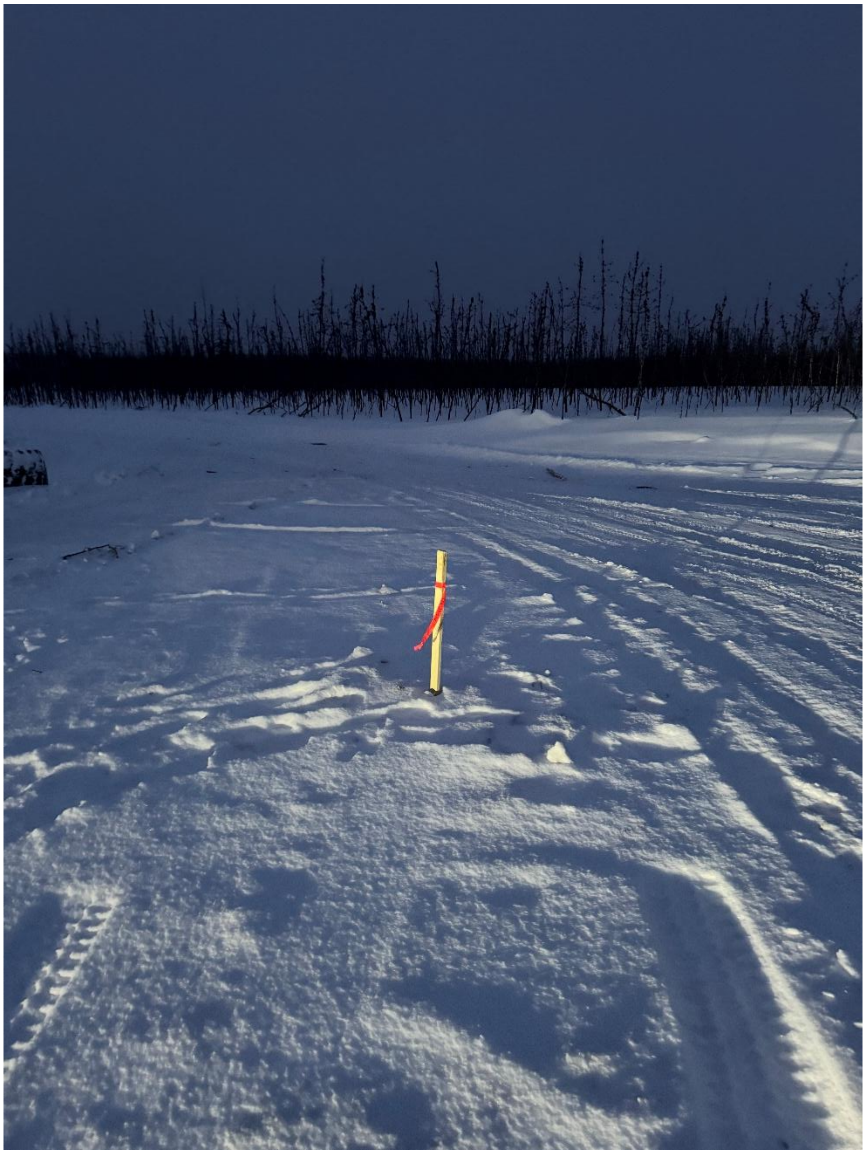


Figure 12-3: Labelled Picket Demarcating KLS-25-053 Collar

12.2 Database Validation

The QP validated the diamond drilling database via the following digital queries:

- 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.
- Lithology, alteration, and structure tables: 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, density, and assay tables: 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

The assay table contains 12,981 entries laboratory records. The author verified 353 records from 8 drill holes against 8 different laboratory certificates. No major discrepancies were found.

12.4 Validation Limitations and Adequacy of the Data

The QP reviewed the adequacy of the exploration information and the visual, physical, and geological characteristics of the mineralization of the Project and found no significant issues or inconsistencies that would cause one to question the validity of the data provided by Abasca.

In the QP's opinion, the project exploration data are free of any material or systematic errors, well validated and of sufficient quality for use in this Technical Report.

Based upon the QP's evaluation of the drilling, sampling and QA/QC programs completed by Abasca, which meet or exceed industry standards, and based on the QP's own validation of the data, it is Mr. Batty's opinion that the Loki drill and assay data are appropriate for use as presented in this Technical Report.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Abasca plans to build a new plant to process graphite mineralization mined from an open pit at the Deposit. The main production target is to produce 65,000 t/a of graphite concentrate with a grade of 95% Cg.

A preliminary test program for recovery of the graphite from the Deposit was conducted by SGS Lakefield during 2025 and 2026. The test program included chemical characterization, mineralogical analyses, grindability testing, and flotation testing.

A shipment of samples, representing three areas (Comp 1, Comp 2, and Comp 3) of the Deposit as defined by Abasca, was sent to SGS in 2025 for the testwork. Four composite samples, labelled as Comp 1, Comp 2, Comp 3, and Master Composite, were prepared.

The master composite was formed by three area composites at the same weight ratio for comminution testing. A subsample was taken for a Bond abrasion test. The remainder was crushed to 100% passing 12.7 mm for a Bond rod mill grindability test, then crushed to 100% passing 3.36 mm (6 mesh) for a Bond ball mill grindability test.

All samples for the flotation tests were crushed to 100% passing 3.36 mm and split into 2 kg test charges. The lab results indicate that the cleaner flotation on the fine fraction rougher concentrate produces better metallurgical performances in upgrading graphite concentrate. It is possible to achieve a concentrate with a greater than 95% graphite grade, though the coarse cleaner circuit produces a graphite concentrate with a graphite grade of 95% under the tested flotation conditions.

13.1 Test Results

13.1.1 Head Assay

The results of chemical analyses (total and graphite carbon, sulphur, and ICP-OES) on the samples are presented in Table 13-1.

Table 13-1: Results of Chemical Analysis

	Master Composite	Comp 1	Comp 2	Comp 3
C(t), %	8.24	8.97	7.34	8.46
C(g), %	8.01	8.88	7.12	8.36
TOC, %	<0.05	<0.05	0.12	0.39
TIC, %	0.24	0.05	0.22	0.41
S, %	1.23	1.76	1.09	1.09
S2-, %	1.22	1.78	1.09	1.02
C(g):C(t), %	97.2	99.0	96.9	98.8

The graphitic carbon is the dominant carbon species in all the samples and grades in the range of 7.12% to 8.88%.

13.1.2 Mineralogical Analysis

Four samples, referred to as Comp 1, Comp 2, Comp 3, and Master Comp, were analyzed for mineralogical examination by the SGS Advanced Mineralogy Facility. The four samples are dominated by non-sulphide gangue minerals (NSG), and lesser amounts of graphite, pyrrhotite, pyrite, chalcopyrite, and Fe-oxides. The graphite grains in this composite range from 5 to 800 µm for the Master Comp, from 5 to 1600 µm for Comp 1 and from 5 to 1200 µm for Comp 2 and Comp 3. Because the as-received samples are coarse in particle size, the liberation analysis of graphite grains showed poor graphite liberation at the as-received particle sizes.

For the Master Comp sample, the analysis indicates that while individual elongated, platy, and acicular flakes exist, the dominant feature is the presence of large, massive aggregates made up of thousands of tightly packed micro-flakes. The graphite grains in this composite range from 5 to 800 µm. The sample shows classic, strongly foliated textures with perfectly aligned bands interleaved with NSG. There are also highly chaotic, deformed structures where the graphite is twisted, torn, bent, and crenulated, giving graphite a "shredded" appearance. Graphite exhibits extremely poor liberation and is intimately locked with gangue on a fine microscopic scale. The sample also features complex, stringy networks where graphite, NSG, and pyrite are intergrown along distinct veinlets or shear planes. Figure 13-1 shows photomicrographic images. Graphite liberation analysis indicates 30% liberated, 25% exposed, and 45% locked at the as-received particle sizes.

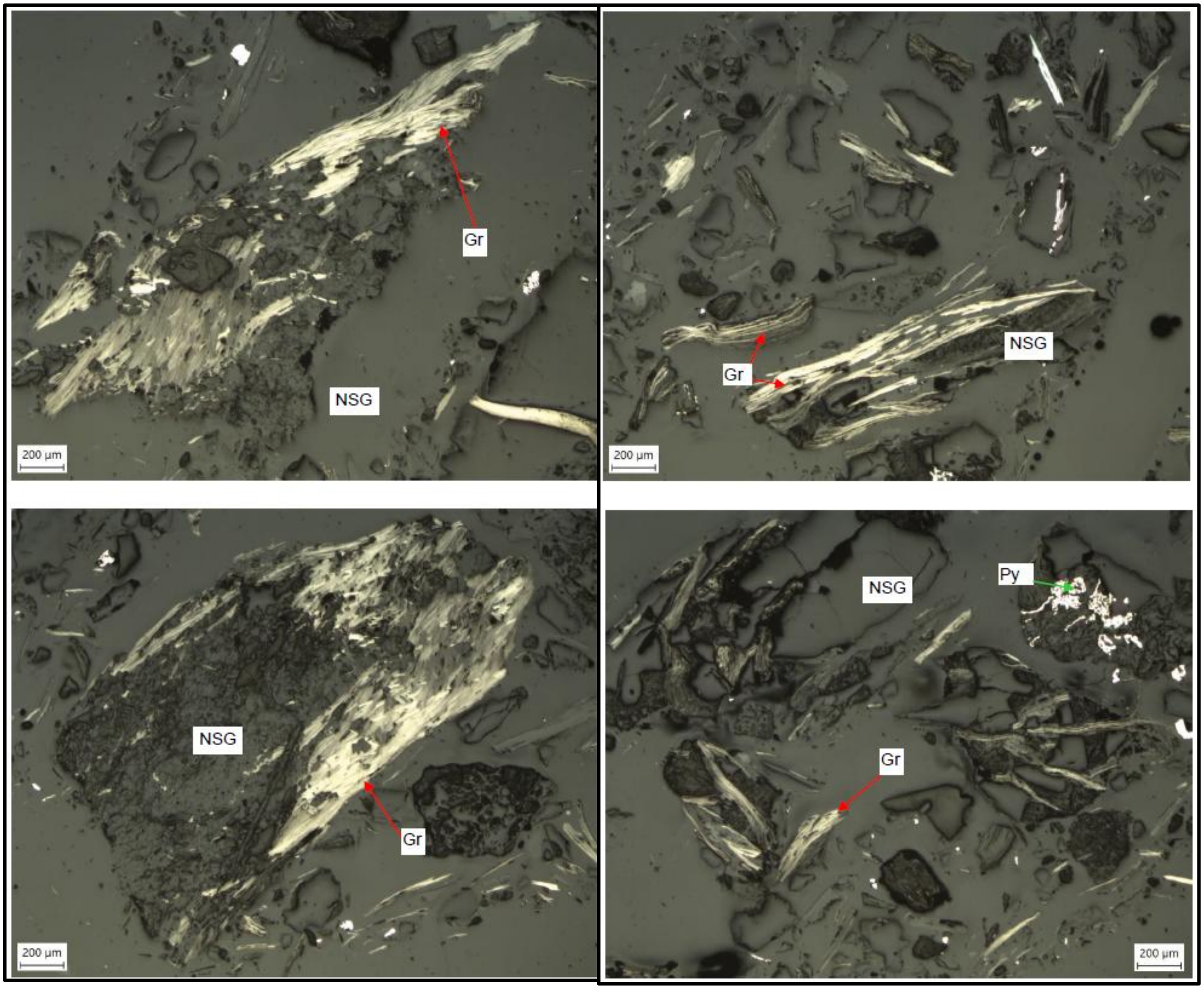


Figure 13-1: Photomicrographs from the Optical Microscope from the Master Comp

13.1.3 Comminution Tests

The results of comminution tests on the Master Composite are summarized in Table 13-2. The Rod Mill Work Index (RWi) at 8.0 kWh/t places the mineralization into the very soft category at the 5th percentile of hardness of more than 4,000 samples tested by SGS, while the Ball Mill Work Index (BWi) at 13.6 kWh/t places the mineralization into the medium-soft hardness category at the 42nd percentile of hardness of more than 9,500 samples tested by SGS.

The Bond Abrasion Index (Ai) was measured at 0.061 g which can class the mineralization into the category of low abrasive materials.

Table 13-2: Summary of Comminution Test Results

Sample	RWI	BWI	AI
	kWh/t	kWh/t	g
Master Composite	8	13.6	0.061

13.1.4 Flotation Tests

A flotation test program was conducted to assess the amenability of the mineralization to conventional flotation. Reagents used were fuel oil as a graphite collector and methyl isobutyl carbinol (MIBC) as a frother.

The flowsheet for rougher flotation tests is presented in Figure 13-2, and the results of the rougher flotation tests are summarized in Table 13-3.

Table 13-3: Results of Rougher Flotation Test

Test	Sample	Flotation Reagent	Particle Size	Product	Yield	Assay	Cg Recovery
			80% Passing, µm		%wt	%, Cg	%
F01	Master Sample	Fuel Oil 90 g/t MIBC 50 g/t	361	Flash Concentrate	11.1	40.01	49.97
				Rougher Concentrate	28.4	15.43	49.31
			254	Tailings	60.6	0.12	0.82
			297	Feed	100	8.89	100.00
F02	Master Sample	Fuel Oil 90 g/t MIBC 50 g/t	248	Flash Conc	8.7	36.3	39.18
			329	Rougher Conc	30.9	15.7	60.19
			197	Tailings	60.4	0.06	0.45
			242	Feed	100	8.06	100.00

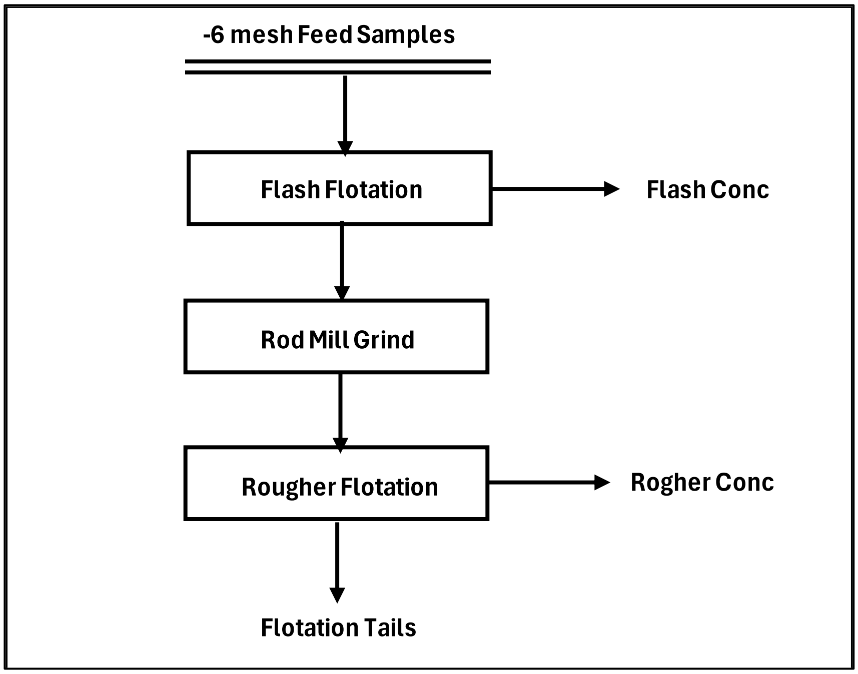


Figure 13-2: Flowsheet for Rougher Flotation Tests

The test results indicate that:

- At the grinding fineness of around P80 300 µm, more than 99% graphitic carbon can be recovered in the rougher concentrate.
- For 6 mesh master samples, more than 40% graphitic carbon can be concentrated into the flash flotation froth.
- It is also noticed that the performances of the flash flotation were not stable.

Because larger graphite flakes have a much higher selling price, the flowsheets with multi-stage regrinding and multi-stage cleaning were tested on the combined flash and rougher flotation concentrates, in an effort to maximize graphite concentrate purity while protecting flake size.

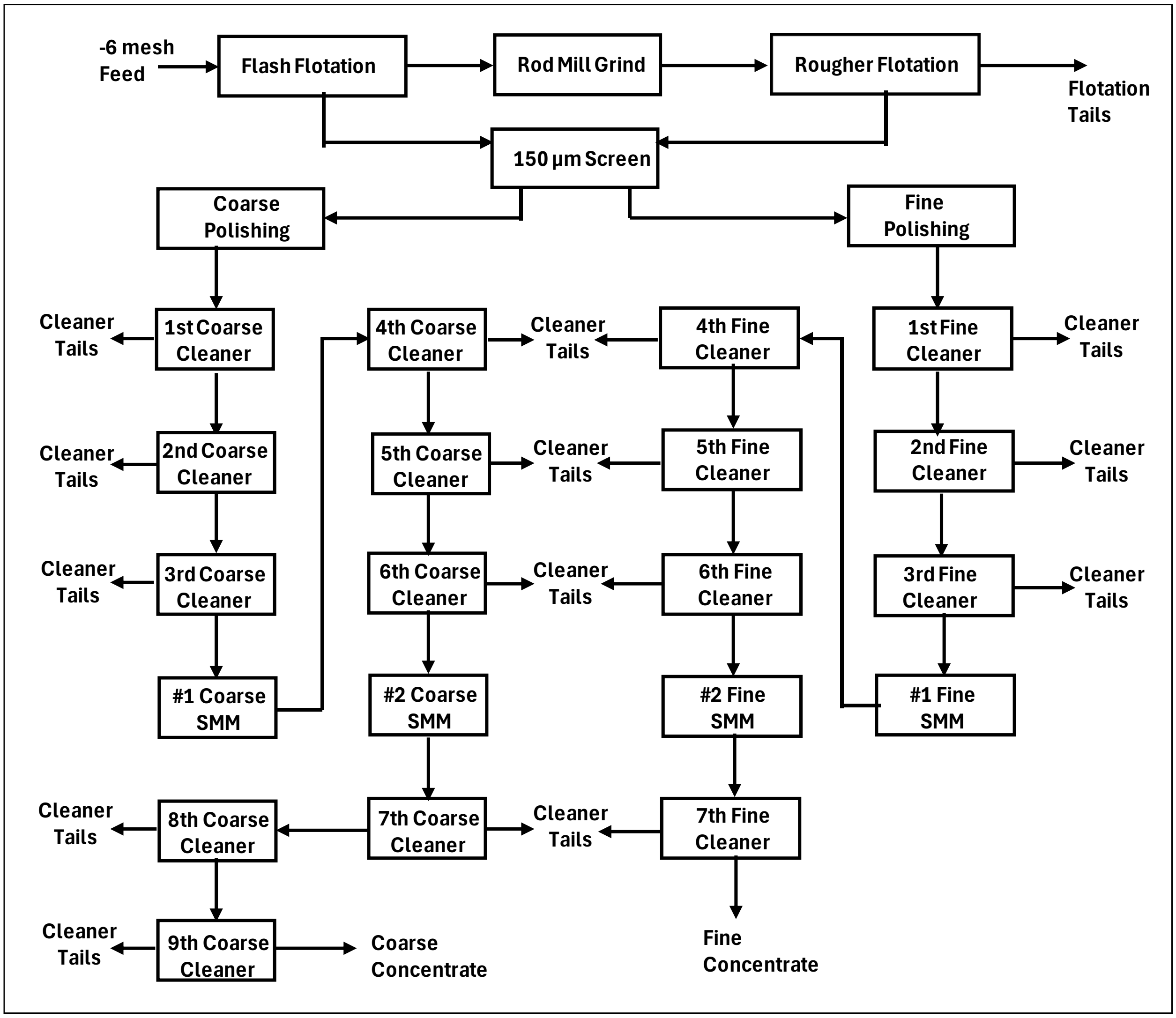
Furthermore, two stages of stirred media mill with ceramic media were also tested in the cleaner regrinding circuit to minimize damage to the graphite particle size.

An open-circuit flotation test was carried out to examine the effect of polishing grind and stirred media grind on flotation response. The flowsheet is presented in Figure 13-3. The cleaner flotation was separately conducted on two different particle sizes, coarser than 150 um and finer than 150 um fractions, split from the combined flash and rougher flotation concentrates. The results of the open-circuit flotation test are summarized in Table 13-4.

The test results indicate that:

- The yield of the combined flash and rougher flotation concentrate was approximately 40% with a graphite grade of 20.7%.
- There was no significant difference in the graphite grade between the coarse cleaner feed (+150 µm) and the fine cleaner feed (-150 µm).

- It seems that at the tested flotation conditions, it would be difficult for the coarse cleaner circuit to produce a graphite concentrate with the graphite grade greater than 95%.



SMM: Stirred Mill Milling

Figure 13-3: Flowsheet for Open-Circuit Flotation Test

Table 13-4: Results of Open-Circuit Flotation Test

Flotation Reagents	Size P80 µm	Product	Yield %wt	Assay %, Cg	Cg Recovery %
Fuel Oil 100 g/t MIBC 50 g/t		Flash Concentrate + Rougher Concentrate	39.58	20.74	99.22
		Scavenger Concentrate	2.14	1.12	0.29
	224	Flotation Tailings	58.28	0.07	0.49
		Feed	100.00	8.27	100.00

TETRA TECH

Flotation Reagents	Size P80 µm	Product	Yield %wt	Assay %, Cg	Cg Recovery %
		Feed to 150 µm Screen	39.58	20.74	99.22
		Coarse Polishing Grinding	28.37	20.65	70.80
Fuel Oil 20 g/t MIBC 20 g/t		Coarse 1st Cleaner Concentrate	19.02	30.62	70.37
		Coarse1st Cleaner Tailings	9.36	0.38	0.43
Fuel Oil 5 g/t MIBC 0 g/t		Coarse 2nd Cleaner Concentrate	17.40	33.18	69.76
		Coarse 2nd Cleaner Tailings	1.62	3.11	0.61
Fuel Oil 5 g/t MIBC 0 g/t		Coarse 3rd Cleaner Concentrate	15.15	37.25	68.20
		Coarse 3rd Cleaner Tailings	2.25	5.75	1.56
		#1 Coarse SMM Feed	15.15	37.25	68.20
Fuel Oil 20 g/t MIBC 5 g/t		Coarse 4th Cleaner Concentrate	11.76	47.76	67.87
		Coarse 4th Cleaner Tailings	3.39	0.79	0.32
Fuel Oil 15 g/t MIBC 5 g/t		Coarse 5th Cleaner Concentrate	7.64	71.37	65.85
		Coarse 5th Cleaner Tailings	4.12	4.06	2.02
Fuel Oil 15 g/t MIBC 5 g/t		Coarse 6th Cleaner Concentrate	6.99	77.26	65.29
		Coarse 6th Cleaner Tailings	0.64	7.23	0.56
		#2 Coarse SMM Feed	6.99	77.26	65.29
Fuel Oil 20 g/t MIBC 5 g/t		Coarse 7th Cleaner Concentrate	6.31	85.33	65.03
		Coarse 7th Cleaner Tailings	0.69	3.14	0.26
Fuel Oil 20 g/t MIBC 5 g/t		Coarse 8th Cleaner Concentrate	6.02	88.90	64.73
		Coarse 8th Cleaner Tailings	0.28	8.78	0.30
Fuel Oil 10 g/t MIBC 5 g/t	233	**Coarse 9th Cleaner Concentrate**	5.87	91.00	64.55
		Coarse 9th Cleaner Tailings	0.16	9.81	0.18
		Fine Polishing Grinding	11.21	20.98	28.42
Fuel Oil 30 g/t MIBC 30 g/t		Fine 1st Cleaner Concentrate	5.00	44.20	26.72
		Fine 1st Cleaner Tailings	6.21	2.27	1.70
Fuel Oil 15 g/t MIBC 10 g/t		Fine 2nd Cleaner Concentrate	4.09	52.77	26.07
		Fine 2nd Cleaner Tailings	0.91	5.84	0.64
Fuel Oil 15 g/t MIBC 5 g/t		Fine 3rd Cleaner Concentrate	3.68	57.68	25.67
		Fine 3rd Cleaner Tailings	0.41	8.22	0.40
		#1 Fine SMM Feed	3.68	57.68	25.67
		Fine 4th Cleaner Concentrate	2.67	76.73	24.75

Flotation Reagents	Size P80 µm	Product	Yield %wt	Assay %, Cg	Cg Recovery %
Fuel Oil 20 g/t MIBC 20 g/t		Fine 4th Cleaner Tailings	1.01	7.52	0.92
Fuel Oil 15 g/t MIBC 5 g/t		Fine 5th Cleaner Concentrate	2.48	81.56	24.43
		Fine 5th Cleaner Tailings	0.19	14.00	0.32
Fuel Oil 15 g/t MIBC 5 g/t		Fine 6th Cleaner Concentrate	2.23	89.83	24.18
		Fine 6th Cleaner Tailings	0.25	8.13	0.25
		#2 Fine SMM Feed	2.23	89.83	24.18
Fuel Oil 20 g/t MIBC 0 g/t	141	**Fine 7th Cleaner Concentrate**	2.08	94.30	23.73
		Fine 7th Cleaner Tailings	0.15	25.80	0.45

Figure 13-4 and Figure 13-5 show the changes in the graphite grades of flotation concentrates and tailings with the cleaner stages.

The results indicate that the cleaner flotation on the fine fraction rougher concentrate produces better metallurgical performances in upgrading graphite concentrate. It is possible to achieve a concentrate with a greater than 95% graphite grade.

For the coarse cleaner circuit, although the grades of cleaner tailings were low, especially for the last stages of cleaning flotation, the grades of the cleaner concentrates were relatively low. It should be noted that coarse graphite flakes may experience inferior flotability using the small size laboratory flotation cells. Large scale tests should be conducted to verify coarse graphite flake flotation performances.

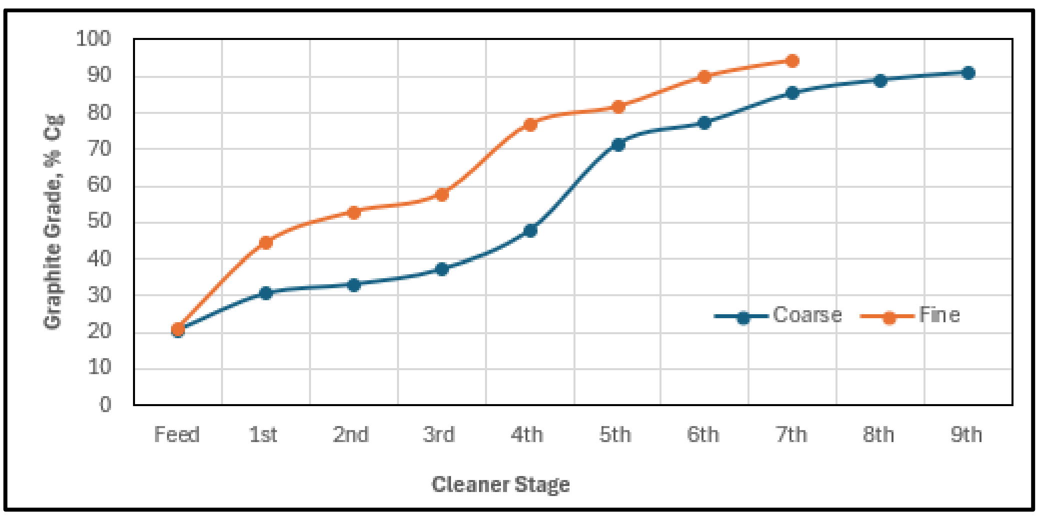


Figure 13-4: Cleaner Concentrate Grade vs Cleaner Flotation Stage

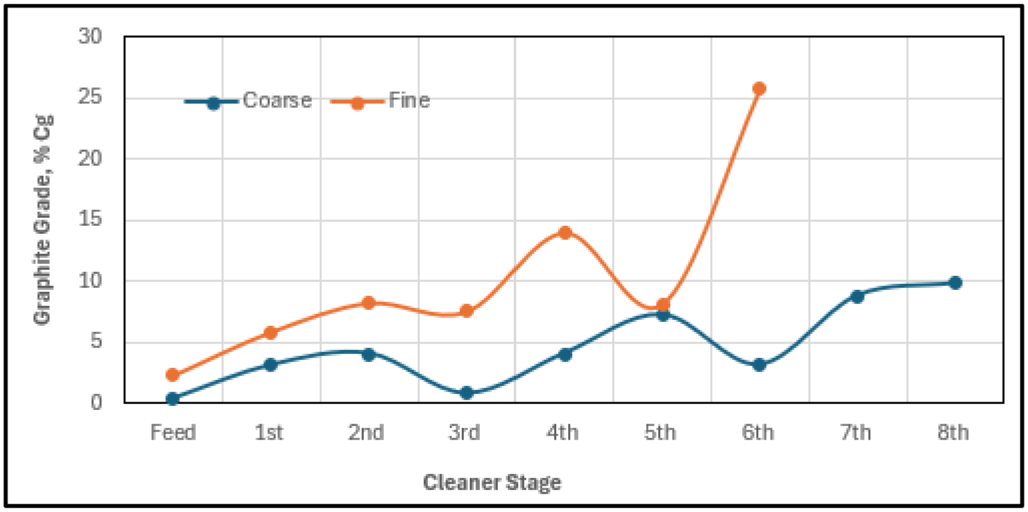


Figure 13-5: Cleaner Tailings Grade vs Cleaner Flotation Stage

13.1.5 Concentrate and Tailings Chemical Analysis

Detailed chemical analysis was conducted on the final concentrate and various tailings produced from the metallurgical test work. The assay results are shown in Table 13-5 to Table 13-7.

Table 13-5: Concentrate Multi-Element Assay

Method/Element	Blended Concentrate	Method/Element	Blended Concentrate
Elemental Combustion		Fe, g/t	3,320
C(t), %	**94.8**	K, g/t	2,104
S, %	< 0.1	Li, g/t	< 20
S^{2-}, %	< 0.05	Mg, g/t	930
Ash, %	5.32	Mn, g/t	25.9
ISE Probe		Mo, g/t	24
Cl, g/t	23	Na, g/t	< 100
F, %	0.007	Ni, g/t	18
ICP-OES		P, g/t	< 50
Ag, g/t	< 0.8	Pb, g/t	< 20
Al, g/t	8,880	Sb, g/t	< 10
As, g/t	< 30	Se, g/t	< 30
Ba, g/t	17	Si, g/t	12,700
Be, g/t	< 0.1	Sn, g/t	< 20
Bi, g/t	< 10	Sr, g/t	1.7

Method/Element	Blended Concentrate	Method/Element	Blended Concentrate
Ca, g/t	389	Ti, g/t	44
Cd, g/t	< 0.9	Tl, g/t	< 30
Co, g/t	< 3	V, g/t	5
Cr, g/t	63	Y, g/t	< 0.5
Cu, g/t	49	Zn, g/t	29

The major impurities in the concentrate are silicon (Si), aluminum (Al), iron (Fe), potassium (K), magnesium (Mg), calcium (Ca), and sulphur (S) with other minor elements. It is expected that the impurities can be removed or reduced by further purification treatments. Multi-element, whole rock, modified ABA (Acid-Base Accounting) and shake flask extraction analyses were conducted on several tailings samples produced from the flotation tests. The whole rock and ABA analysis results are shown in Table 13-6 and Table 13-7.

Table 13-6: Whole Rock Analysis Results – Tailings Samples

Parameter	Unit	Master Comp F22 Ro Tail	Comp 1 F27 Ro Tail	Comp 2 F28 Ro Tail	Comp 3 F29 Ro Tail	Master Comp Comb 1st Cl Tail	Comp 1 Comb 1st Cl Tail	Comp 2 Comb 1st Cl Tail	Comp 3 Comb 1st Cl Tail
SiO_2	%	65.3	65.7	65.1	65.9	59.3	56.0	61.8	55.6
Al_2O_3	%	12.7	12.7	12.9	12.2	15.9	17.7	14.4	17.8
Fe_2O_3	%	7.97	7.93	8.05	7.75	8.53	9.22	9.07	8.59
MgO	%	2.45	2.49	2.80	2.14	2.77	2.91	3.2	2.35
CaO	%	0.81	0.75	0.88	0.5	0.85	0.88	0.53	0.60
Na_2O	%	0.33	0.30	0.26	0.16	0.31	0.31	0.25	0.15
K_2O	%	3.29	3.29	3.54	3.27	3.21	3.42	3.48	3.11
TiO_2	%	1.03	0.98	0.99	0.95	0.94	0.85	0.95	0.90
P_2O_5	%	0.13	0.10	0.10	0.08	0.08	0.07	0.07	0.09
MnO	%	0.08	0.08	0.08	0.07	0.08	0.08	0.08	0.06
Cr_2O_3	%	0.02	0.01	0.02	0.02	0.02	0.06	0.04	0.04
V_2O_5	%	0.02	0.03	0.03	0.04	0.03	0.03	0.03	0.03
LOI	%	5.83	5.81	5.44	7.15	8.12	8.55	6.28	10.7
Sum	%	99.9	100.2	100.2	100.2	100.1	100.1	100.2	100

Table 13-7: ABA Analysis Results – Tailings Samples

Parameter	Unit	Master Comp F22 Ro Tail	Comp 1 F27 Ro Tail	Comp 2 F28 Ro Tail	Comp 3 F29 Ro Tail	Master Comp Comb 1st Cl Tail	Comp 1 Comb 1st Cl Tail	Comp 2 Comb 1st Cl Tail	Comp 3 Comb 1st Cl Tail
Paste pH		7.98	7.44	7.27	7.39	7.60	7.32	7.22	7.37
Fizz Rate		2	2	2	2	2	2	2	2
Sample weight	g	1.99	1.99	2.02	2.00	2.01	2.01	2.01	2.00

Parameter	Unit	Master Comp F22 Ro Tail	Comp 1 F27 Ro Tail	Comp 2 F28 Ro Tail	Comp 3 F29 Ro Tail	Master Comp Comb 1st Cl Tail	Comp 1 Comb 1st Cl Tail	Comp 2 Comb 1st Cl Tail	Comp 3 Comb 1st Cl Tail
HCl added	mL	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00
HCl	Normality	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10
NaOH	Normality	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10
Vol NaOH to	mL	17.35	17.22	16.38	17.09	17.10	17.80	17.35	17.75
Final pH	no unit	1.38	1.40	1.38	1.49	1.49	1.65	1.49	1.86
NP	t $CaCO_3$/1000 t	6.7	7.0	9.0	7.3	7.2	5.5	6.6	5.6
AP	t $CaCO_3$/1000 t	52.8	51.4	42.7	41.3	41.1	38.9	37.2	33.8
Net NP	t $CaCO_3$/1000 t	-46.1	-44.41	-33.69	-34.04	-33.89	-33.44	-30.56	-28.24
NP/AP	ratio	0.13	0.14	0.21	0.18	0.18	0.14	0.18	0.17
S	%	1.62	1.63	1.37	1.32	1.48	1.29	1.15	1.04
Acid Leachable	%	< 0.04	< 0.04	< 0.04	< 0.04	0.17	0.05	< 0.04	< 0.04
Sulphide	%	1.69	1.64	1.37	1.32	1.32	1.25	1.19	1.08
C	%	0.440	0.362	0.322	0.946	1.01	0.811	0.401	1.12
CO_3	%	1.01	1.26	1.11	2.12	0.86	1.42	1.09	2.17
C(g)	%	0.18	0.09	0.10	0.48	0.78	0.46	0.17	0.65
CO_3 NP	t $CaCO_3$/1000 t	16.77	20.92	18.43	35.19	14.28	23.57	18.09	36.02
Net CO_3 NP	t $CaCO_3$/1000 t	-36.03	-30.48	-24.27	-6.11	-26.82	-15.33	-19.11	2.22
CO_3 NP/AP	ratio	0.32	0.41	0.43	0.85	0.35	0.61	0.49	1.07

The main mineral components in the flotation tailings samples are SiO_2, Al_2O_3, Fe_2O_3, K_2O and MgO. The ABA determination results indicate that the graphite flotation tailings are expected to be potential acid generators. The shake flask leach tests show that the metal levels in the solutions extracted from the procedure are expected to be lower than the general water discharge levels, however, some of the metal concentrations may exceed the requirements for some special usages, such as for aquatic life (surface water).

Further test work should be conducted to optimize the flotation conditions and improve graphite flotation performances, including conservation of coarse graphite flakes. Also, additional purification test work on flotation concentrates should be conducted to understand the optimum purification treatments, including ultra-purity graphite chemical analysis. The recommended test work is summarized in Section 26.0.

14.0 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The updated Mineral Resource estimate for the Loki Flake Graphite Deposit was completed by Matt Batty, MSc, P. Geo of UMR. The effective date of the enclosed mineral resource is April 23, 2026, coincident with the date the last assay received.

One wireframe was created to represent mineralization at the Deposit and an additional five wireframes were created to characterize the local lithology. Samples were composited to 8 m lengths within the mineralized domain. The composites were reviewed for outliers and declustered, resulting in a decided representative dataset for the domain.

A block model was constructed to encompass the mineralization and lithology domains using a parent block of 20 m (strike direction) by 20 m (thickness direction) by 16 m (vertical direction) with 5 m by 5 m by 4 m sub-blocks. The blocks were populated using Ordinary Kriging (OK) as informed by the directional variogram. The lithology domains were assigned a density based on specific gravity measurements observed within the domains or observations from analogous deposits. A linear regression was used to assign density to blocks with estimated graphite grades.

The block model was validated via volume comparison, mean grade comparison, visual inspection, swath plots, and change of support comparison. The QP found the block model to reflect the geological interpretation, grade continuity to be reasonable and confirmed that the block grades were reasonably consistent with local drill hole composite grades.

The estimate is based on 79 diamond drill holes defining the mineralized domain. The Mineral Resource is composed of Indicated and Inferred Mineral Resources. The Indicated Mineral Resource totals 6.99 million tonnes at an average grade of 8.27 % Cg for a total of 0.58 million tonnes Cg, and the Inferred Mineral Resource totals 15.83 million tonnes at an average grade of 6.93 % Cg for a total of 1.10 million tonnes Cg. The Mineral Resources are reported at a cut-off grade of 2.30 % Cg within a conceptual open pit design. No Mineral Reserves have been estimated at the Project.

The reported Indicated Mineral Resources are generally supported by an approximate drill hole spacing of 50 m and a minimum of three drill holes informing the estimate, whereas the Inferred Mineral Resources are supported by wider drill spacing of 100 m with a minimum of three holes.

In Understood's opinion, the estimation methodology is consistent with standard industry practice and the Mineral Resource Estimates for Loki Deposit are reasonable and acceptable.

14.2 Source Database

The drill hole data was provided as individual spreadsheets, including collar surveys, downhole surveys, lithology logs, mineralization logs, alteration logs, structural logs, recovery logs, geochemistry data, and specific gravity data.

The collar spreadsheet composed of 247 entries that details the drill hole name, the collar locations in Universal Transverse Mercator (UTM) North America Datum 1983 (NAD 83) zone 13N coordinates, and the end depth of the drill hole. According to the collar spreadsheet, which contains historic, recent, and ongoing drilling, a total of 51,890 metres have been drilled on the Project.

The survey spreadsheet contains 3,065 records from 243 drill holes, which averages to a survey data point approximately every 16.9 m of borehole. There are four holes with no downhole survey measurements. These holes are short, historic holes that are distal to the Loki Deposit.

The lithology spreadsheet contains 2,377 logged intervals from 216 drill holes. The entries in the lithology spreadsheet define the drill hole, the interval, and the lithology log. The drill holes with no lithology logs were either drilled distal to the deposit or were incomplete at the time the data was exported.

The geochemistry spreadsheet contains 12,981 entries for an array of analytes, including graphite (Cg) assay results of mineralization sections, uranium assay results of wall rocks, and specific gravity measurements of mineralization sections. A total of 7,965 graphite (Cg) assay results are present in the dataset, with values ranging from below detection to 26.4% Cg. Uranium assays were reported using multiple analytical methods, including ppm measurements of total and partial digestion elemental uranium from ICP-MS and ICP-OES. Samples in areas of elevated radioactivity were also submitted for SRC's U3O8 Assay package by ICP-OES, and were reported as a weight percentage of U_3O_8 %. These values were consolidated into a single variable, elemental uranium ppm, to provide a consistent uranium dataset for evaluation. The resulting field contains 12,880 uranium assay entries, 99% of which have values less than 200 ppm.

A total of 1,305 specific gravity measurements are available from the water immersion method, with values ranging from 1.60 to 4.16 and an average of 2.63.

Additional to drill hole data, the QP was provided with a Digital Terrain Model (DTM) with 5 m resolution for a topography surface.

Historic boreholes KS-CC16-11 and KS-CC16-12 were destroyed in a fire and could not be resampled; thus, these two holes are excluded from the resource estimate due to incomplete assay records for graphite. The other three historic holes drilled on the deposit, KS-CC16-02, KS-CC16-13 and KS-CC16-14, were resampled and assayed for graphite and are included in the model.

14.3 Drill Hole Validation

The drill hole data spreadsheets were organized and imported into a Vulcan database and checked for the following:

- Unique collar locations
- Overlapping assays
- Empty table check for assays, collars, lithology, and surveys
- Increasing depth field in surveys, assays, lithology, and specific gravity field
- Consecutive variation tolerance (max of 30 degrees) for dip and azimuth
- Unique sample ID for assay and specific gravity measurements
- Ensure azimuth survey measurements are between 0 and 360
- Ensure dip survey measurements are between -90 and 0
- Ensure total carbon (%) grades are between 0 and 100

There are no overlapping assays, duplicate assay sample IDs, depth errors, or gross numerical errors in recorded assay grades. There are 2 holes that share collar coordinate locations with at least one other hole, but this is not in error and represents an abandoned hole that was restarted. There are 35 holes with no assay, lithology, and/or survey data, all of which are distal to the deposit or were recently drilled and awaiting assay results. The data was also reviewed in cartesian space. No significant errors were identified in the digital queries or visual review.

14.4 Geologic Domaining

The Loki Deposit is a broad graphite bearing shear-zone along an orientation of 127° azimuth and -45° dip to the southwest. The QP created one vein wireframe in Leapfrog Geo (version 2025.3.1) to constrain the estimate in the predominate orientation of mineralization (Figure 14-1). The grade-shell wireframe is modelled using a grade intercept limit equal to or greater than 5 m with a minimum grade of 1 % Cg, although small intervals (~2 to 3 m) of internal waste (e.g. < 1% Cg) are included to maintain continuity. Extension distance for the mineralized wireframes was halfway to the next hole, or approximately half the local average drill hole spacing vertically and horizontally past the last intercept. The wireframe is approximately 1700 m long in the strike direction with an upper contact 55 m below surface and extends to 335 m below surface. The thickness of the modelled graphite ranges between 10 to 55 m with an approximate average of 35 m. Concordant thinner graphite intersections are observed above the modelled domain but not modelled as the intersections did not meet the criteria for thickness and/or grade continuity.

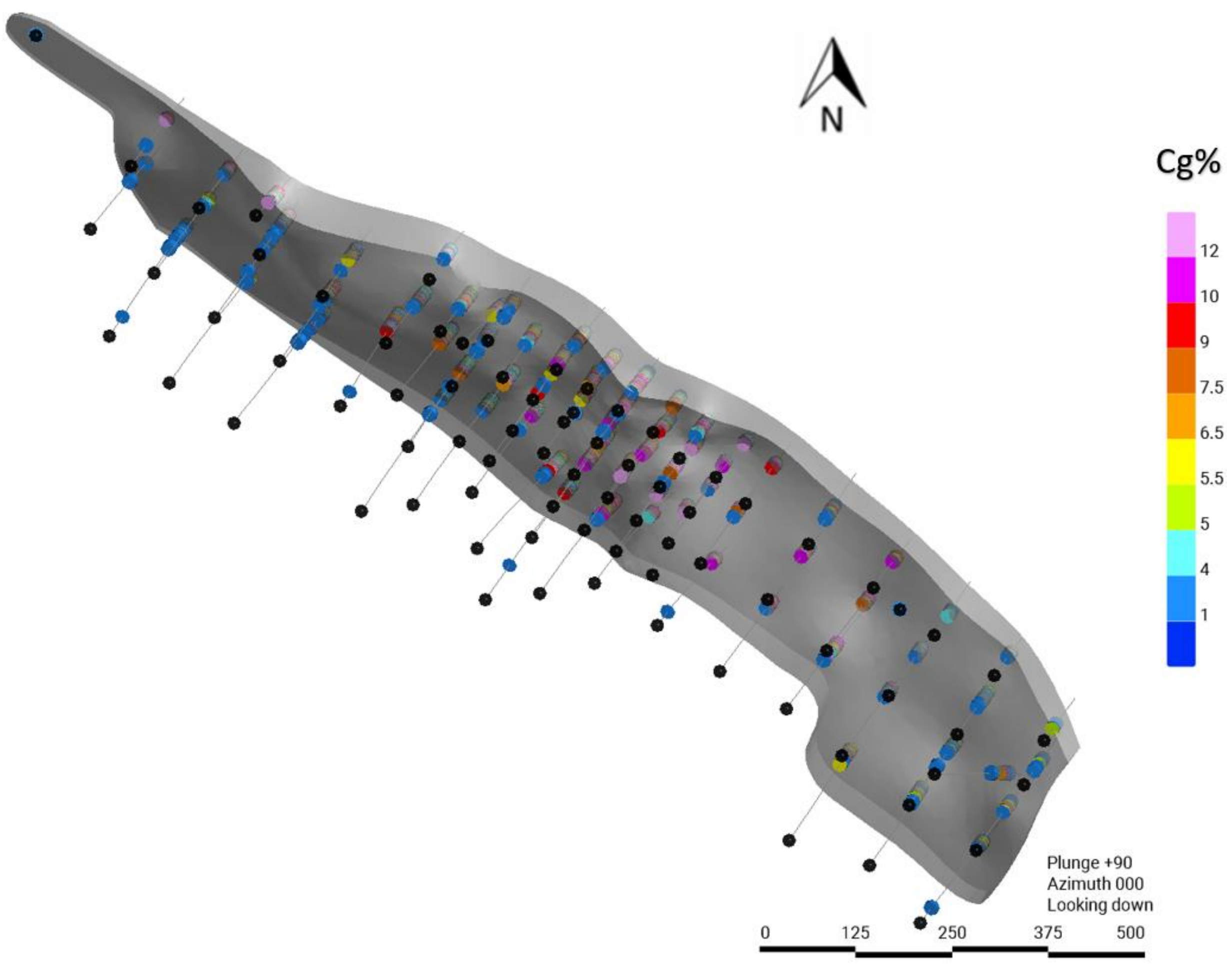


Figure 14-1: Plan View of Graphite Wireframe Interpretation underlain by Drillhole Cg Assay Grades.

A geologic model describing the bedrock to overburden contact was used to clip upper extents of graphite wireframes. Other major units observed in the deposit area were also modelled, including pegmatite units, granitic gneiss, and biotite gneiss (Figure 14-2). The modelled graphite vein follows the lower pegmatite boundary to honour this observation made in the core. All generated wireframes were exported from Leapfrog and imported into Maptek's Vulcan (version 2025.1) for estimation.

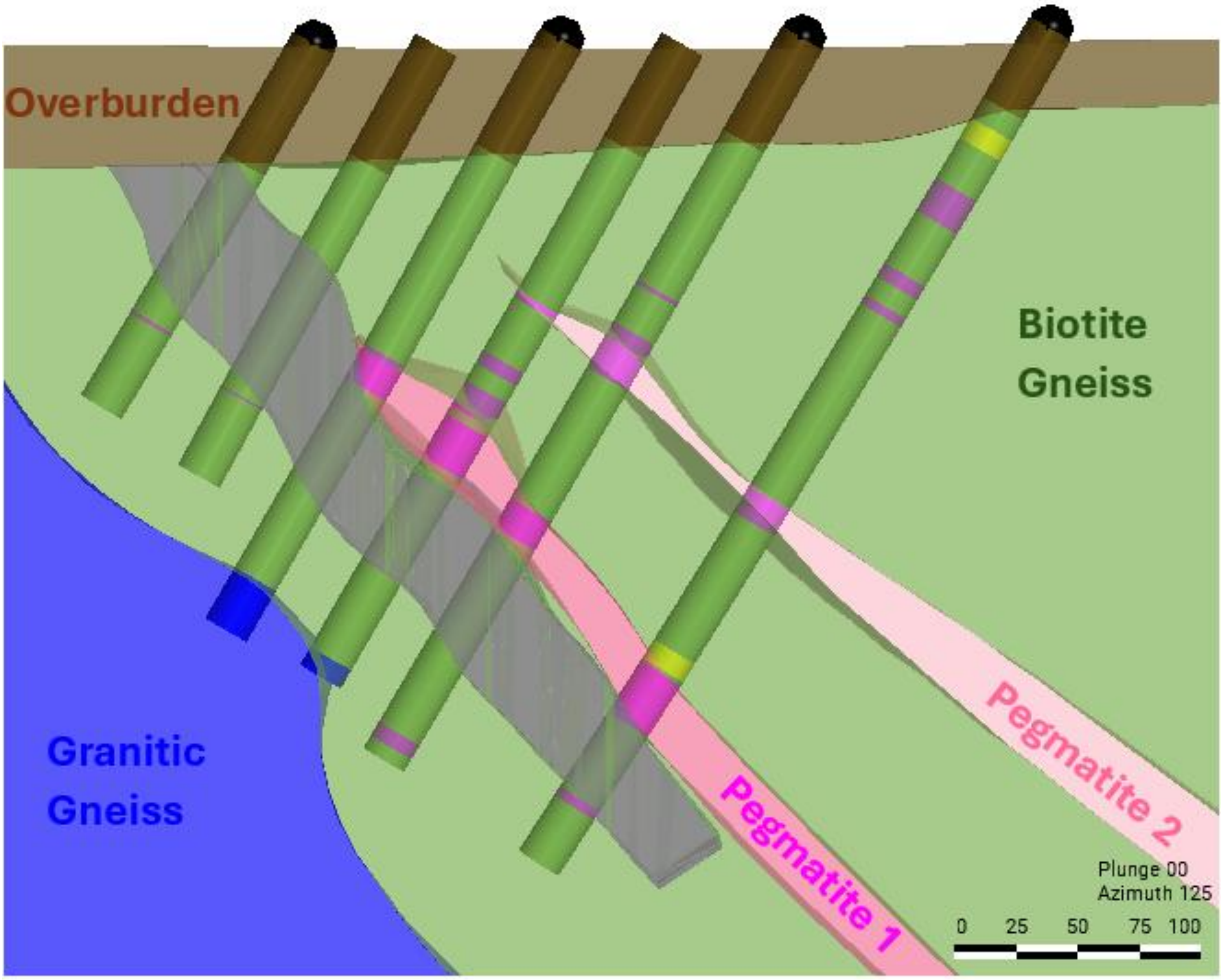


Figure 14-2: Cross-Section of Graphite and Lithology Wireframe Interpretations

14.5 Compositing and Statistical Analysis

14.5.1 Compositing

Assays were composited at 8 m lengths in Vulcan starting at the graphite wireframe boundary nearest to the collar. Composites less than the composite length, which were located at the bottom of the mineralized intercept, triggered a recompositing of the drill hole where the extra length is equally distributed to the composites along the drill hole string. Drillhole locations with no or missing values were assigned a value of 0.0 % Cg.

14.5.2 Declustering

A global representative distribution of variables is essential for unbiased resources estimation, and one step in determining a representative distribution is the consideration of the spatial arrangement of the data. Declustering techniques assign each datum a weight based on its closeness to surrounding data.

The declustering weights were created through a block model approach. A block model was created at block dimensions (4 m x 4 m x 4 m) smaller than the composites (8 m) with the blocks coded to each domain. The composites were assigned a unique ID per composite and a nearest neighbour estimation was completed on the unique IDs. The number of blocks assigned the unique ID were divided by the total blocks of the domain, resulting in the declustering weight for that composite (Figure 14-3). The weights were rescaled to be manageable for databases.

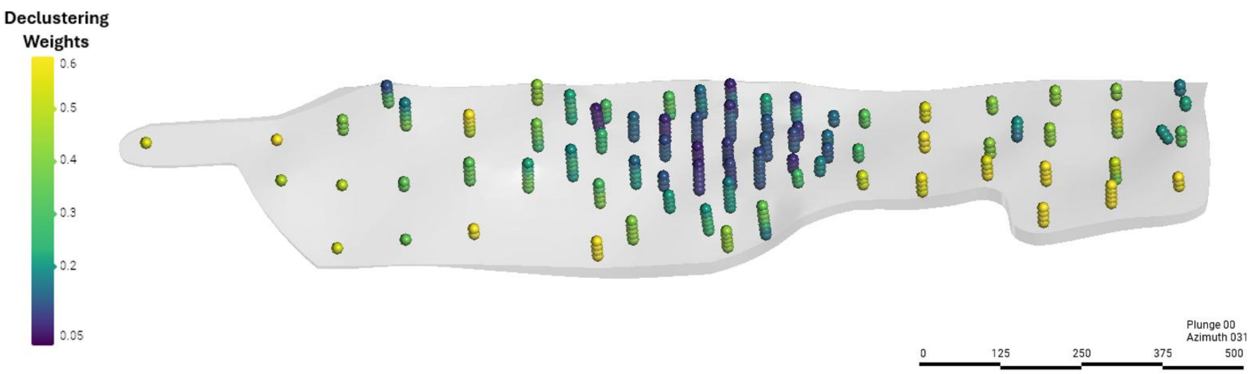


Figure 14-3: Long Section (Looking along an Azimuth of 031°) of Composites (Spheres) Displaying Declustered Weights in Graphite Domain.

14.5.3 Outlier Capping and Representative Distribution

Where assay distributions are positively skewed or approach log-normal behaviour, high-grade values can exert a disproportionate influence on the estimated mean. As such, the treatment of outliers is an important step in developing a representative dataset for Mineral Resource estimation.

The graphite composite dataset was reviewed using uncapped 8 m composites. Statistical analysis included histograms, and log-probability plots, to assess distribution characteristics, stationarity, and the presence of anomalous high-grade values. Additionally, the dataset was reviewed in cartesian space to review for local outliers. The summary statistics of the representative distributions, such as count, average, variance, and CV, for the domain is listed in Figure 14-4.

The analysis indicates that the composite dataset forms a stable population with no evidence of a distinct high-grade sub-population or erratic outliers. Based on this assessment, no direct grade capping was applied to the composite data prior to estimation.

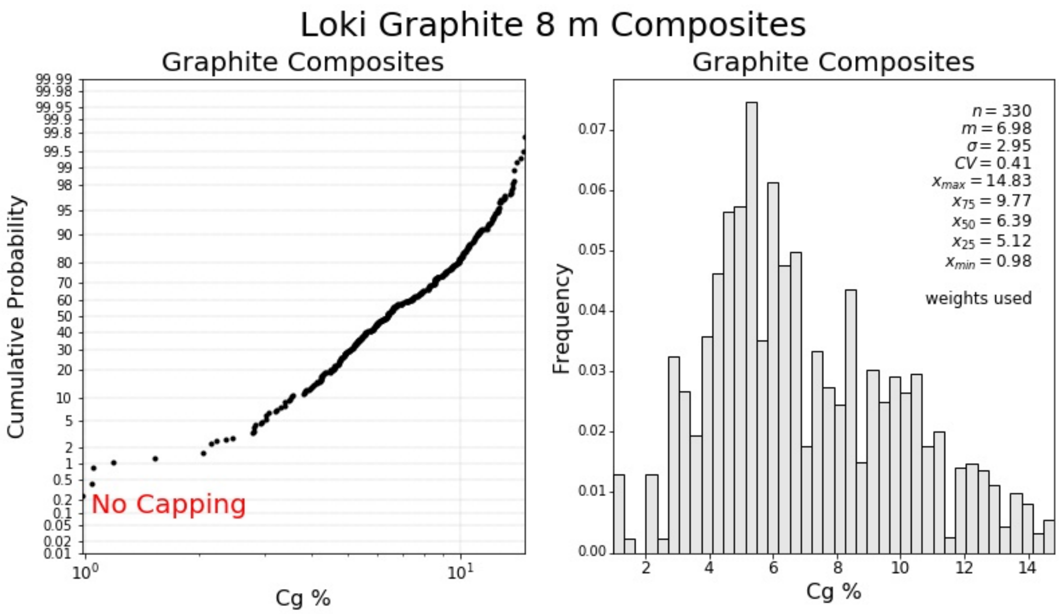


Figure 14-4: Probability Plot and Histogram of Composite Distribution of the Graphite Domain

A High Yield Limit (HYL) was implemented during estimation to control the spatial influence of higher-grade samples. The HYL approach does not modify composite grades directly; rather, it restricts the distance over which samples above a specified threshold can inform block estimates. This allows high-grade composites to retain their local influence while limiting their impact on more distant blocks.

The estimate was completed using a multi-pass Ordinary Kriging strategy. The initial pass targeted well-informed areas using a restricted search and no HYL constraint. Subsequent passes expanded the search distances to estimate remaining blocks, and the HYL was applied to composites exceeding a threshold of 9% Cg. These high-grade samples were restricted to a reduced search ellipse of 60 x 60 x 30, thereby limiting their spatial influence during interpolation.

14.6 Variography

As Rossi and Deutsch (2014) summarize, “The single biggest problem in variogram interpretation is a lack of data to calculate a reliable variogram. Too few data for reliable variogram interpretation does not mean there is no variogram; it is just hidden or masked by data paucity. Geological analogues or expert judgment may be required.”

Experimental semi-variograms for the major and semi-major directions were created on the 8 m Cg % composites in the graphite domain. In the opinion of the QP, the 8 m composites did not provide an adequate measure of short-range variability in the minor direction and elected to use 1 m composites for the minor direction experimental variogram. A variogram model was fitted to the experimental variograms with the nugget contribution being applied to each model as observed in the downhole variogram. The model is summarized in Table 14-1 and a visualized in Figure 14-5 and Figure 14-6.

Table 14-1: Variogram Model Parameters of Cg in Graphite Domain

Variable	Structure	Contribution	Type	Direction			Range		
				Azi	Plunge	Dip	Major	Minor	Vertical
Cg %	C_0	0.20	Nugget	-	-	-	-	-	-
	C_1	0.40	Spherical	127	0	-45	115	135	8
	C_2	0.40	Spherical	127	0	-45	225	225	25

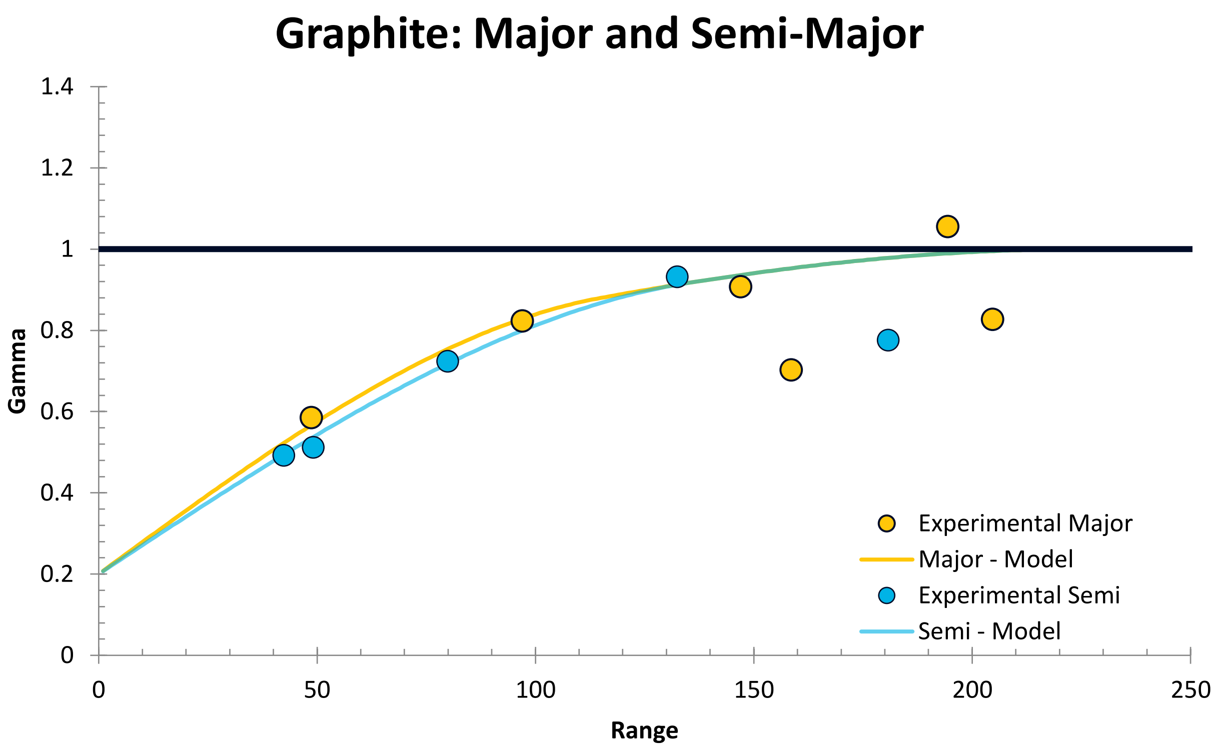


Figure 14-5: Variogram Model of Cg% in Major and Semi-Major Directions underlain with Experimental Variogram Model

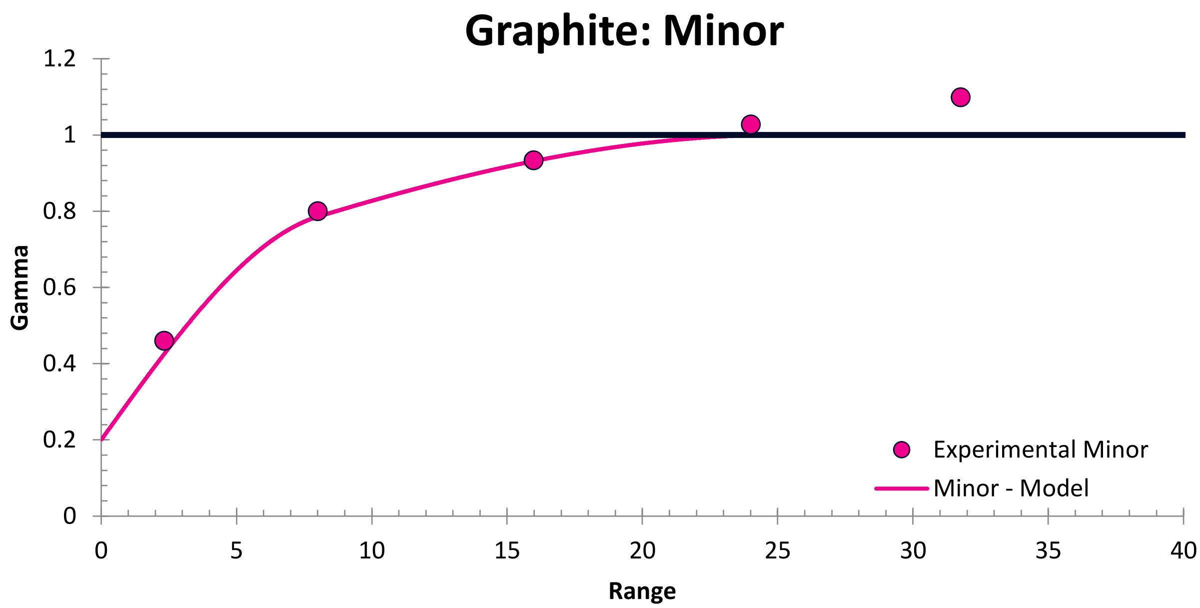


Figure 14-6: Variogram Model of Cg% in Minor Direction underlain with Experimental Variogram Model

14.7 Density

In the estimation domain, the graphite grade variable is nearly exhaustively sampled at all drill hole locations, but the density variable is missing at most locations. The collocated grade and density variables (n = 237) are weakly and negatively correlated. A linear regression was fitted to the collocated data and the regression was applied to the block estimates with Cg % grades. The equation is listed below, and the regression is visualized in Figure 14-7.

$$SG = 2.671 - 0.0118 \times \%\mathrm{Cg}$$

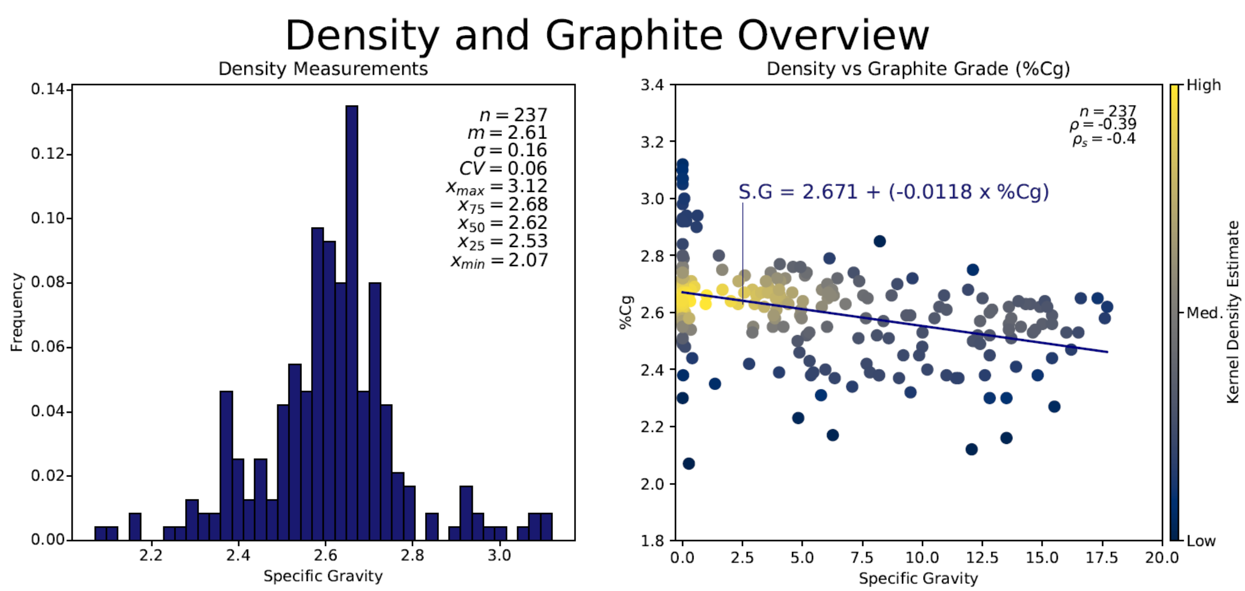


Figure 14-7: Specific Gravity Measurements Histogram and Bivariate Plot vs Cg wt% with Fitted Linear Regression

The lithology domains were assigned a density based on specific gravity measurements observed within the domains and, in the case of the overburden and granitic gneiss, an assumed density was used based on observations from analogous deposits (Table 14-2).

Table 14-2: Assigned Domain by Lithology Domain

Lithology Domain	Assigned Density
Overburden*	2.00
Biotite Gneiss	2.68
Granitic Gneiss*	2.60
Pegmatite 1	2.63
Pegmatite 2	2.63

*Assumed Density Based on Observations of Analogue Deposits

14.8 Block Model Estimation

14.8.1 Block Model Definition

A block model was constructed to encompass the graphite and lithology using a parent block of 20 m (strike direction) by 20 m (thickness direction) by 16 m (vertical direction) with 5 m by 5 m by 4 m sub-blocks. The parent block size was selected to approximate the size of an open pit mining unit, and the sub-blocks were used to adequately capture the geologic features of the modelled estimation and lithology domains. The origins, extents (offsets from origins), and rotation of the block model are summarized in Table 14-3.

Table 14-3: Block Model Definitions

Origins		
X	**Y**	**Z**
446220	6329570	150
Offset		
X	**Y**	**Z**
560	1900	448
Rotation*		
Bearing	**Plunge**	**Dip**
29	0	0

* Rotation expressed in Vulcan's rotation convention

14.8.2 Estimation Strategy and Testing

A Mineral Resource Estimate is to honour the data, replicate the relationships therein, be globally unbiased, and be geologically reasonable. An estimation strategy was constructed, and multiple models were tested to meet the stated criteria.

Minor localizations of high-grade and low-grade concentrations of graphite is observed within the domains. Ordinary Kriging (OK) assumes an unknown mean (and consequently a local varying mean during estimation) and was therefore selected as the kriging method to manage the local trends within the domains.

The estimate is to closely replicate the declustered mean of Cg % within the domains to ensure the estimate is globally unbiased. Target Cg % variance was derived from the composite data using a Discrete Gaussian Model (DGM). The DGM accounts for change of support using a variogram model, a normal score transformation, and Hermite polynomials (Harding & Deutsch, 2019). Change of support means that as the support of the core sample increases to the size of a mining unit (or block size), the observed variability will decrease and the distribution will become more symmetric (Harding & Deutsch, 2019; Figure 14-8). The QP completed a series of OK models to test different composite restrictions per estimate, targeting the declustered mean and variance from the DGM. Through this testing it was determined that 7 to 10 samples are to be used per estimate with 2 max samples per drillhole in well drilled area, and 5 to 7 samples per estimate with 2 max samples per drillhole in areas of sparser drilling.

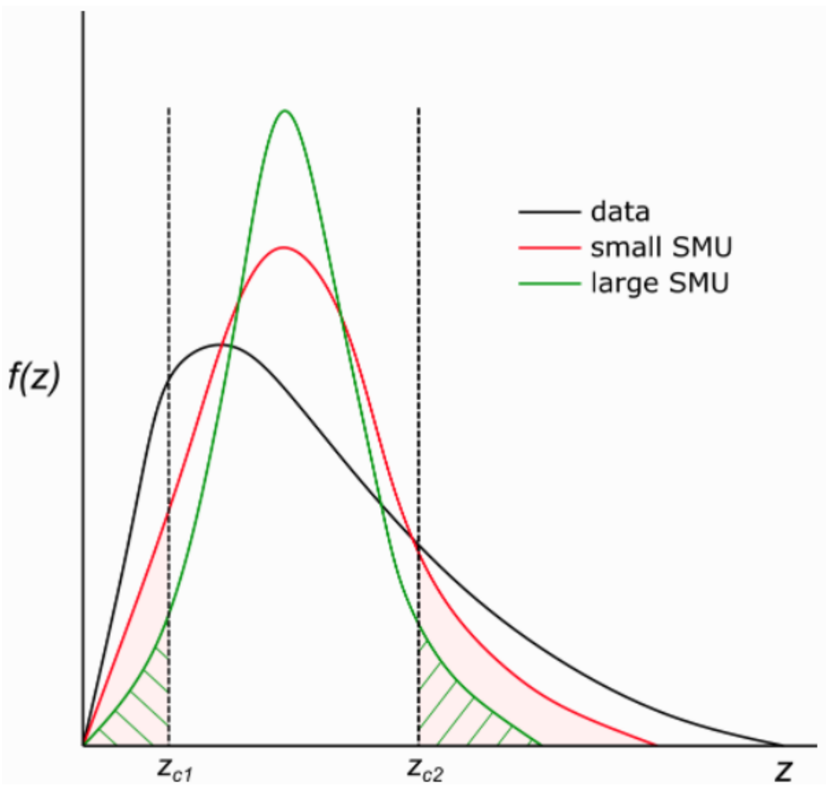


Figure 14-8: Diagram Demonstrating the Change of Support Principle (Harding & Deutsch, 2019)

14.8.3 Interpolation Methods

The blocks were independently estimated in Maptek Vulcan using Ordinary Kriging (OK) as informed by the directional variogram model. Block estimation was limited to blocks and composites within the wireframe domain. Block discretization was completed at a 4 by 4 by 4 configuration, with the estimated value of the parent blocks assigned to the sub-blocks.

The interpolation of Cg (%) was completed using a three-pass estimation strategy designed to prioritize well-informed blocks while progressively expanding into less densely drilled areas. The estimation parameters are summarized in Table 14-4. All estimation passes used a consistent search orientation of approximately 127° bearing, 0° plunge, and -

45° dip, consistent with the interpreted mineralization geometry. A maximum of two composites per drill hole was permitted for all passes to limit the influence of the string effect during kriging.

The first pass utilized a search ellipse of 100 m (major) by 100 m (semi-major) by 50 m (minor) and required a minimum of seven and maximum of ten composites per estimate. No High Yield Limit (HYL) restriction was applied in this pass. The intent of this pass was to estimate blocks in well-informed areas (e.g. blocks classified as Indicated) using a relatively tight search and higher sample support to reproduce the expected variance at the block scale.

The second pass expanded the search ellipse to 250 m by 250 m by 50 m, with a minimum of five and maximum of seven composites per estimate. The third pass further expanded the search to 400 m by 400 m by 100 m, with a minimum of three and maximum of five composites per estimate. These subsequent passes were applied only to blocks that remained unestimated after the previous pass.

A High Yield Limit (HYL) was applied during Pass 2 and Pass 3 to control the influence of high-grade composites. Composites exceeding a threshold of approximately 9% Cg were restricted to a reduced search ellipse of 60 m by 60 m by 30 m, thereby limiting their spatial influence relative to the full search range. This approach allows high-grade samples to inform nearby blocks while preventing excessive grade smearing into less-informed areas of the model.

Density values were assigned to the block model based on Cg grade or lithology, as described in Section 14.7.

The estimation parameters are summarized in Table 14-4.

Table 14-4: Estimation Parameter Summary

Domain	Bearing	Plunge	Dip	Pass 1				Pass 2				Pass 3				HYL Restriction		
				Major (m)	Semi (m)	Minor (m)	Samples Per Est.	Major (m)	Semi (m)	Minor (m)	Samples Per Est.	Major (m)	Semi (m)	Minor (m)	Samples Per Est.	Major (m)	Semi (m)	Minor (m)
1	127	0	-45	100	100	50	7-10	250	250	50	5-7	400	400	100	3-5	60	60	30
				Max 2 Samples/DDH Selects Indicated Blocks				**Max 2 Samples/DDH Targets Inferred Blocks**				**Max 2 Samples/DDH Inferred Blocks**				**Applied to Passes 2 & 3. Threshold of 9% Cg**		

14.9 Model Validation

The block model was validated via volume comparison, mean grade comparison, visual inspection, swath plots, and change of support comparison. Overall, there is a good correlation between the block estimates and the supporting composite grades.

14.9.1 Volume Comparison

The volume of the graphite domain was compared to contained block volume within the domain, as summarized in Table 14-5. Results show that there is good agreement between the wireframe and block model volume with a difference of -0.03%.

Table 14-5: Domain Wireframe Volume versus Contained Block Volume

Domain	Wireframe Volume (m^3)	Block Volume (m^3)	Percent Difference
Graphite 1	13,688,000	13,683,600	-0.03%

14.9.2 Global Bias Assessment

The average block grade was compared to the mean of the representative distribution for an assessment of global bias. Percent error was calculated for each domain, where:

$$\%\,Error = 100 * \frac{Block\ Average - Declustered\ Composite\ Average}{Declustered\ Composite\ Average}$$

Table 14-6 shows the percent error of the estimated blocks relative to the target mean for the domain. The comparison shows a slight underestimation (percent error of -3.6%) relative to the composites, which is attributed to the use of the HYL function to control the influence of high-grade samples in sparsely drilled areas. In the QP's opinion, this is a reasonable outcome to avoid grade smearing given the wide spacing of the informing drill holes. Further drilling may facilitate the removal of the HYL.

Table 14-6: Block Cg Grades vs. Target Cg Mean

Declustered Composite Avg. Cg %	Block Cg % Avg. Tonnes Weighted	Percent Error
6.98	6.73	-3.60%
*Limited to Indicated and Inferred blocks		

14.9.3 Visual Inspection

Block grades were visually compared with drill hole composites on cross-sections, longitudinal sections, and plan views. The block grades and composite grades correlate well visually within the deposit. Figure 14-9 is an example of the review, comparing the composites of the graphite domain to the estimated blocks in cross-section.

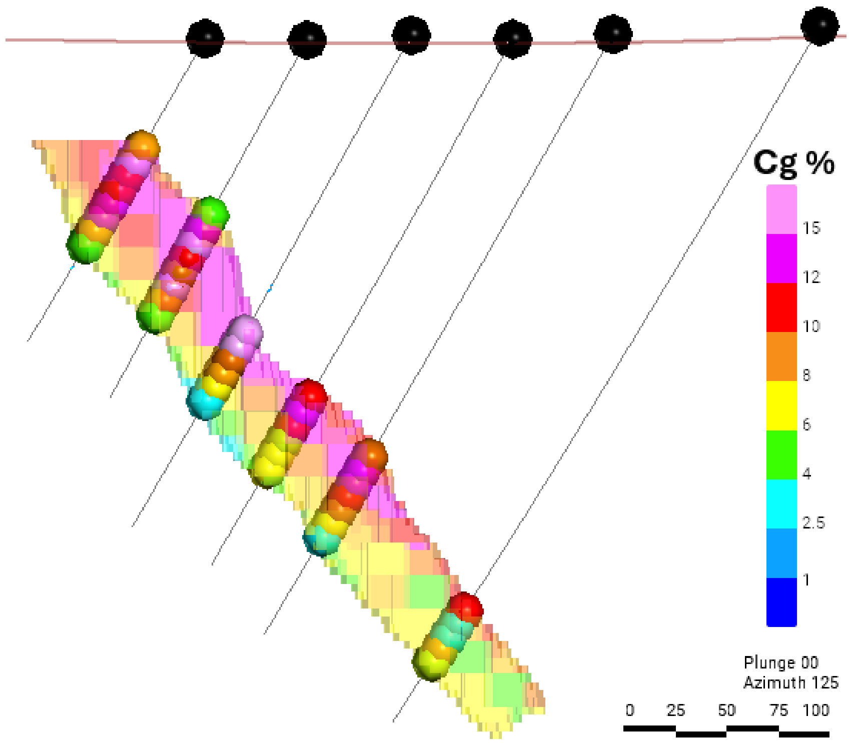


Figure 14-9: Cross-section (Looking at an Azimuth of 125°) of Estimated Cg % Blocks Underlain with the Cg Composites (Spheres) and Drillhole Traces

14.9.4 Swath Plots

Swath plots were generated in the three orthogonal directions relative to the estimation domain, denoted in the figures as X (strike), Y (dip direction) and Z (thickness) directions (Figure 14-10, Figure 14-11, and Figure 14-12). The swath plots compare the volume weighted block model grades derived from OK against the declustered composite grades and Nearest Neighbour (NN) block grades.

In all directions, the OK block model reproduces the overall grade trends observed in the composite data and is generally in good agreement with the NN estimate, indicating that the model honours the global distribution of grades. As expected, the NN estimate more closely follows the variability of the composites, while the OK estimate exhibits smoothing due to the kriging process.

The effects of the HYL are observed in the swath plots as localization of underprediction relative to the composites. These local deviations between the block model and composites are most pronounced in areas of lower sample density.

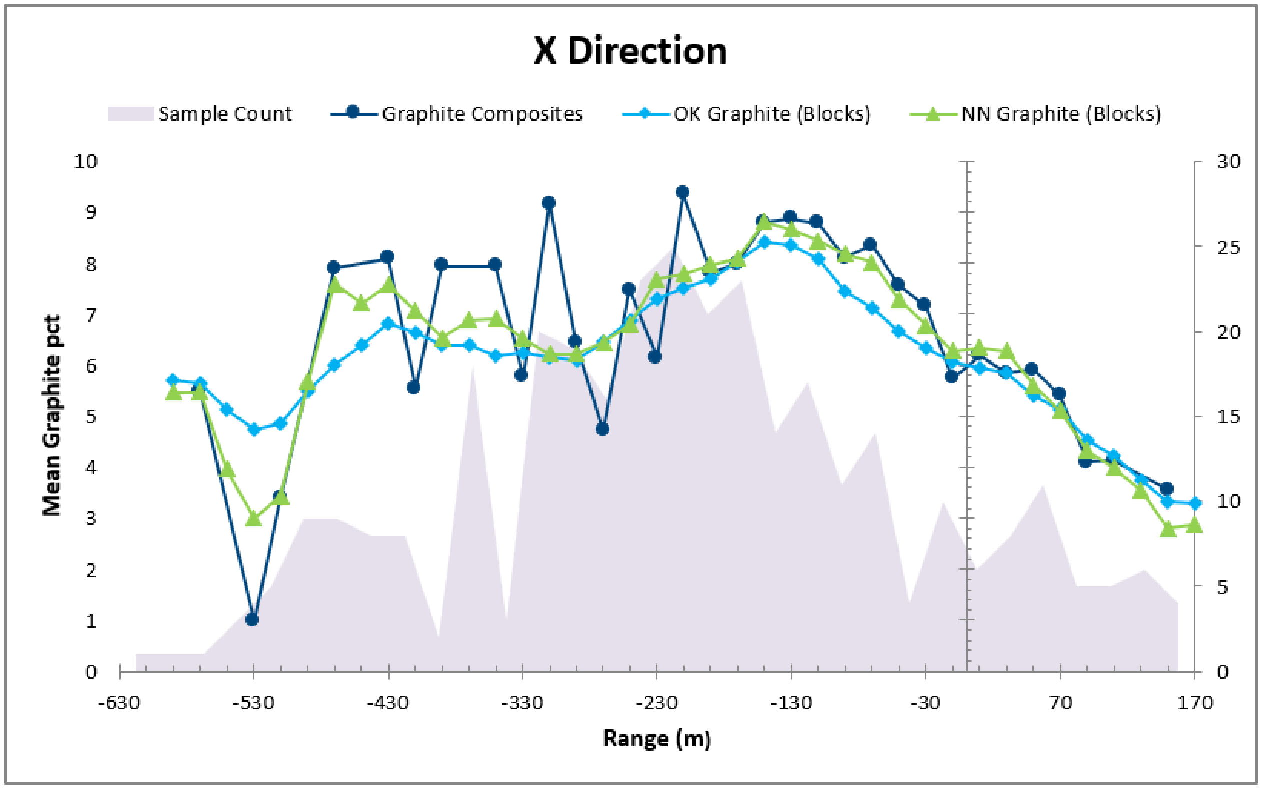


Figure 14-10: Swath Plot of the Graphite Domain along X Direction

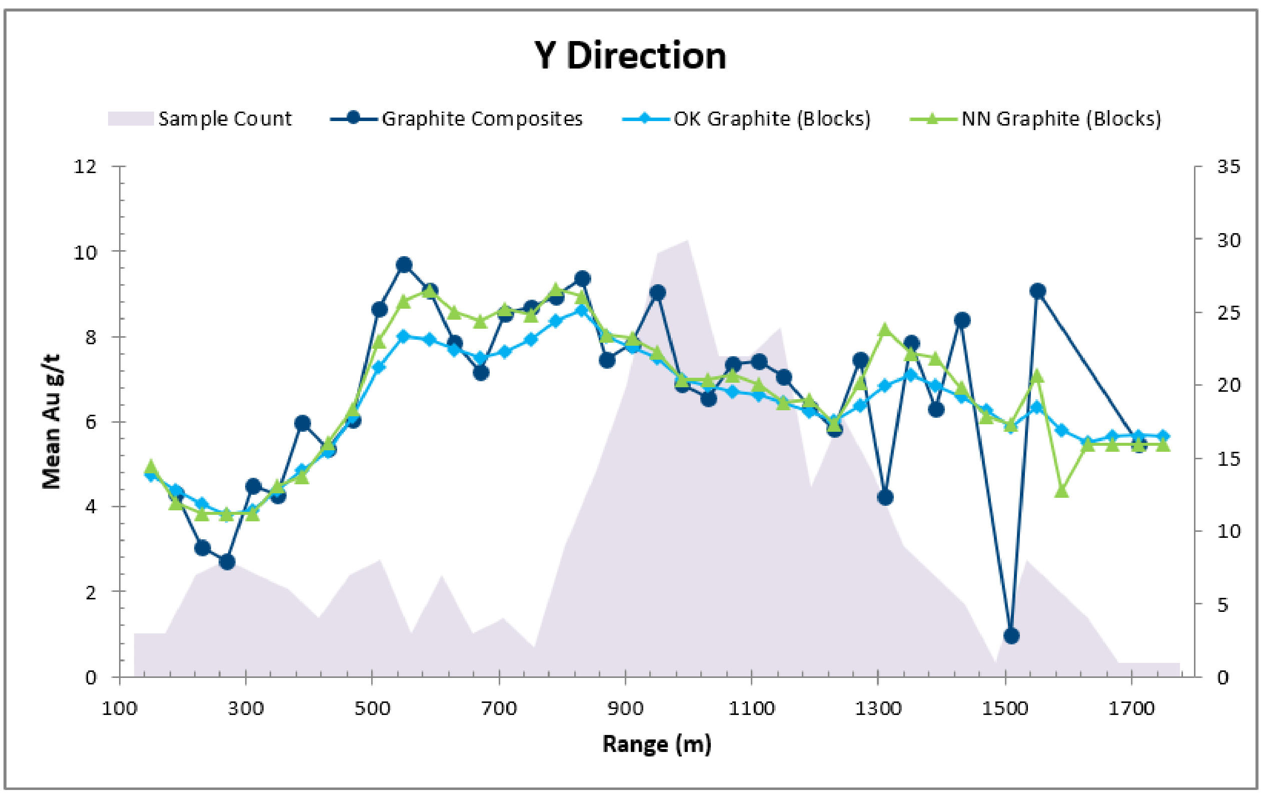


Figure 14-11: Swath Plot of the Graphite Domain along Y Direction.

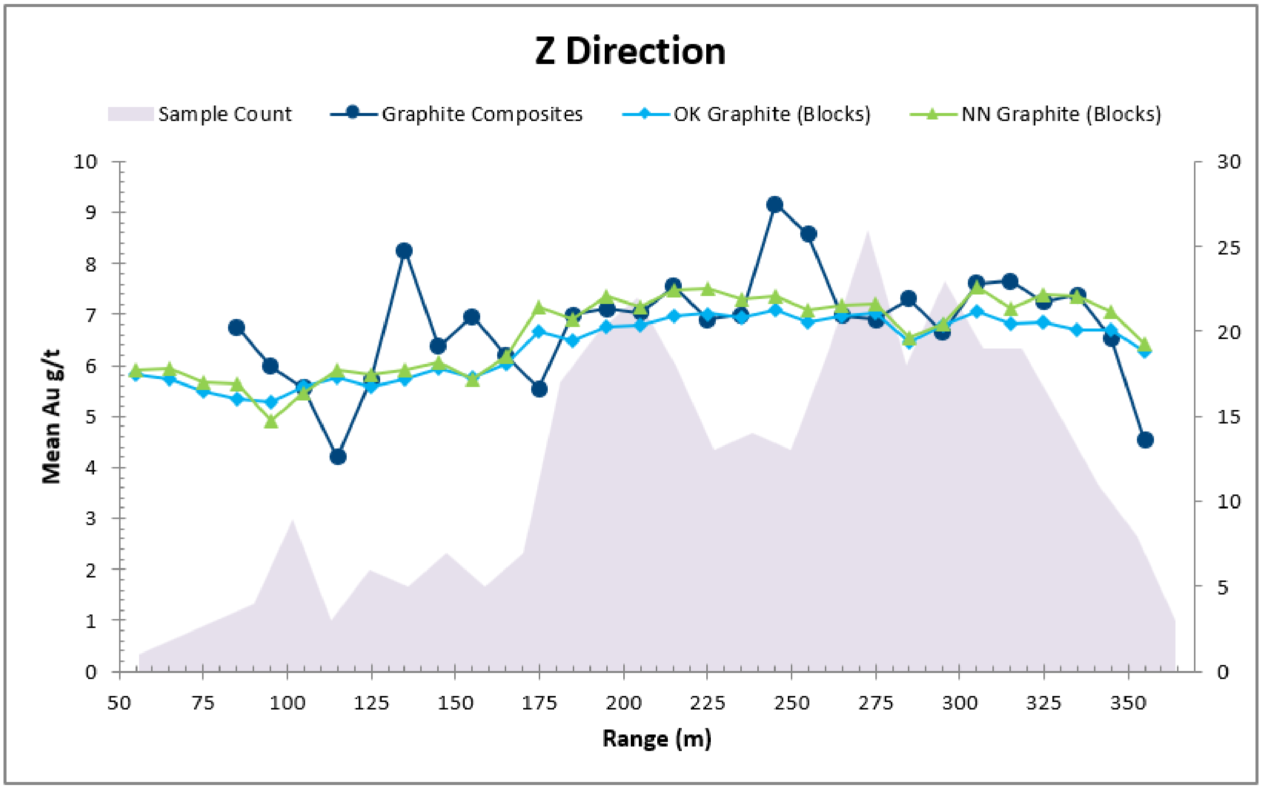


Figure 14-12: Swath Plot of the Graphite Domain along Z Direction

14.9.5 Change of Support Comparison

A Discrete Gaussian Model (DGM) was created using the 8 m graphite composites to account for change of support through application of a variogram model, normal score transformation, and Hermite polynomials.

A quantile–quantile (Q–Q) comparison of block estimates against the DGM distribution, based on reportable material (Indicated and Inferred), shows strong agreement across the full grade range (Figure 14-13); indicating that the block model appropriately reflects the change of support from composite to block scale.

The block model exhibits a minor global underestimation, with an average grade of 6.75% Cg compared to 6.98% Cg for the DGM. Estimates plot at or slightly below the 1:1 line across most of the low-grade end of the distribution. This behaviour is attributed to the application of the HYL, which restricts the influence of high-grade composites during estimation.

In the QP's opinion, the observed underestimation is minor and acceptable given the drill spacing and grade variability, and the block model is considered to appropriately reproduce the representative distribution after change of support.

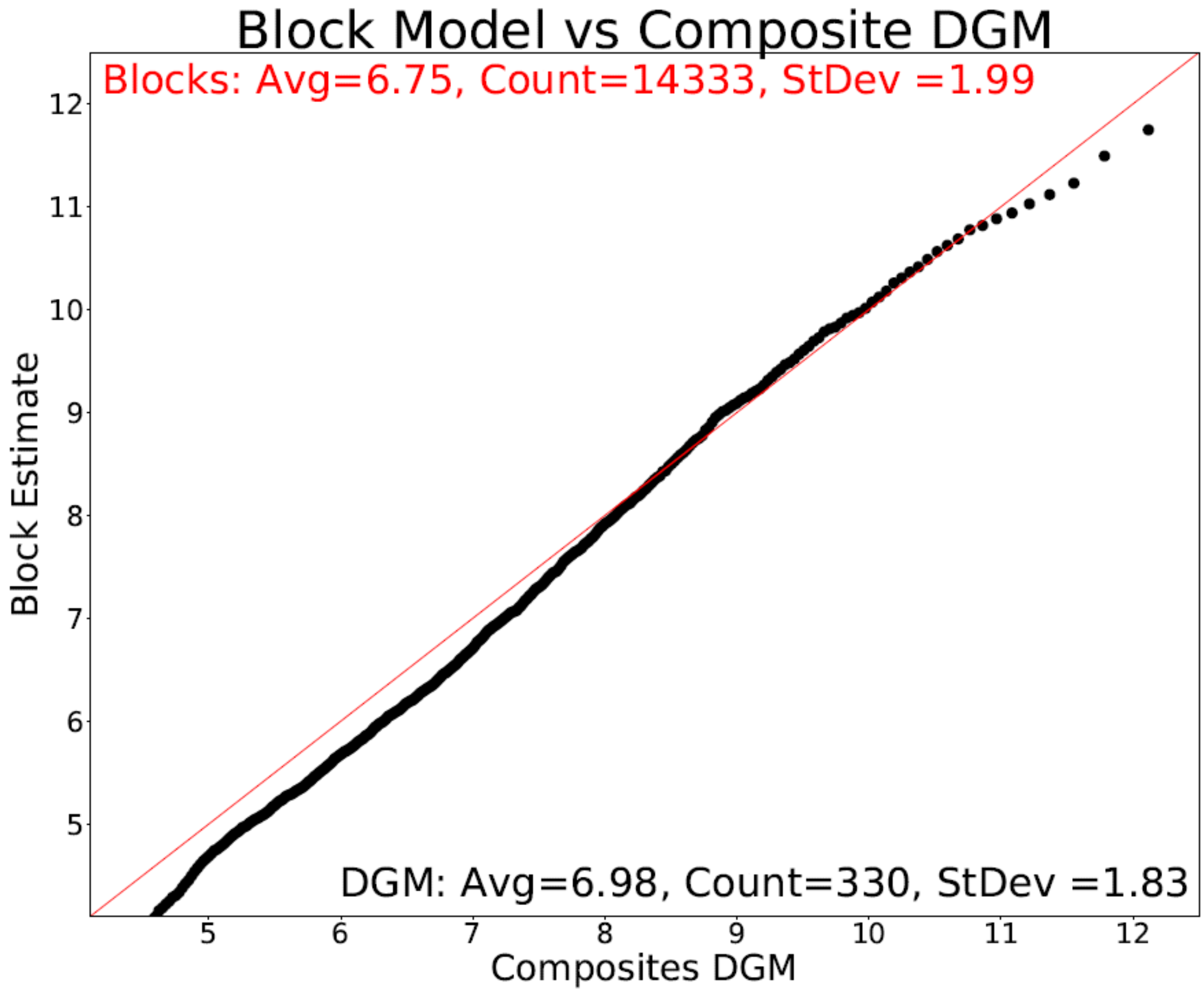


Figure 14-13: Quantile-Quantile Plot of Graphite DGM comparison to Estimated Blocks

14.10 Mineral Resource Classification

Mineral Resource classification for the Loki Deposit is based primarily on drill hole spacing and the Qualified Person's assessment of estimation quality, including geological continuity and confidence in grade interpolation.

Classification criteria were defined using a drill spacing framework, where Indicated Mineral Resources are defined by a grid spacing of approximately 50 m, where continuity is reasonably established, and Inferred Mineral Resources are defined by a grid spacing of approximately 100 m, where continuity is assumed but not verified.

No Mineral Resources have been classified as Measured due to the current drill spacing and associated uncertainty in geological and grade continuity.

An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated based on limited geological evidence and sampling. It has a lower level of confidence than an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that additional drilling may upgrade portions of the Inferred Mineral Resources to the Indicated category.

The classification at the Loki Deposit is visualized in Figure 14-14.

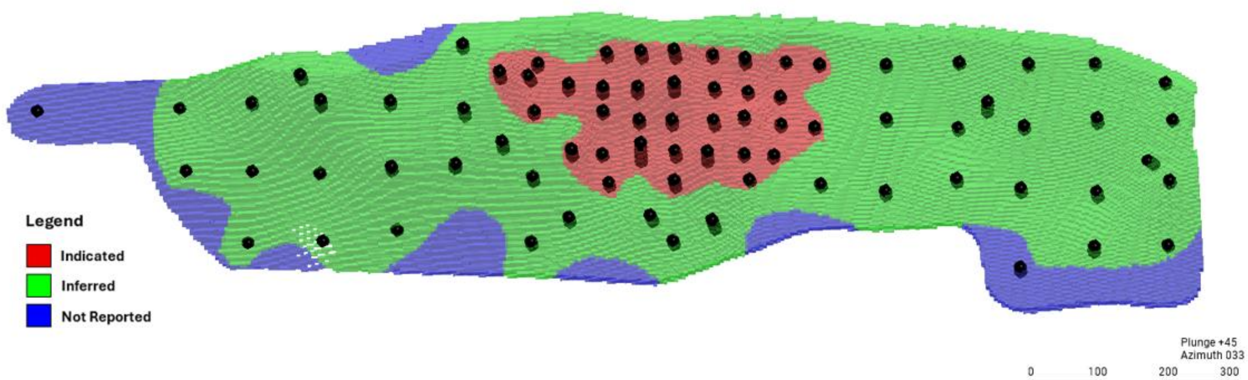


Figure 14-14: Long Section (Looking along an Azimuth of 033°) of Composites (Spheres) in Graphite Domain denoted with Classification

14.11 Cut-off Grade and Reasonable Prospects of Eventual Economic Extraction

The Deposit is located near the surface under a relatively thin overburden (~55 m thickness) and is currently defined to extend 335 m below surface; thus, it is envisioned to be mined via open pit mining methods if the project advances. A break-even cut-off grade was calculated to define the potentially economic portion of the deposit constrained within an open pit scenario. The work associated with the cut-off grade and generation of the conceptual open pit was completed by qualified engineers from Tetra Tech, and the resulting deliverables were reviewed and accepted by the QP of the Mineral Resource Estimate.

The break-even cut-off grade is estimated to be 2.30 % Cg based on the following parameters:

- A graphite price of $1,986(CAD)/t based on comparable projects and market research.
- Open pit production at a rate of approximately 2,400 tonnes per day.
- Process recovery of 90.0% and mining recovery of 95 % (5% dilution).

- $4.50(CAD)/t mining cost for overburden material.
- $6.00(CAD)/t mining cost for mineralized and waste material.
- $17.00(CAD)/t processing cost for processed material.
- G&A and surfaces services costs of $16.00 (CAD)/t and 9.00 (CAD)/t, respectively, for processed material.
- Transportation costs of $60.00(CAD)/t.

Tetra Tech performed pit optimization analysis to determine the Reasonable Prospects of Eventual Economic Extraction (RPEEE) for the project. The pit shell was generated with a 25° slope angle for Overburden, and 45° otherwise. The shell only considered material classified as Indicated or Inferred Mineral Resource (Figure 14-15 and Figure 14-16).

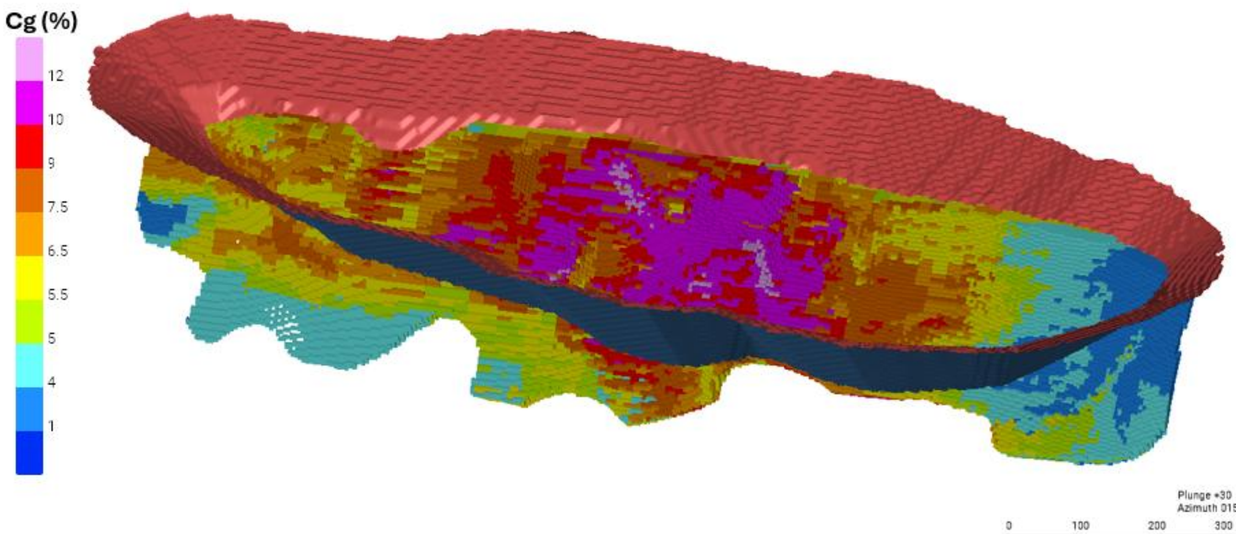


Figure 14-15: Oblique View (Looking along an Azimuth of 016° and Plunge of +29°) of Cg Block Grades Classified as Indicated and Inferred relative to Conceptual Open Pit Design

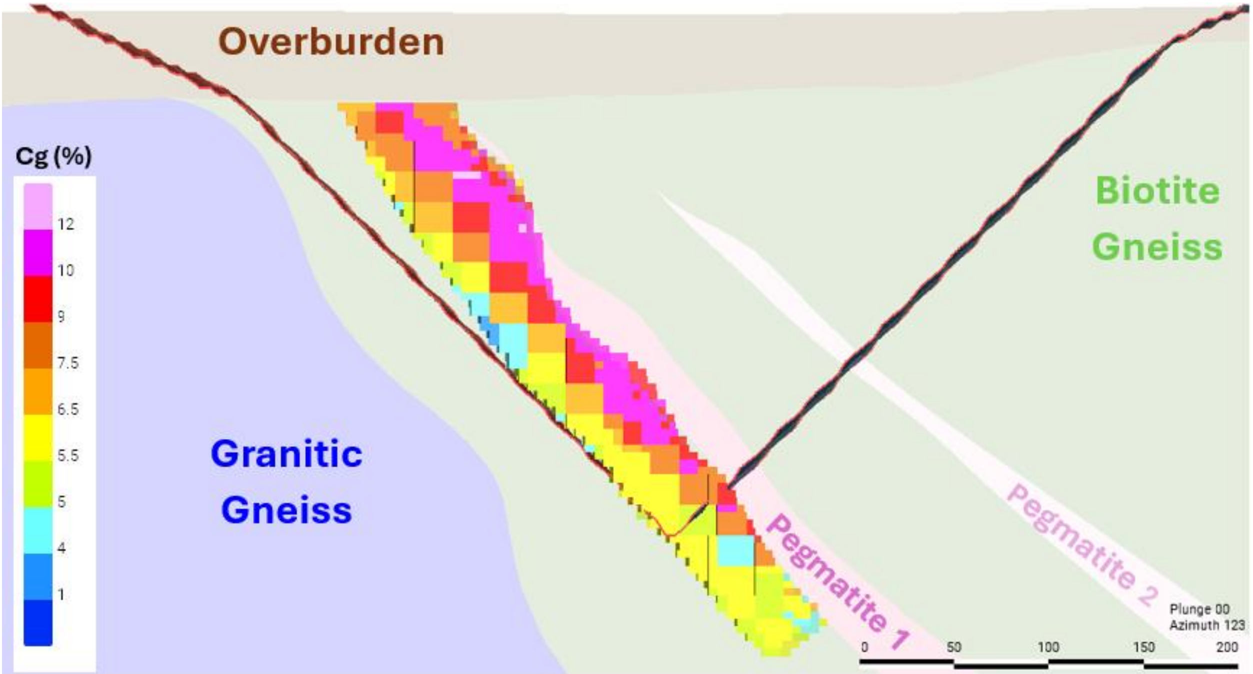


Figure 14-16: Cross-section (Looking at an Azimuth of 123°) of Cg Block Grades Classified as Indicated and Inferred relative to the Conceptual Open Pit Design underlain with the Lithology Domains

The parameters for the cut-off grade and open pit are considered to be approximate, as sufficient engineering and economic studies have not been conducted to generate authoritatively accurate values, and most or all of those parameters can be expected to change with time. This approach is not considered to represent an abnormal risk with respect to the validity of the Mineral Resource Estimate, as it meets the definition of “reasonable” in the context of reasonable prospects.

14.12 Mineral Resource Statement

The updated Mineral Resource Estimate for the Loki Deposit adheres to the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) and was reported at a cut-off grade of 2.30% Cg within a conceptual open pit design. The effective date of this Mineral Resource estimate is April 23, 2026 coincident with the date of the last assay result received from the analytical laboratory.

The Mineral Resource is composed of Indicated and Inferred Mineral Resources. The Indicated Mineral Resource totals 6.99 million tonnes at an average grade of 8.27 % Cg for a total of 0.58 million tonnes Cg, and the Inferred Mineral Resource totals 15.83 million tonnes at an average grade of 6.93 % Cg for a total of 1.10 million tonnes Cg (Table 14-7).

Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resource will be converted into a Mineral Reserve. No Mineral Reserves have been estimated at the Project.

Table 14-7: Loki Flake Graphite Mineral Resources (April 23, 2026)

Category	Cg Grade Cut-off (%)	Tonnage (Mt)	Cg Grade (%)	Contained Cg (Mt)
Indicated	2.30	6.99	8.27	0.58
Inferred	2.30	15.83	6.93	1.10

Notes:

1. The reporting standard for the Mineral Resource Estimate uses the terminology, definitions and guidelines given in the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Mineral Reserves (May 2014) as required by NI 43-101.
2. Reported Mineral Resources are constrained to a conceptual pit-shell above a cut-off grade of 2.30% Cg.
3. Numbers may not add up due to rounding.
4. The effective date of this Mineral Resource estimate is April 23, 2026.
5. The qualified person knows of no environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors that may materially affect the Mineral Resource Estimate in this report.
6. Mineral Resources are not Mineral Reserves and have not demonstrated economic viability.

14.13 Mineral Resource Uncertainty

Mineral deposits, including the Loki Deposit, are inherently uncertain because of variability at all scales and sparse sampling. In addition to uncertainty associated with estimation, there are specific risks and sources of uncertainty associated with the Loki Deposit. These risks should be evaluated by potential and current investors.

NI 43-101 and other similarly purposed International Codes (JORC, 2012; S-K 1300, 2019) are to disclose risks to the public as identified and evaluated by the QP. The QP addresses the technical risks in various sections, including this section, for the disclosure of such risks.

The risks listed below are not considered exhaustive and there may be additional risks and uncertainties not presently known, such as market or technology changes, that are currently deemed immaterial but may also affect the business.

14.13.1 Specific Identified Risks

The drill sampling methods used at the Loki Deposit meet or exceed industry standards and the assay results have been comprehensively reviewed and validated. The geostatistical estimates of in situ tonnages and grades are reasonable and validated thoroughly. The QP considers that these methods are appropriate to produce the declared Mineral Resources. However, the QP has identified a specific potential risk that require further investigation: uranium contamination of graphite mineralization.

Uranium Contamination Potential

All core was scanned with a scintillometer for elevated radioactivity that may indicate the presence of uranium. Core that measured elevated radioactivity or had elevated uranium from the traditional analysis underwent uranium assaying.

Of the 12,981 uranium assays, approximately 69% are at or below the 0.001% U_3O_8 detection limit, and approximately 27% are above the detection limit and less than or equal to 0.01% U_3O_8. The remaining approximately 4% of the population is represented by elevated uranium values above 0.01% U_3O_8. Beyond these described samples, the three highest uranium grade samples in the deposit area are:

- 0.36% U_3O_8 in KLS-24-026 from 199.7 m to 199.8 m (0.1 m)
- 1.19% U_3O_8 in KLS-24-033 from 59.1 m to 59.2 m (0.1 m)

- 1.10% U_3O_8 in KLS-24-033 from 59.2 m to 59.6 m (0.4 m)

The two samples grading above 1% U_3O_8 are located in the hanging wall, approximately 130 m perpendicular to the graphite domain and show no continuity (Figure 14-17). The 0.36% U_3O_8 sample is hosted in pegmatite of the hanging wall, approximately 35 m from the main graphite shear (Figure 14-18). Notably, uranium mineralization within the graphite estimation domain appears limited. Hole KLS-25-061 intersected a localized sample grading 445 ppm U within the current graphite zone model. Otherwise, there are no materially elevated uranium values identified within the interpreted graphite-bearing shear zone. In the QP's opinion, there is not enough meaningful uranium assay data within the graphite domain to create an estimation of uranium impurity for the model.

Although the graphitic shear does not appear to host significant uranium grade, additional drilling could prove otherwise. Furthermore, purity of the graphite is a typical customer specification. Some impurities can be removed during processing, but some cannot be removed without excessive grinding that leads to reduced graphite flake sizes. The QP recommends that future sample testing include impurity removal testing.

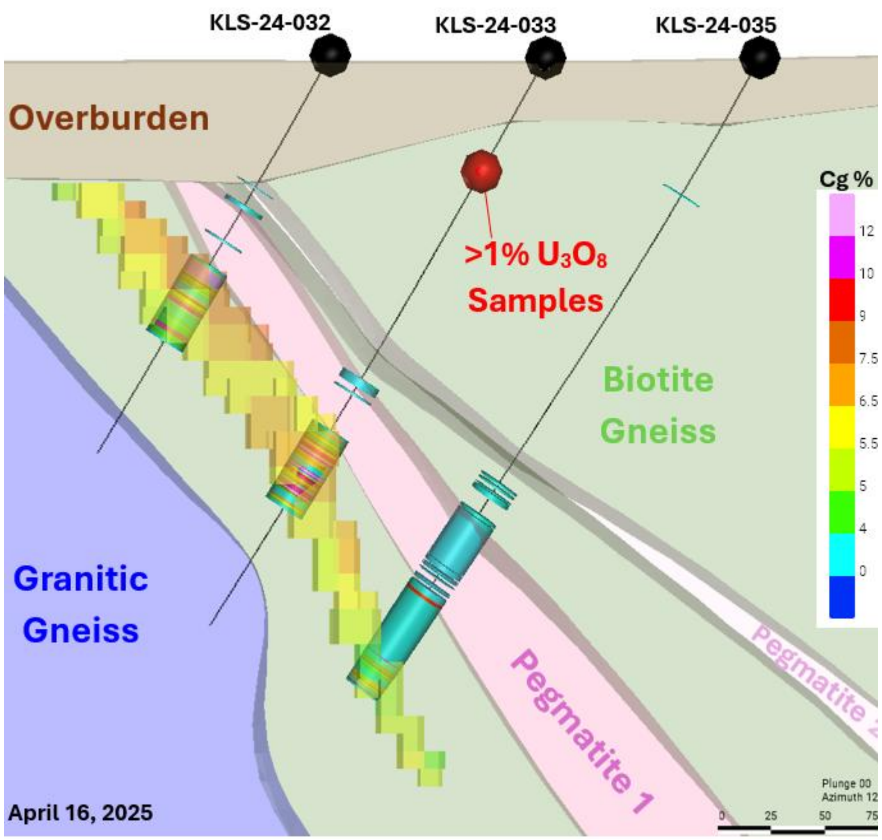


Figure 14-17: Cross-section (Looking at an Azimuth of 125°) of Cg Block Grades, Drill Hole Cg Assay Grades, and the two uranium samples above 1% U_3O_8 (spheres) underlain with the Lithology Domains.

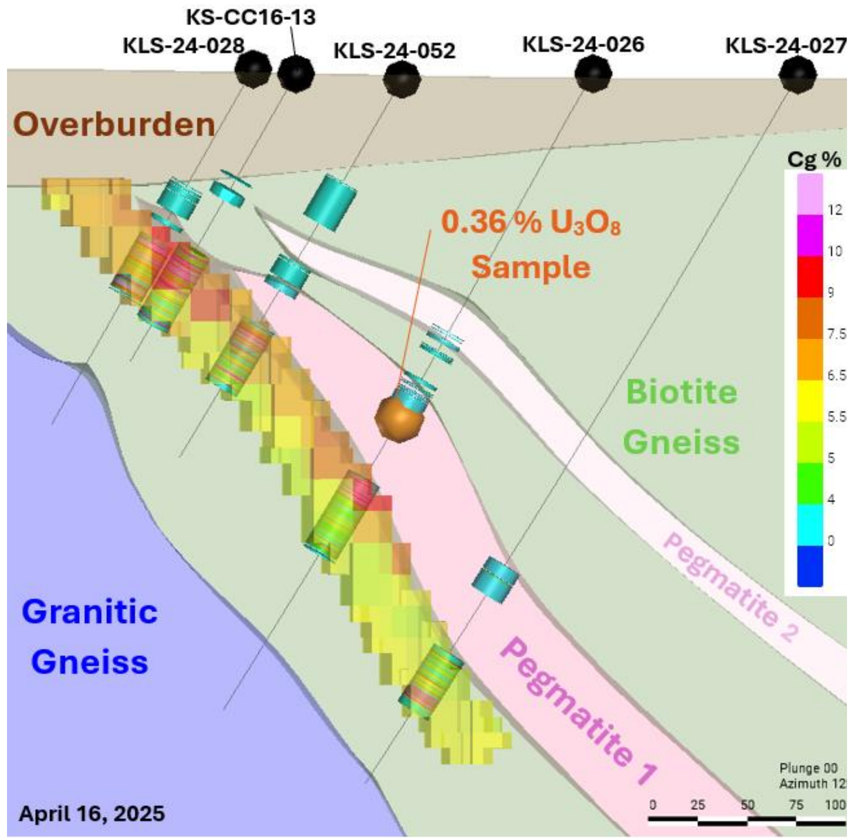


Figure 14-18: Cross-section (Looking at an Azimuth of 125°) of Cg Block Grades, Drill Hole Cg Assay Grades, and the uranium sample grading 0.36% U_3O_8 (spheres) underlain with the Lithology Domains.

14.13.2 Generic Mineral Resource Uncertainty

Mineral resources are uncertain because of variability at all scales and sparse sampling. The variables constituting the mineral resource, the volume of the geological interpretation and the grade estimates within that volume, are the sources of uncertainty. These uncertainties are typically a function of drill spacing, with denser spacing equating to less uncertainty and sparser spaced areas having more uncertainty.

Changes to the geologic interpretation would greatly alter the estimation. If new interpretations of geological complexities are presented, the Mineral Resource would need to be updated to reflect the new interpretations.

Abasca cannot be certain that any part or parts of a deposit or Mineral Resource estimate will ever be confirmed or converted into Mineral Reserves or that mineralization can in the future be economically or legally extracted.

14.14 Mineral Resource Sensitivity

The resources within the conceptual open pit were calculated at various cut-off grade thresholds as a review of the deposit's sensitivity to change in mining costs. The QP notes that there is no appreciable change in grade or tonnage between 0 and 5.0 % Cg, but appreciable changes are noted for cut-off grades above 5.0 % Cg; reflecting the sharp contact between the mineralization and waste rock. Table 14-8 summarizes the sensitivity of the tonnes and grade to the various cut-off grades and Figure 14-19 visualizes the table.

Table 14-8: Grade and Tonnage Sensitivity to Various Cut-off Grades

Classification	Cutoff (Cg %)	Tonnes (000s)	Grade (Cg %)	Contained Metal (Mt Cg)
Indicated	-	6,990	8.27	0.58
	2.3	**6,990**	**8.27**	**0.58**
	4	6,990	8.28	0.58
	5	6,870	8.34	0.57
	6	6,080	8.69	0.53
	7	4,800	9.29	0.45
	8	3,860	9.72	0.38
	9	2,860	10.15	0.29
	10	1,500	10.79	0.16
Indicated	-	15,850	6.93	1.1
	2.3	**15,830**	**6.93**	**1.1**
	4	15,080	7.1	1.07
	5	14,070	7.28	1.02
	6	11,130	7.74	0.86
	7	7,480	8.35	0.62
	8	4,200	9.04	0.38
	9	1,930	9.73	0.19
	10	500	10.51	0.05

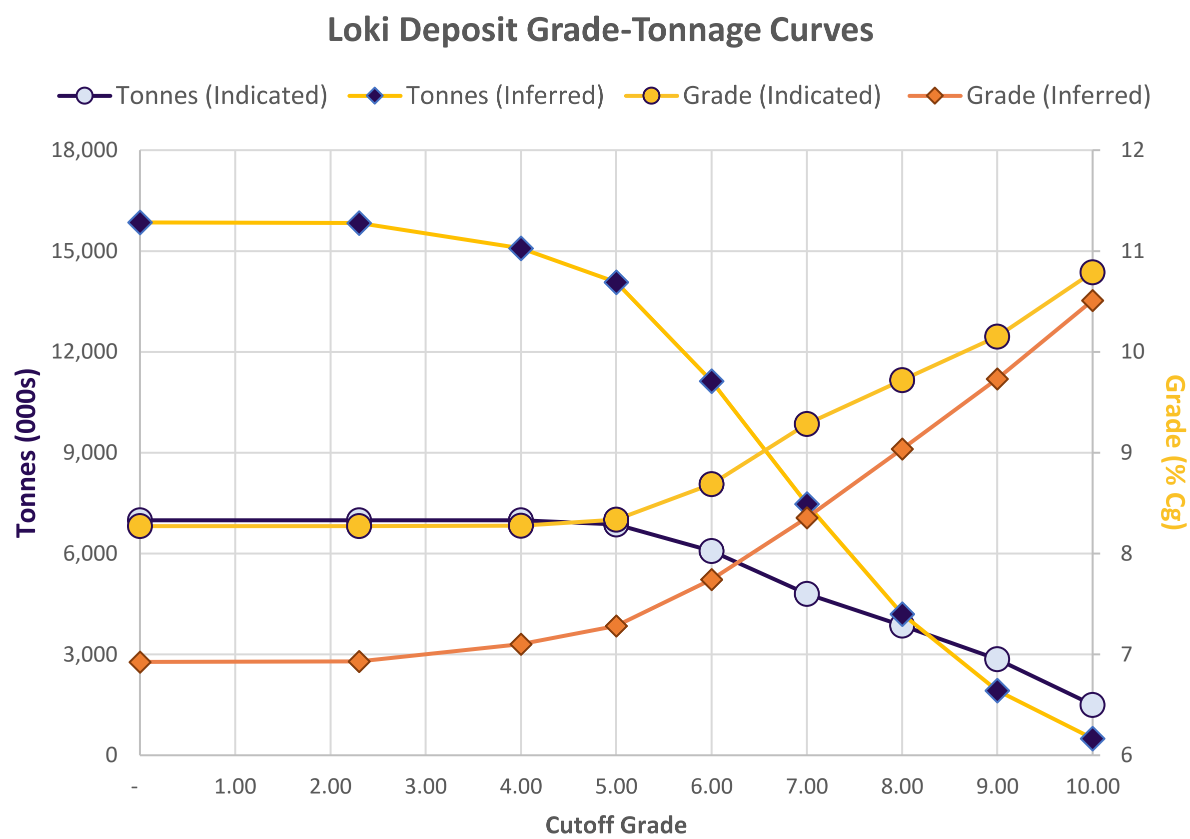


Figure 14-19: Graph of Grade and Tonnage Sensitivity to Various Cut-off Grades

14.15 Comparison with Previous Estimate

The 2026 Mineral Resource Estimate for the Loki Deposit represents an update to the initial 2025 Mineral Resource Estimate (effective date of April 10, 2025) completed by Understood Mineral Resources Ltd. The overall estimation methodology remains consistent with the previous estimate.

The updated estimate incorporates an addition 57 drill holes completed since the 2025 Mineral Resource Estimate, which was based on 22 drill holes. The overall geometry and orientation of the mineralized zone remain consistent with the previous interpretation, although additional drilling significantly expanded the interpreted extent of graphite mineralization (Figure 14-20). The interpreted graphite domain volume increased from approximately 6.9 million cubic metres in the 2025 estimate to approximately 13.7 million cubic metres in the updated 2026 interpretation. Refinement of the mineralized wireframe was also completed in areas where infill drilling was available.

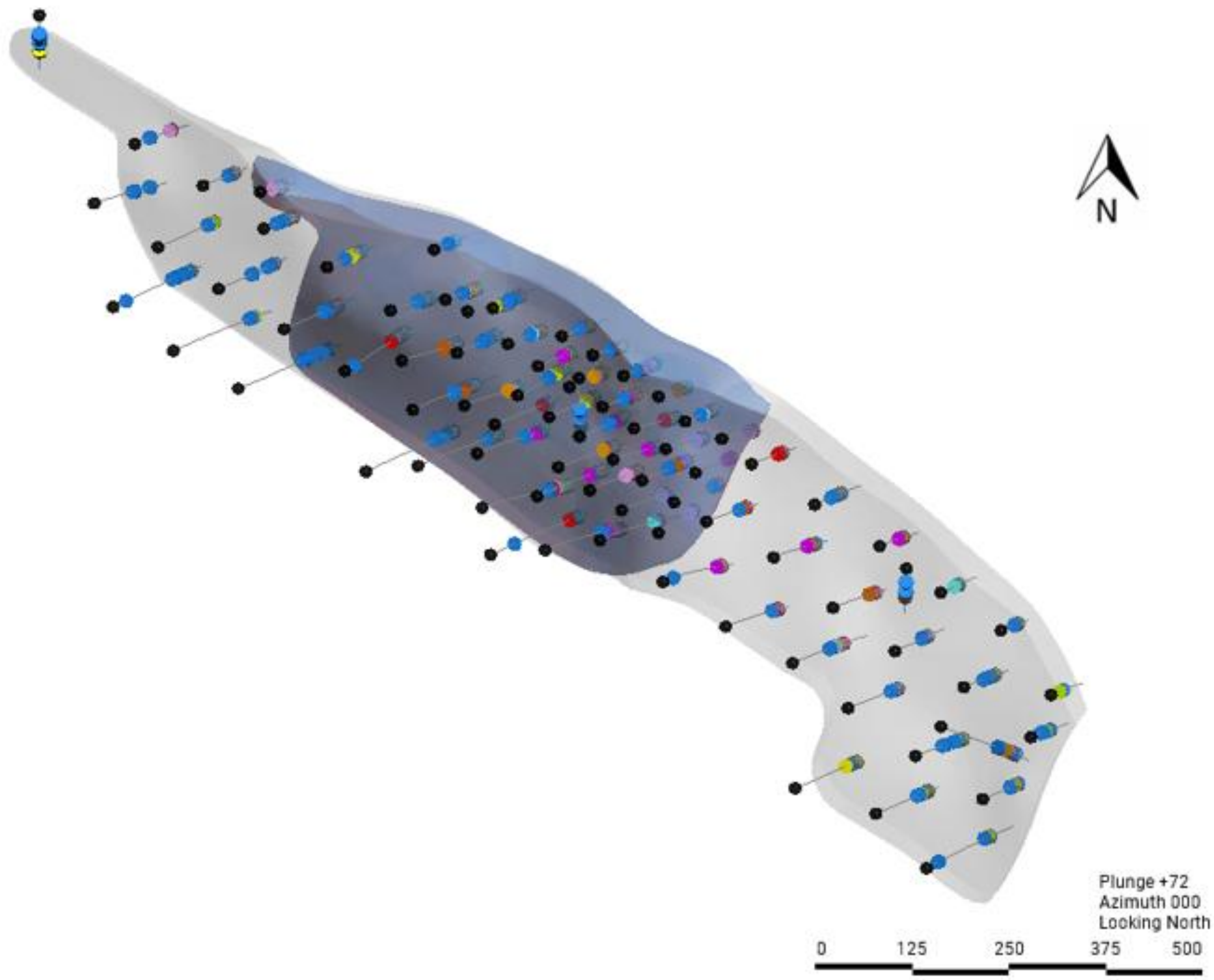


Figure 14-20: Oblique View Looking North Comparing the 2025 Graphite Domain Wireframe (Blue) to the Updated 2026 Graphite Domain Wireframe (Grey) for the Loki Deposit.

The updated interpretation resulted in localized thinning in the addition of the western end of the graphite wireframe. To better honour the updated geometry the block model sub-block dimensions were reduced relative to the 2025 estimate from 10 x 10 x 8 m to 5 x 5 x 4 m. The overall estimation approach remained consistent between the two models, and both were refined and calibrated to a DGM model.

The density assignment methodology remained consistent with the 2025 estimate; however, additional specific gravity measurements collected since the previous estimate allowed for revisions to several lithology density assignments. Relative to the 2025 estimate, the assigned density for the biotite gneiss domain increased from 2.63 t/m³ to 2.68 t/m³, while the granitic gneiss domain decreased from 2.70 t/m³ to 2.60 t/m³. Minor revisions were also made to the pegmatite domains.

The cut-off grade for the current model was estimated at 2.30% Cg, representing a small decrease from the 2025 cut-off grade of 2.78%. This change is attributed to updated prices and costs used in the calculation. The method for generating the conceptual pit shell is similar to that used in 2025 but reflects the updated prices and costs.

The 2025 Mineral Resource Estimate for the Loki Deposit was classified entirely as Inferred Mineral Resources, totalling 11.31 million tonnes at an average grade of 7.65 % Cg for a total of 0.86 million tonnes Cg. Additional infill drilling in 2025 and 2026 allowed some material to be classified as Indicated, while step-out drilling supported the expansion of Inferred Mineral Resources.

14.16 Relevant Factors

The QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource Estimate that is not discussed in this Technical Report.

A variety of factors may affect the 2026 Loki Mineral Resource Estimate, including but not limited to: changes to product pricing assumptions, re-interpretation of geology, geometry and continuity of mineralization zones, mining and metallurgical recovery assumptions, and additional infill or step out drilling.

In UMR's opinion, the estimation methods used are consistent with standard industry practice and the Inferred Mineral Resource Estimates for Loki Deposit are considered to be reasonable and acceptable.

15.0 MINERAL RESERVE ESTIMATE

There is no current Mineral Reserve estimate for the Loki Flake Graphite Deposit as it is currently at a PEA stage. CIM defines a Mineral Reserve as:

"A Mineral Reserve is the economically mineable part of a measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at pre-feasibility or feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.

The public disclosure of a Mineral Reserve must be demonstrated by a Pre-Feasibility Study or Feasibility Study."

16.0 MINING METHODS

The Project will be developed using conventional open-pit mining methods, including drilling, blasting, loading, and hauling with conventional mining equipment. Material will be drilled and blasted, then loaded into diesel-powered haul trucks using a fleet of hydraulic shovels and front-end wheel loaders. The mine plan is based on a nominal average processing capacity of approximately 2,750 tpd.

16.1 Pit Optimization

Tetra Tech performed the open pit optimizations using Geovia Whittle™ software. Pit optimization parameters were prepared from first principles estimate, technical studies and experience from other projects.

16.1.1 Block Model

Mineral Resources for the Project are documented in Section 14.0 and form the basis of the open pit optimizations. A regularized block model was provided to the mining team as a CSV file with 5 m by 5 m by 4 m blocks.

16.1.2 Geotechnical Information

Due to the limited geotechnical information at this early stage, overall pit slope assumptions of 25° and 45° for overburden and rocks, respectively, were applied in the optimizations.

16.1.3 Hydrogeological Information

No hydrogeological information was considered at this stage of the study.

16.1.4 Dilution and Mine Recovery

Dilution and mining recovery have been assumed based on similar open pit mining operations and are shown in Table 16-1.

Table 16-1: Mining Recovery and Dilution

Parameter	Unit	Value
Mining Recovery	%	95
Dilution	%	5

16.1.5 Pit Optimization Parameters

Pit optimization parameters are outlined in Table 16-2. Costs are in US dollars, unless otherwise stated. All other units, including tonnages, are in metric, unless otherwise stated. Graphite concentrate (95% Cg) was considered the final product with a price of US$1,450/t based on Section 19.0 of this report. A three-year average CAD1.00:USD0.73 exchange rate was applied. It is assumed that the concentrate will be delivered to a potential buyer in Saskatoon by trucking.

Table 16-2: Pit Optimization Parameters

Items		Units	Value
Exchange Rate		1 $CDN= X US$	**0.73**
Discount Rate		%	**8.00%**
Production Rate	Daily Processing Capacity	t/day	**2,750**
Product Price (Market)	Graphite Concentrate	US$/t	**1,450**
Processing	Method		**Flotation**
	Concentrate Grade	% Cg	**95.00%**
	Recovery: Overall Processing	**%**	**90.00%**
Off-site costs	Transportation - Graphite Concentrate (Trucking cost from site to Saskatoon)	$US/t graphite concentrate	**44**
	Insurance	%	**0.15%**
	Royalty, Third Party	%	**0.00%**
Operating Cost	**Mining:**		
	Mineralized Materials	$US/t mined	**4.38**
	Overburden	$US/t mined	**3.29**
	Waste	$US/t mined	**4.38**
	Processing:		
	Processing	$US/t processed	**12.41**
	G&A	$US/t processed	**11.68**
	Surface Services	$US/t processed	**6.57**
Block Model	Block Model	m	**5 x 5 x 4**
	Graphite Grade	%	**graphite**
Density	Provided in the Block Model	tonne/m^3	
Mining Technical	Mining Recovery	%	**95%**
Assumptions	Mining Dilution	%	**5%**
Pit Slope Angles	Overall, Pit (Overburden)	degrees	**25°**
	Overall, Pit (Rock)	degrees	**45°**

16.1.6 Pit Optimization Results

Using the pit optimization parameters in Table 16-2, pit shells were generated using Geovia Whittle™ software, corresponding to price factors ranging between 0.52 and 1 at 0.02 intervals. The pit optimization results are summarized in Table 16-3 and Figure 16-1. Pit shell corresponding to revenue factor of 0.72 was selected as the ultimate pit for further pit designs.

Figure 16-1: Pit Optimization Results by Revenue Factor

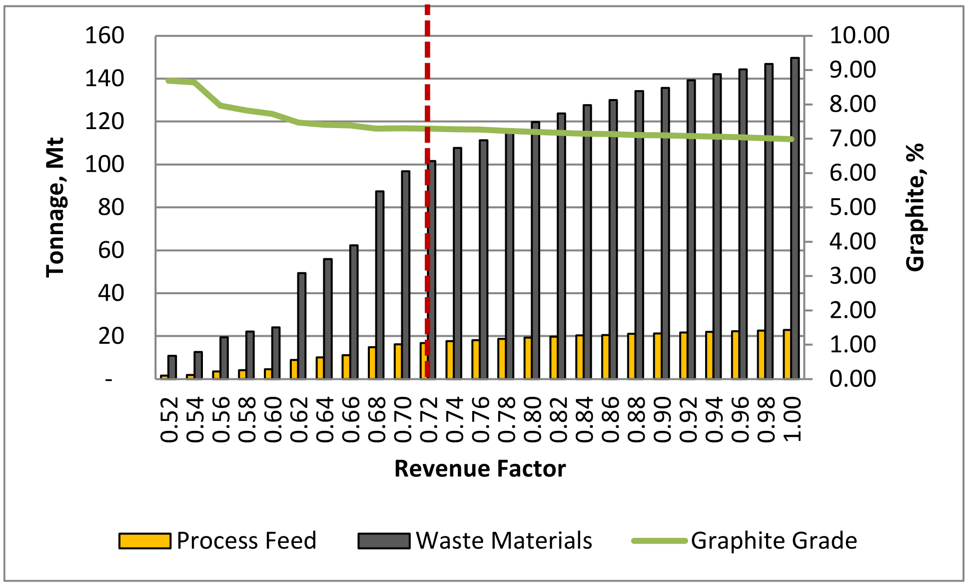


Table 16-3: Pit Optimization Results

Pit	Revenue Factor	Total Mined	Processed	Waste	Graphite Feed Grade	Graphite Concentrate	Strip Ratio	Net Operating CF	Discounted Net Operating CF	Production Life
#		kt	kt	kt	%	kt		US$ M	US$ M	Years
1	0.52	12,419	1,636	10,782	8.69	134.7	6.6	94.5	80.5	2.7
2	0.54	14,483	1,930	12,553	8.64	158.0	6.5	110.5	91.7	3.2
3	0.56	22,950	3,528	19,422	7.96	266.2	5.5	181.5	137.1	5.3
4	0.58	26,285	4,135	22,151	7.82	306.5	5.4	206.6	150.4	6.1
5	0.60	28,721	4,585	24,136	7.72	335.4	5.3	223.8	158.9	6.7
6	0.62	58,255	8,876	49,379	7.47	628.4	5.6	389.6	215.4	12.6
7	0.64	65,891	10,052	55,840	7.41	705.6	5.6	431.0	222.7	14.1
8	0.66	73,433	11,054	62,379	7.39	773.9	5.6	465.6	226.1	15.5
9	0.68	102,438	14,914	87,524	7.29	1,030.5	5.9	588.2	234.4	20.6
10	0.70	112,942	16,120	96,822	7.31	1,115.6	6.0	626.6	236.7	22.3
11	0.72	118,336	16,777	101,559	7.29	1,158.9	6.1	644.8	237.0	23.2
12	0.74	125,334	17,597	107,737	7.27	1,212.7	6.1	665.9	236.4	24.3
13	0.76	129,314	18,051	111,263	7.26	1,241.9	6.2	676.3	235.6	24.8
14	0.78	133,715	18,643	115,072	7.23	1,276.1	6.2	687.7	234.5	25.5
15	0.80	139,084	19,271	119,812	7.20	1,314.0	6.2	699.2	234.0	26.3
16	0.82	143,555	19,791	123,764	7.17	1,344.9	6.3	707.8	233.4	26.9
17	0.84	147,960	20,288	127,671	7.15	1,373.9	6.3	715.0	232.5	27.5
18	0.86	150,600	20,566	130,034	7.14	1,390.6	6.3	718.6	231.9	27.8
19	0.88	155,203	21,078	134,125	7.11	1,419.4	6.4	724.1	230.5	28.4
20	0.90	156,877	21,253	135,624	7.10	1,429.5	6.4	725.7	230.0	28.6
21	0.92	160,890	21,671	139,218	7.08	1,453.0	6.4	728.7	228.6	29.1
22	0.94	164,007	21,988	142,018	7.06	1,470.5	6.5	730.5	227.5	29.4

Pit	Revenue Factor	Total Mined	Processed	Waste	Graphite Feed Grade	Graphite Concentrate	Strip Ratio	Net Operating CF	Discounted Net Operating CF	Production Life
#		kt	kt	kt	%	kt		US$ M	US$ M	Years
23	0.96	166,586	22,271	144,315	7.04	1,485.2	6.5	731.5	226.4	29.7
24	0.98	169,358	22,586	146,772	7.01	1,500.8	6.5	732.1	225.1	30.0
25	1.00	172,547	22,923	149,625	6.99	1,517.7	6.5	732.4	223.7	30.4

16.2 Pit Design

Limited geotechnical data is available for the Loki Flake. Current open pit design is based on the similar operations. The pit design incorporates a bench height of 16 m; to be mined out in two filches of 8 m each. Pit design parameters are shown in Table 16-4.

Table 16-4: Pit Design Parameters

Item	Value
Overburden	
Bench height, m	16
Bench face angle, degrees	40
Inter-ramp angle, degrees	30
Catch bench width, m	8.6
Rock	
Bench height, m	16
Bench face angle, degrees	80
Inter-ramp angle, degrees	55
Catch bench width, m	8.4

16.2.1 Final Pit and Intermediate Mining Phases

Total tonnage contained in the final pit design is 126.3 Mt; includes 17.9 Mt of mineralized materials grading 7.6% graphite and 108.4 Mt of waste materials. Overall strip ratio is 6.1. It is worth noting that the 7.6% graphite grade reported here is the in-situ grade. Due to the projected mining dilution, the mill feed graphite grade will be 7.3%. The final pit design as well as the Five intermediate mining phases are shown in Figure 16-2 to Figure 16-7. Materials summary of the final pit design as well as the intermediate mining phases are shown in Table 16-5.

Table 16-5: Material Summary of the Final Pit and the Intermediate Mining Phases

Mining Phase	Plant Feed	Average Grade	Waste	Total Mined	Strip Ratio
	kt	**% Graphite**	**kt**	**kt**	
Phase 1 (Starter Pit)	853	8.6	11,177	12,030	13.1
Phase 2	1,872	7.8	12,701	14,572	6.8
Phase 3	1,954	7.4	16,472	18,425	8.4
Phase 4	2,531	7.3	15,861	18,393	6.3
Phase 5	3,607	7.3	22,636	26,244	6.3
Final Phase	7,099	7.8	29,552	36,651	4.2
Total	**17,916**	**7.63**	**108,398**	**126,315**	**6.1**

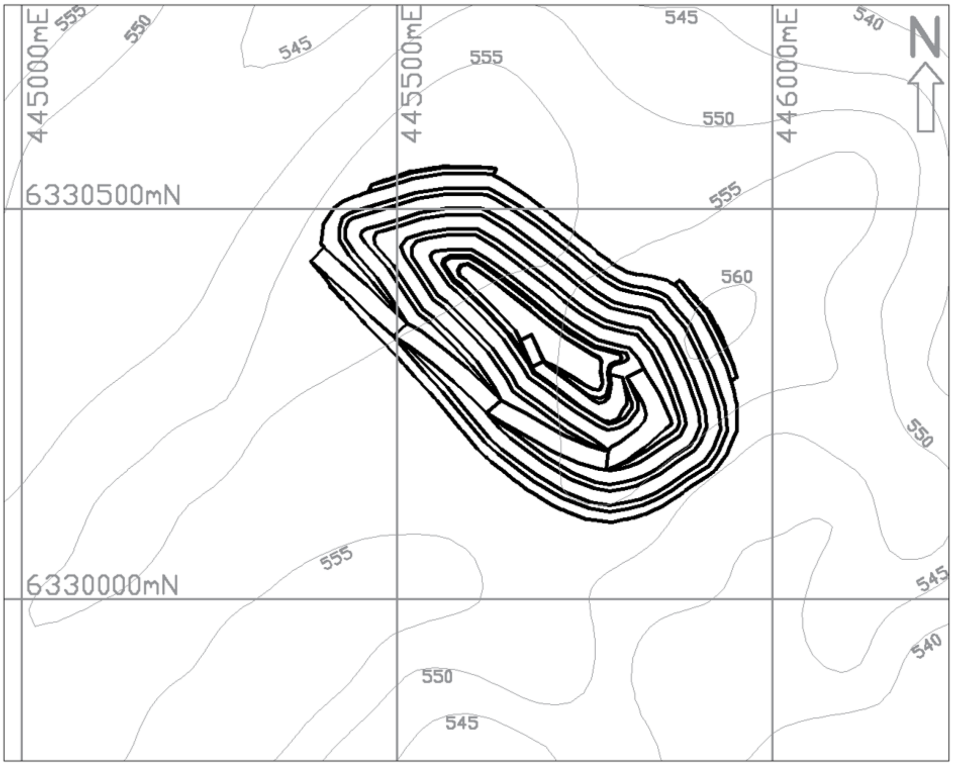


Figure 16-2: Mining Phase 1 (Starter Pit)

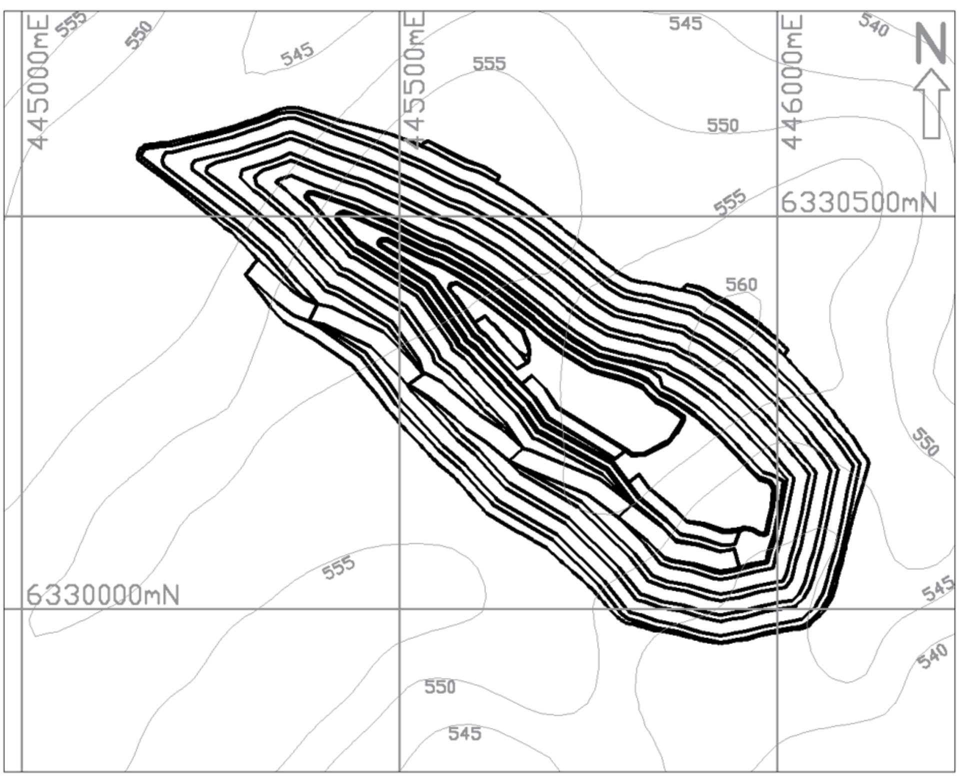


Figure 16-3: Mining Phase 2

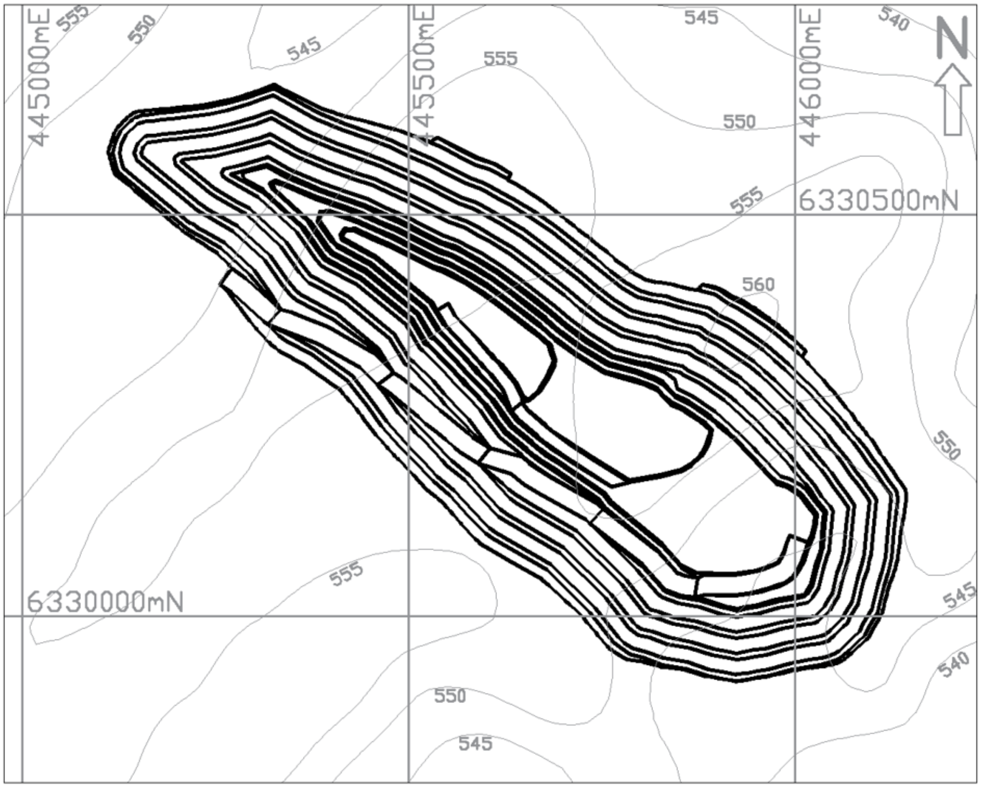


Figure 16-4: Mining Phase 3

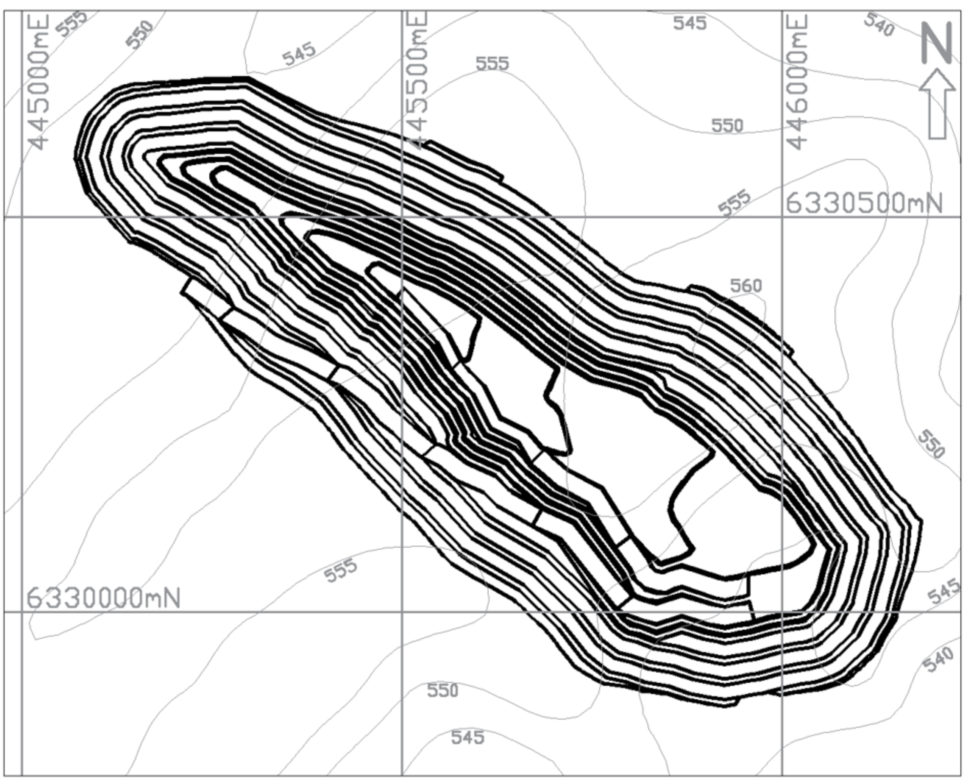


Figure 16-5: Mining Phase 4

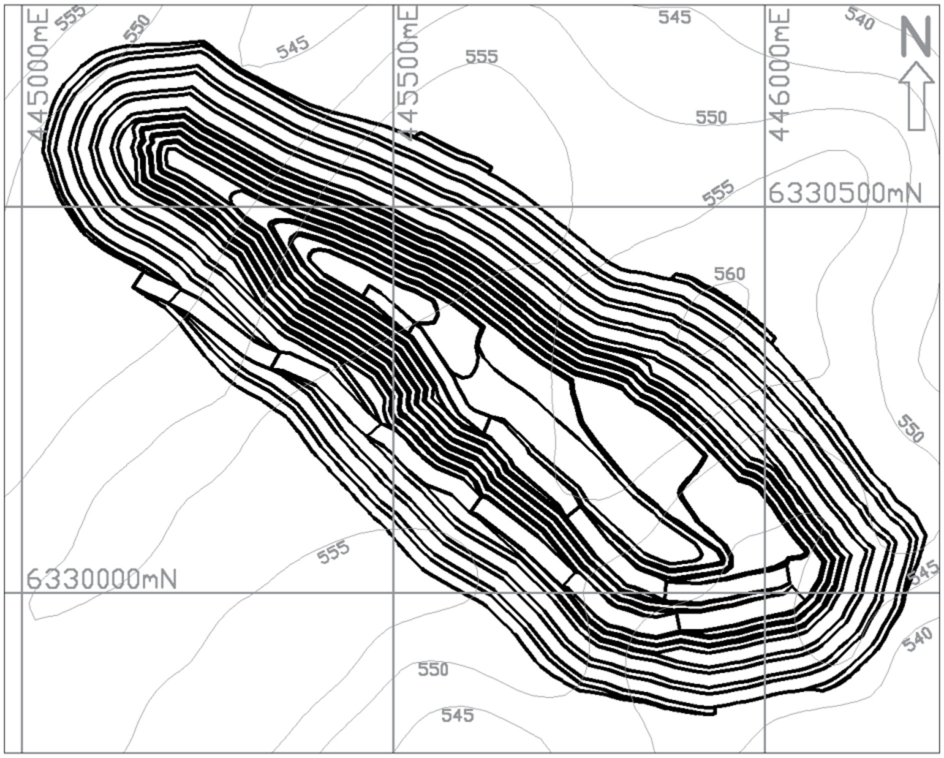


Figure 16-6: Mining Phase 5

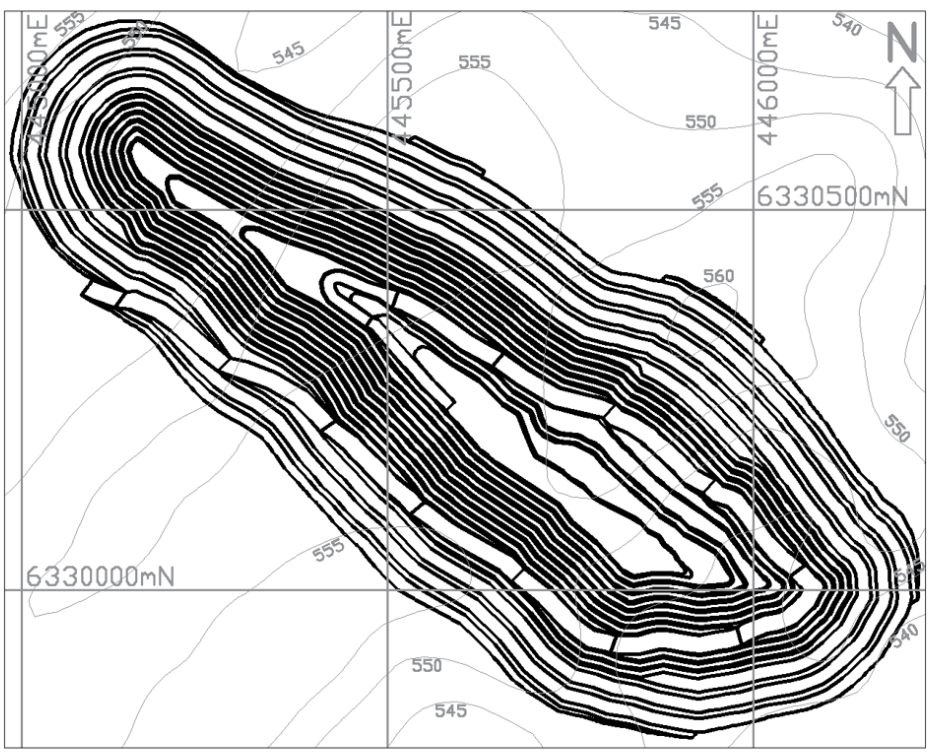


Figure 16-7: Final Pit

16.3 Production Schedule

The mining schedule was developed based on a nominal process plant capacity of 2,750 t/d (approximately 1.0 million tonnes per year). Mill throughput is planned at 75% of nameplate capacity in Year 1, 85% in Year 2, and 100% from Year 3 onward, allowing for a controlled start-up and optimization of plant performance. As shown in Figure 16-8, the Loki Flake Project's total mine life is 20 years, including 1 year of pre-stripping followed by a 19-year production period. The production period comprises 17 years of active mining operations followed by two years of stockpile processing. A stockpiling strategy was implemented, and the cut-off grade was optimized to maximize net present value of the Loki Flake Project. As shown in Table 16-6, mining is conducted at a consistent rate of 10.0 Mt of material annually during the first seven years and slowly diminishing from Y8 to Y17. The mill feed grade fluctuates between 6.2% and 8.5% from Year 1 to Year 17, before reaching 4% to 5% during the processing of the stockpiled material in Years 18 and 19.

Over the 20-year life, the pit is producing 17.9 Mt of mineralized material grading 7.3% graphite and 108.4 Mt of waste material. The LOM stripping ratio is 6.1

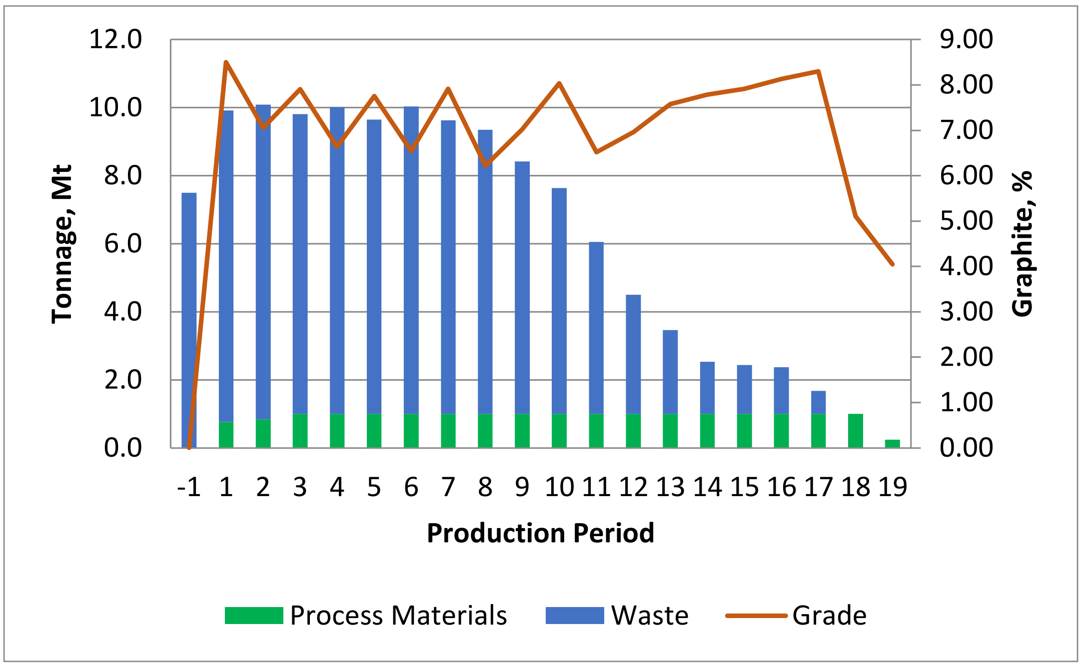


Figure 16-8: Production Schedule

Table 16-6: Production Schedule

Production Year	Tonnage Mined	Waste Mined	Strip Ratio	Pit to Mill	Pit to SPs	SPs to Mill	In SPs	Processed Tonnage	Graphite Grade
	kt	kt		kt	kt	kt	kt	kt	%
-1	7,500	7,500	-	-	-	-	-	-	0.00
1	10,000	9,149	10.8	764	87	-	87	764	8.50
2	10,000	9,243	12.2	756	-	87	-	844	7.05
3	10,000	8,804	7.4	1,003	193	-	193	1,003	7.91

Production Year	Tonnage Mined	Waste Mined	Strip Ratio	Pit to Mill	Pit to SPs	SPs to Mill	In SPs	Processed Tonnage	Graphite Grade
4	10,000	9,012	9.1	987	1	14	180	1,001	6.64
5	10,000	8,643	6.4	1,003	354	-	534	1,003	7.76
6	10,000	9,034	9.4	956	10	46	498	1,001	6.54
7	10,000	8,628	6.3	993	379	8	869	1,001	7.92
8	9,000	8,349	12.8	646	5	355	519	1,001	6.22
9	8,500	7,414	6.8	998	88	3	604	1,001	7.03
10	8,000	6,638	4.9	1,001	361	-	964	1,001	8.03
11	6,000	5,057	5.4	891	52	110	906	1,001	6.52
12	4,500	3,499	3.5	1,000	2	1	906	1,001	6.96
13	3,500	2,462	2.4	999	39	2	943	1,001	7.58
14	2,500	1,532	1.6	963	4	38	910	1,001	7.79
15	2,500	1,430	1.3	1,001	68	-	978	1,001	7.91
16	2,500	1,373	1.2	1,003	125	-	1,103	1,003	8.13
17	1,817	677	0.6	1,001	139	-	1,242	1,001	8.30
18	-	-	-	-	-	1,001	240	1,001	5.11
19	-	-	-	-	-	240	0	240	4.04
LOM	**126,315**	**108,445**	**6.1**	**15,962**	**1,907**	**1,907**	**-**	**17,872**	**7.27**

16.4 Mining Waste Management

Over the LOM, the pit produces 108.4 Mt of waste materials. As shown in Figure 16-9, a Waste Rock and Tailings Co-deposition Facility (WRTCF) was designed to accommodate 17.9 Mt of tailings and 108.4 Mt of waste rock over the life of the mine. The WRTCF will consist of co-deposited dry stacked tailings and waste rock, providing advantages over the conventional slurry Tailings Management Facility design.

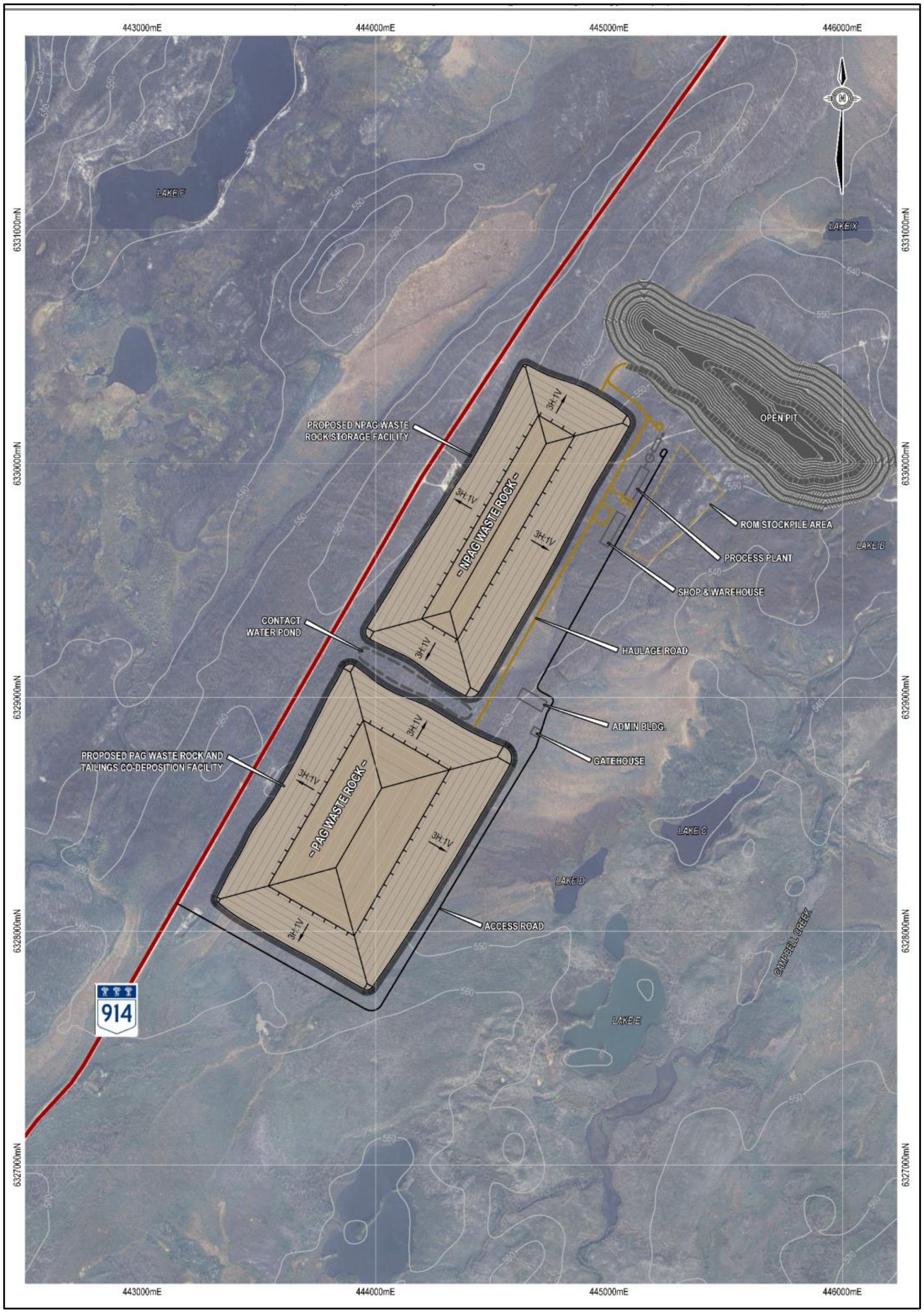


Figure 16-9: Loki Flake PEA Overall Site General Arrangement Plan

16.5 Mining Equipment

Mining equipment selection was based on the material movement schedule, geometry of the open pit, and integration with the rest of the fleet. It was assumed for this PEA that the mining equipment will be leased.

16.5.1 Primary Equipment

Primary equipment was selected to match the scheduled mining operations. Loading will be performed using a 11.0 m^3 diesel hydraulic shovel and a 9 m^3 front-end-loader. Hauling will be performed using 90 t haul trucks.

Blasthole drilling will be performed using 8.5" rotary drills as primary drilling equipment. Blasting will be performed using ammonium nitrate/fuel oil (ANFO) and emulsion with mix proportions of 0.7 and 0.3, respectively. Approximately 37% of the open pit materials is overburden and does not require drilling and blasting.

Table 16-7 lists the primary and support equipment requirements.

16.5.2 Support Equipment

Two 433-kW dozers will be required for waste dump management along with a 372-kW wheel dozer for shovel cleanup. Two 14 ft graders will be used to maintain roads.

Table 16-7: Primary and Support Equipment Requirements

Equipment Requirement	Year																			
	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
Loading Units																				
Shovel (11 m3)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-
Loader (9 m3)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Hauling Units																				
90t Truck	5	8	8	9	9	9	9	9	9	8	8	6	9	8	6	6	6	5	3	1
Drill &Blast																				
Drill (8.5")	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0
Support Equipment																				
Dozer (433-kW)	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1
Dozer (372-kW)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Roller Compactor	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Grader (14 ft)	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1	1	1
Water/Sanding Truck (90 t)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Excavator (3.2 m3)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1

16.5.3 Ancillary Equipment

In addition to the primary and support equipment, Table 16-8 lists the ancillary equipment requirements.

Table 16-8: Ancillary Equipment Requirements

Equipment	Units
50t Rough Terrain Crane	1
Crewcab Pickups	3
Crew Bus	1
Fuel/Lube Truck	1
Shovel/Drill Crew Truck	1
Welding Truck	1
Surface Crew Truck	1
Shop Forklift + Tire Manipulator	1
Skidsteer	1

16.6 Mining Labour

Mining labour requirements were estimated based on a 12-hour long shift, 2 shifts per day and a 2-week-on/2-weeks-off rotation schedule. Operator and maintenance staff requirements are estimated based on the scheduled hours. Operational management labour and staff numbers were estimated from experience with existing operations and anticipated operating conditions for the Loki Flake Project. Mining labour requirements are shown in Table 16-9.

Table 16-9: Mining Labour Requirements

Mining Labour	Year																			
	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
Total Hourly Labour	92	119	119	123	123	123	123	123	123	119	119	111	119	115	103	103	103	99	68	42
Total Salaried Labour	15	15	15	15	15	15	15	15	15	15	15	15	13	13	13	13	13	13	2	1
Total Mining Labour	107	134	134	138	138	138	138	138	138	134	134	126	132	128	116	116	116	112	70	43

17.0 RECOVERY METHODS

The principal economic mineral at the Project is graphite (Cg), occurring within a strongly graphitic shear zone. The mineralized zone is generally bounded by silicified metapelitic rocks of the Wollaston Domain and associated pegmatitic intrusions, with graphite mineralization interpreted to be structurally controlled by the shear zone. The metallurgical test work detailed in Section 13.0, together with the mine plan described in Section 16.0, forms the basis for the selection of recovery methods for the Project.

The Deposit will be mined using conventional open pit mining methods and equipment and processed in a conventional flotation plant, producing a graphite concentrate. This section outlines the metallurgical process, including major design criteria and process description.

17.1 Flowsheet Development

The processing plant is designed to process the plant feed at a nominal throughput of 2,750 t/d, producing a high grade graphite concentrate. The LOM average plant feed grade will be 7.27% Cg (graphite carbon), and the anticipated LOM average recovery will be 92.3%. The LOM average annual concentrate production will be approximately 66,450 t/a at 95% Cg.

The processing plant will consist of the following:

- A run-of-Mine (ROM) mill feed stockpile
- A primary crusher operating in open circuit
- A secondary crusher operating in closed circuit with a vibrating screen
- A crushed mill feed stockpile with reclaim feeders
- A ball mill grinding circuit operating with flash flotation and a classification hydrocyclone cluster
- A rougher flotation circuit comprising rougher and rougher-scavenger flotation, followed by a rougher-scavenger tailings dewatering circuit, including thickening and filtration processes to generate tailings filter cakes for tailings dry stacking at WRTCF
- A rougher concentrate upgrading circuit comprising:
  - one-stage polishing regrinding followed by one stage of cleaner flotation
  - second-stage polishing mill followed by three stages of cleaner flotation
- A fourth-cleaner concentrate sizing circuit to separate the upgraded rougher concentrate into coarse and fine graphite concentrate streams
- A coarse concentrate regrind mill, followed by four additional stages of cleaner flotation (5^{th} to 8^{th} cleaners) to produce a final coarse graphite concentrate
- A fine concentrate regrind mill, followed by five additional stages of cleaner flotation (5^{th} to 9^{th} cleaners) to produce a final fine graphite concentrate
- A concentrate dewatering and product handling circuit, including concentrate thickening, filtration, drying, final product sizing, and packaging.

The simplified process flowsheet is shown in Figure 17-1 and detailed in the following sections.

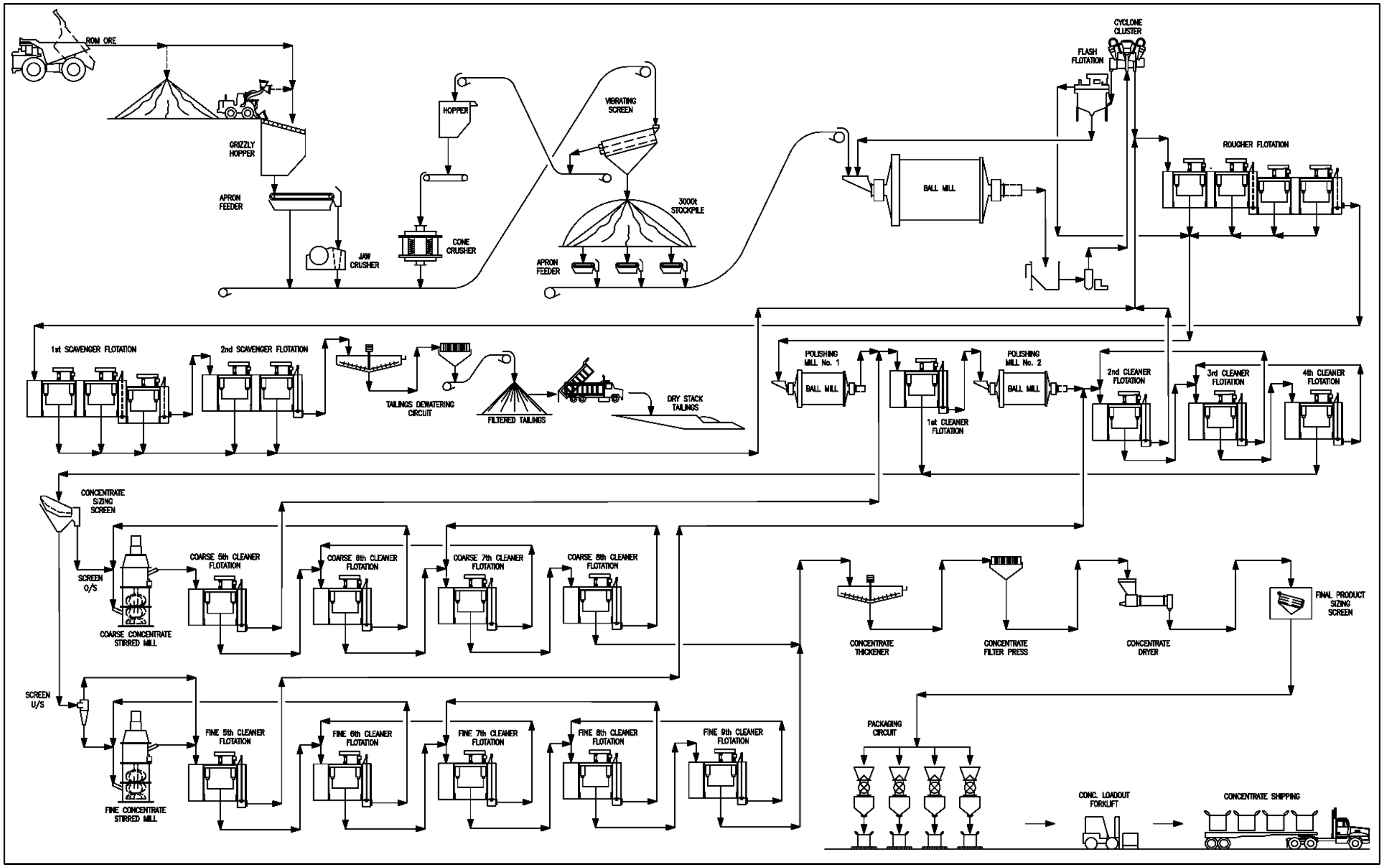


Figure 17-1: Simplified Process Flowsheet

The process flowsheet has been designed to maximize graphite flake preservation while achieving the target concentrate grade through staged grinding and multiple cleaner flotation circuits. Separate treatment of coarse and fine concentrate streams allows optimization of concentrate quality and recovery across the graphite flake size distribution.

17.2 Process Design Criteria

The processing plant is designed at a nominal throughput of 2,750 t/d for an average annual throughput of 1.0 million tonnes at full capacity. The major criteria used for the design of the processing plant are listed in Table 17-1.

Table 17-1: Major Processing Plant Design Criteria

Description	Unit	Value	Source
Plant Feed Characteristics			
Specific Gravity	g/cm^3	2.67	Test Work Results
Bulk Density	t/m^3	1.65	Industry Experience
Moisture Content	% (by wt.)	3.0	Industry Experience
Bond Ball Mill Work Index	kWh/t	13.6	Test Work Results
Abrasion Index	g	0.061	Test Work Results
Operating Schedule			
Shift/Day	--	2	Client
Plant Hours/Shift	h	12	Client
Plant Hours/Day	h	24	Client
Days/Year	days	365	Client
Plant Throughput			
Overall Plant Feed	t/d	2,750	Mine Plan
Overall Plant Feed	Million t/a	1.0	Calculation
Availability			
Crushing Circuit Availability	%	70	Industry Experience
Grinding & Flotation Circuit Availability	%	92	Industry Experience
Grades and Productions Estimates			
Average LOM Head Grade	% Cg	7.27	Mine Plan
Average LOM Recovery	%	92.3	Calculation
Concentrate Grade	% Cg	95	Test Work Results
Annual Concentrate Production (LOM Average)	t/a, dry	66,450	Calculation

The sizing and selection of the crushing and grinding circuits are based on the comminution test work results. The flotation cells are sized based on optimum flotation residence time determined by laboratory test work and scale-up factors and slurry volumetric flow rates. The dewatering circuit is sized based on industry experience on the thickener unit area and filtration rates for similar operations.

17.3 Process Description

ROM material will be transported from the open-pit mine to the ROM stockpile using 90-t haul trucks. The mineralized material will be dumped into a jaw crusher receiving pocket, where a static grizzly screen with a 500 mm aperture will retain oversize material. The oversize material will be broken using a hydraulic rock breaker. The jaw crusher will crush the grizzly screen undersize to a particle size of 80% passing 70 mm. Crushed mill feed will be conveyed to a vibrating screen with a 12.5 mm aperture. The screen oversize material is directed to a cone crusher operating in closed circuit with the screen, producing a final crushed product with a particle size of 80% passing 9.5 mm. The screen undersize material is sent to the crushed mill feed stockpile, with a live capacity of approximately 3,000 t.

The mineralized material will be reclaimed from the crushed mill feed stockpile using two apron feeders (one operational and one standby), each rated at 125 t/h, and conveyed to the ball mill grinding circuit. The grinding circuit will consist of a ball mill operating in closed circuit with a hydrocyclone cluster and will produce a grind size of 80% passing 225 µm. The hydrocyclone underflow will report to a flash flotation cell, while hydrocyclone overflow will be directed to the rougher flotation circuit at a pulp density of approximately 35% to 40% solids by weight. A particle size analyzer will be installed to continuously monitor hydrocyclone overflow and facilitate process control of the target grind size. Steel balls will be added to the ball mill to maintain the grinding media inventory at the required operating rate.

The grinding area will be equipped with sumps and pumps for the collection and recycling of process spillages.

The hydrocyclone overflow from the grinding circuit will feed by gravity to a flotation circuit consisting of four rougher cells and two scavenger cells, each with a volume of 30 m^3. Rougher concentrate will be pumped to the rougher concentrate polishing circuit, while scavenger concentrate will be recycled to the head of the rougher flotation cell bank. Scavenger tailings will be directed to the tailings dewatering circuit, comprising a thickener and filter press, with filtered tailings transported by trucking to the dry-stack WRTCF.

The flotation area will be equipped with sumps and pumps for collection and recycling of process spillages. Rougher and scavenger flotation will be conducted at natural pH using diesel as the collector and pine oil as the frother.

The combined rougher and flash flotation concentrate will undergo first-stage polishing grinding, followed by first cleaner flotation in three 10 m^3 bulk cleaner cells. The first cleaner concentrate will bypass the additional bulk cleaner circuit, while the first cleaner tailings will undergo the second stage of polishing grinding before being upgraded through three additional cleaner flotation stages (2nd to 4th cleaner flotation), comprising two 10 m^3 cells for the second cleaner flotation, one 5 m^3 cell for the third cleaner flotation and one 5 m^3 cell for the fourth cleaner flotation.

The concentrates from the first cleaner flotation and the fourth cleaner flotation will be combined and classified using a concentrate sizing screen to produce coarse and fine graphite concentrate streams, which will be directed to the respective coarse and fine cleaning circuits. The coarse concentrate stream will undergo stirred media milling, followed by four additional stages of cleaner flotation (5th to 8th cleaner flotation), comprising two 5 m^3 fifth-cleaner cells, two 5 m^3 sixth-cleaner cells, one 5 m^3 seventh-cleaner cell, and one 4.2 m^3 eight-cleaner cell.

The fine concentrate stream will similarly undergo stirred media milling, followed by five additional stages of cleaner flotation (5th to 9th cleaner flotation), comprising one 5 m^3 fifth-cleaner cell, one 5 m^3 sixth-cleaner cell, one 5 m^3 seventh-cleaner cell, one 3 m^3 eight-cleaner cell and one 3 m^3 ninth-cleaner cell. The cleaner flotation area will be equipped with sumps and pumps to recycle any spillage.

The combined final graphite concentrate from the coarse and fine upgrading circuits will undergo three stages of dewatering: thickening, filtration, and drying. The graphite concentrate slurry will be pumped to a high-rate concentrate thickener, where flocculant will be added to the thickener feed to assist with the settling of fine particles. The thickener overflow will be recycled to the process water tank for reuse. The thickener underflow will be pumped to a pressure filter at 50% solids (w/w) via a concentrate slurry stock tank. The concentrate dewatering area will be equipped with sumps and pumps to recycle any spillage.

The filtered cakes discharging from the pressure filter are expected to contain approximately 15% moisture (w/w). The filtered cake will be conveyed to a surge bin and then be delivered to an indirect fired rotary dryer, where the moisture content will be reduced to less than 1% (w/w). The water filtrate from the filter press will be recycled to the concentrate thickener to recover any entrained fine solids in the filtrate.

The dried graphite concentrate will be classified into four size fractions (+300 µm, -300+180 µm, -180+106 µm, and -106 µm) using a vibrating screen. The screened products will be discharged to dedicated storage bins prior to packaging.

An automated bagging system will package the dried, sized concentrate into 1 t tote bags. The bags will be weighed and sampled before being packed into shipping containers for storage and transport.

A dust collection system will be installed in the graphite concentrate drying and packing to collect any dust generated during the operation.

17.4 Reagents and Consumables

Reagent consumption estimates at the nominal plant throughput of 2,750 t/d are summarized in Table 17-2. Reagent handling facilities will include unloading, storage, mixing, and distribution systems, with appropriate spill prevention, ventilation, fire protection, and safety controls. All reagent systems will be labelled in accordance with Workplace Hazardous Materials Information Systems (WHMIS) requirements, and operating personnel will receive WHMIS and reagent-handling training.

Major process consumables include flotation reagents, grinding media, crusher and mill liners, screen media, filter cloths, concentrate packaging materials, and laboratory supplies. Maintenance spares for the crushing, grinding, flotation, reagent handling, and assay systems will also be maintained to support plant operations.

Table 17-2: Reagents and Major Consumables Consumption

Reagents	Preparation Method	Estimated Consumption* (t/a)
Diesel	Received as liquid, either in bulk tankers or drums and pumped to an intermediate holding tank. Added to flotation process without dilution by individual metering pumps. The containment area will be equipped with sumps and pumps to recycle any spillage.	230
Pine Oil	Received as liquid, either in bulk tankers or drums and pumped to an intermediate storage tank. Added to flotation process without dilution by individual metering pumps. The containment area will be equipped with sumps and pumps to recycle any spillage.	85
Flocculant	Received as powder in 1t bags; mixed to 0.5% storing strength; diluted to 0.05% strength by inline mixing and	40

Reagents	Preparation Method	Estimated Consumption* (t/a)
	stored in a holding tank; added via progressive cavity pumps to the thickeners.	
Antiscalant	Received as liquid in barrels and pumped to process water and fresh/fire water tank without dilution	5
Grinding Media (75 mm)	Received in 1 t bags; stored in ball bin; manually added to Ball mill	400
Coarse Re-grind Media	Received in 1 t bags; stored in ball bin; manually added to regrind mill	40
Fine Re-grind Media	Received in 1 t bags; stored in ball bin; manually added to regrind mill	20

* The values are rounded and preliminary in nature.

17.5 Assay and Metallurgical Laboratory

An onsite assay laboratory will provide routine analytical support for mining, processing, geological, and environmental activities. Major analytical equipment will include an atomic absorption spectrophotometer (AAS), a LECO carbon analyzer, and an inductively coupled plasma mass spectrometer (ICP-MS).

A metallurgical laboratory will support process control, operational optimization, and ongoing metallurgical investigations. The laboratory will be equipped with sample preparation, comminution, flotation, filtration, particle size analysis, and analytical equipment, including laboratory crushers, grinding mills, flotation cells, sieve analysis equipment, pulverizers, filtration units, sedimentation devices, laser particle size analyzers, pH meters, drying ovens, weighing equipment, fume hoods, dust collection systems, and associated laboratory supplies and reagents.

Appropriate sampling equipment will be provided to support routine plant sampling, metallurgical accounting, process control, and plant survey activities.

17.6 Air and Water Services

The compressed air system for the plant will be provided and is outlined in Section 18.10, while the water supply system is outlined in Section 18.7.

17.6.1 Water Supply and Distribution

Two separate water supply systems for fresh and process water will support the operations.

Process water will consist of the concentrate thickener overflow, the water collected from the WRTCF and waste rock storage area and fresh make-up water. Process water will be directed to a 7 m diameter by 8 m high process water surge tank and pumped to the distribution points in the processing plant. The process water supply and distribution pipelines outside the heated buildings will be buried below frost level or provided with insulation and heat tracing to prevent freezing.

Freshwater will be pumped from wells near the mine and plant site, supplied to a freshwater storage tank and distributed by pumping. Fresh water will be stored in an 8 m diameter by 10 m high freshwater storage tank at the plant site. The freshwater tank will always be full and capable of providing at least two hours of firewater in an emergency.

All freshwater pipelines outside heated buildings will be buried below frost level or provided with insulation and heat tracing to prevent freezing. Freshwater will be used primarily for the following: firewater for emergency use, reagent preparation, mill cooling water, dust suppression, and potable water supply.

The potable water will be treated via chlorination and ultraviolet lamps and stored in a potable water tank at the plant site before delivery to various service points. Potable water for the mine site will be provided as bottled water from the plant site system.

17.6.2 Air Supply and Distribution

Air service systems will be provided at the plant site for the following applications:

- Plant service: high-pressure air will be provided for the process plant for various maintenance activities by dedicated air compressors (one operating and one standby).
- Flotation: Two air blowers (one operating and one standby) will provide low-pressure air for flotation cells.
- Instrumentation: dried and oil-free instrument air will come from the plant air compressors and be stored in a dedicated air receiver.
- Tailings and concentrate dewatering filters: high-pressure air will be provided by dedicated air compressors for filter cake pressing and drying.

17.6.3 Process Control and Instrumentation

The process plant will be controlled using a distributed control system (DCS) integrated with PC-based operator control stations (OCSs) located in the central control room and operated by trained plant personnel.

The DCS will provide process monitoring and control, equipment interlocking, alarm management, data trending, event logging, and report generation. Distributed input/output (I/O) panels will be located throughout the plant and connected via a fiber-optic communication network.

Packaged equipment control systems, including programmable logic controllers (PLCs) associated with major equipment such as mills and filter presses, will interface with the DCS through an Ethernet-based plant control network.

18.0 PROJECT INFRASTRUCTURE

18.1 Overview

In addition to the open pit mine and the processing plant described in Sections 16.0 and 17.0, the Project will require the development of a number of other infrastructure items. The locations of project facilities and other infrastructure items were selected to take advantage of local topography, accommodate environmental considerations, and for efficient and convenient operation of the mine equipment fleet. Buildings will be equipped with high pitched roofs for efficient snow management.

The Project surface infrastructure will include the following:

- An internal access road network with access to Provincial Highway 914 and a gatehouse at the site entrance
- A surface mobile equipment shop and warehouse with mine dry and offices
- A cold storage warehouse
- A surface maintenance laydown and storage area
- A power substation with a 13.8 kV power distribution system
- Water supply and distribution
- Fuel storage and distribution
- A site communication system
- Waste management facilities
- A sewage treatment facility
- Surface water management structures
- Mine and waste rock storage facility
- ROM plant feed storage
- Detonator and explosive storage magazines
- A high-voltage power transmission line connecting the provincial power grid to the power substation on site.
- An existing offsite modular accommodation camp owned and operated by a third party for Abasca’s use.

For infrastructure and facilities related to open pit mining, refer to Section 16.0.

For the processing plant, including the processing equipment, refer to Section 17.0.

18.2 Site General Arrangement

The Project site general arrangement and access road alignments are presented in Figure 18-1. Details of the mineral resource model and the mine design are described in Section 14.0 and 16.0, respectively.

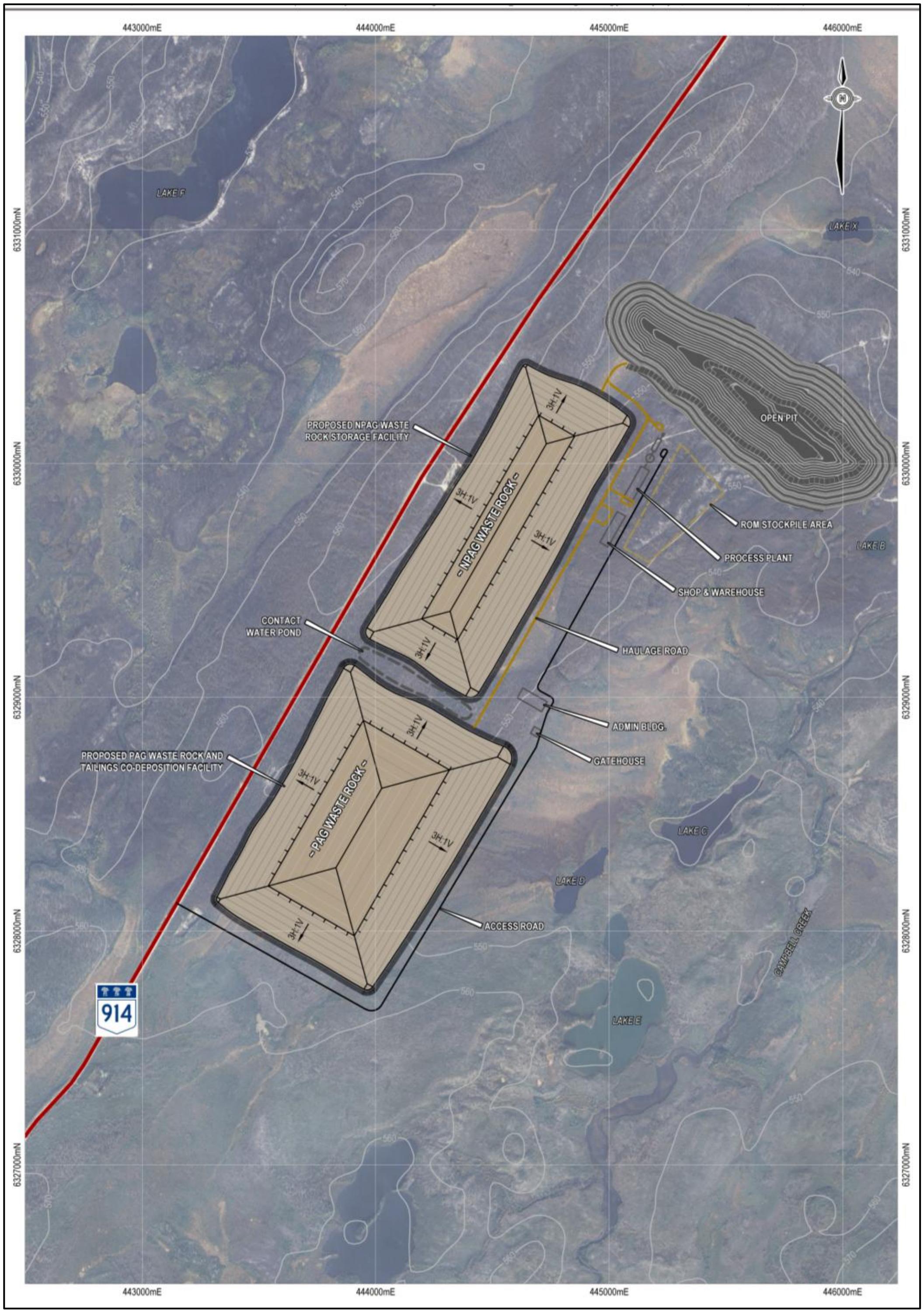


Figure 18-1: Overall Site General Arrangement Plan (Tetra Tech, 2026)

18.3 Access Roads

The Project is located adjacent to the Provincial Highway 914, which also provides vital logistic support year-round for the Key Lake Mine Operations 15 km north of the Project. A 2.3 km two-way access road will provide access from Highway 914 to the gatehouse on the Project site, where incoming vehicles will report to the access controls prior to entering the Project site. A 1.3 km two-way access road will provide internal access from the gatehouse to major buildings and facilities on site. A 2 km two-way haulage road will provide internal haulage access between the open pit mine, the processing plant and the WRTCF. The alignments of these access roads are presented in Figure 18-1.

18.4 Maintenance Shop, Warehouse, Mine Dry and Offices

The maintenance shop, warehouse, mine dry and office complex will be a pre-engineered tension fabric building to accommodate maintenance and repair, warehouse storage, offices, dry areas, and general storage facilities. There are successful examples of pre-engineered tension fabric buildings used for mine operations in sub-arctic regions, such as the Meadowbank Gold Mine in Nunavut, Canada (Figure 18-2). The building structure is supported by standard cargo containers at the base. The material and installation costs of pre-engineered tension fabric buildings are considerably less than traditional steel buildings.

Figure 18-2: Maintenance Facility at Meadowbank Gold Mine/Amaruq Open Pit (Agnico-Eagle Mines, 2025)

This building will comprise:

- Three (3) heavy equipment maintenance bays
- Two (2) light vehicle repair bays
- A 20-ton overhead service crane
- A machine shop
- A welding shop
- An electrical shop
- A storage warehouse with an upper level mezzanine area for offices
- A mine dry area including 100 lockers, washrooms and showers

- A first aid room
- Emergency vehicle storage
- Building services such as power, water, drainage, a septic tank, HVAC, a mechanical service room and an electrical room.

18.5 Storage Warehouses

The 400 m^2 storage warehouse will be located adjacent to the warehouse, truck shop, and mine dry building. It will be a pre-engineered tension fabric building with metal roll-up doors. This building will store equipment, parts, and supplies sensitive to ambient temperature or environment. Equipment, parts, and supplies that are not sensitive to temperature will be stored in cargo containers in the laydown area near the shop.

18.6 Power Supply and Distribution

Power supply at the Project site is assumed to be provided by the provincial utility company (SaskPower) via the existing power grid system near the Project. A new 24 km high-voltage power transmission line will transmit power from the provincial grid system to the power substation on site. Power will be stepped down to 13.8 kV and distributed from the power substation to all end users via the 13.8 kV distribution system. Prefabricated electrical Houses (E-houses) will be used for housing the transformers, switchgears, instrumentation, controls, cooling fans, lightning protection, and grounding. E-houses will step down 13.8 kV to 4,160 V for major equipment, 600 V for ordinary equipment and 240/120 V for lighting and general usage.

All E-houses will be modularized in shipping containers, fabricated, tested, and pre-commissioned at the factory before shipping to the Project site to minimize the field installation and commissioning labour hours. Cable and wire connections will be of the “plug and play” type to minimize installation hours on site. Modularized E-houses can be positioned on compacted gravel pads without concrete building foundations.

Emergency Power

There will be two 500 kW emergency generators at the plant site, retained and refurbished after construction use. The emergency generators will be distributed around the site to pick up critical loads via transfer switches. The generators will be located near the critical loads. Critical loads will be critical process equipment loads, emergency lighting, safety equipment, communications, and minimal mine loads. Critical electronics and safety devices that must stay operative during power outage will be equipped with back-up battery packs.

18.7 Water Supply and Distribution

Supply of fresh water is assumed to be drawn from groundwater sources, via water wells located within or near the Project site, one of which will be operational and one standby. A freshwater tank will be located at the maintenance complex and will be distributed by pumps for potable water feed and for general use.

Fresh water for potable water use will be treated via multimedia filtration and disinfection by chlorination and ultraviolet rays. The potable water treatment module will provide potable water for approximately 100 personnel on site. Treated water will be distributed in a piping ring to serve all potable water users in the buildings.

The clean service/fire water tank located between the process plant and the maintenance complex will have a reserve in the lower portion of the water tank and will be drawn from below the primary water nozzles by the fire water pumps.

Fire water pump skids complete with a diesel-driven fire pump, jockey pump, and controls will be installed. Dedicated fire mains complete with hydrants will be provided at the major site buildings. Fire extinguishers will also be provided throughout the facilities. Sprinkler systems will be installed in the warehouse, the main office and the shop. Fire alarm systems at the site facilities will report to the emergency response/first aid unit at the mine site, which will be monitored 24 h/d.

The feasibility of using groundwater as a water supply source should be further reviewed during future phases of the Project with respect to sustainability of yield, water quality, as well as environmental and permitting considerations.

18.8 Diesel Fuel Storage and Distribution

Certified modular double-walled diesel fuel tanks (also known as “ISO tanks”) will be delivered to the Project site by the fuel supplier and serve as mobile fuel storage tanks on site. These tanks will be laid down in an HDPE-lined and contained fuel farm area near the power plant and the portals. A modular fuel dispensing unit equipped with fuel guns will provide a means for equipment refueling.

The double-walled modular tanks will have visual vacuum gauges to indicate any possible leakage between the walls. The leak and gas vapour detection Instrumentation in the fuel farm area will provide audible alerts locally and wireless alerts remotely to the site operation center. Any fuel leakage will be contained by the HDPE-liner and containment berms in the fuel storage area.

18.9 Propane Fuel Storage

A propane tank on site will store propane fuel for heating and other general uses.

18.10 Mine Plant and Instrument Compressed Air

Compressed air service for plant and instruments will be provided by local air compressors, receivers, and dryers located inside the process plant and the maintenance complex, where required.

18.11 Communications

The telecommunications design for the Project will incorporate proven, dependable, and state-of-the-art systems to ensure that personnel at the mine site will have adequate data, voice, and other communications channels available.

The telecommunications system will be supplied as a design-build package. The base system will be installed during the construction period, then expanded to encompass the operation. The design will include:

- A Voice over Internet Protocol (VoIP) telephone system
- Satellite communications for voice and data
- Ethernet cabling for site infrastructure
- Wireless Internet access
- Two-way radio communications at site
- Satellite TV

A main telecommunication central equipment office will consist of a pre-manufactured trailer in which the main communications contractor will install and test all the main sub-systems for the facilities, prior to shipment. The trailer will form the first block in a system that must support the construction needs of the Project first, and the operating needs of the Project following construction.

Spare parts for critical and main components will be provided to ensure maximum reliability and minimum downtime. A variety of communications media (copper and wireless during the construction phase and fibre optic during the operating phase) will be incorporated in the overall design.

The requirements for communications, particularly satellite bandwidth, are a function of the voice and data requirements of the active participants in the Project.

Technologies and services to be provided include the following:

Construction Phase:

- Local VoIP wireless network
- Satellite link for voice, data, and video services
- PC Local/Wide Area Network (LAN/WAN)
- Trunked mobile radio system
- Internet service
- Private telephone system for voice and fax
- Video conferencing to minimize travel during design and construction
- Ground-to-air communications system (VHF radio)
- Cable television on independent satellite system

Operations Phase (includes selected services above):

- Process monitoring and control for efficient operation and maintenance
- Fibre optic cabling for plant wide communications
- Security access control
- Remote equipment monitoring capacity from corporate offices in Stewart, BC and Vancouver, BC.
- Internet services for workers

A satellite based back up communication system, such as Starlink™, will provide redundancy.

Communications in the underground mine will be supported by a leaky feeder system.

18.12 Waste Management

Domestic solid waste generated over the course of the Project is proposed to be incinerated on site, as per typical mining operations. Domestic waste generated during the operations phase is approximately 0.2 t/d based on an expected onsite workforce of 100.

Hazardous and non-hazardous industrial solid wastes such as tires, waste lumber, scrap metals, plastics, and electrical wastes will be collected and sorted prior to off-site removal by a qualified local contractor for recycling and/or disposed of at an appropriate facility. Hazardous liquid wastes such as laboratory chemical wastes and paints will be collected and temporarily stored on site, with final disposal off site by a licensed contractor. No long-term, on-site waste storage areas have been included as part of the Project.

18.13 Sewage Treatment

The maintenance complex will include a modular sewage treatment plant which will treat the organic wastewater generated in the offices and mine dry inside the maintenance complex and process plant, prior to discharging to the surroundings. The sewage treatment plant will be designed to meet the relevant federal and provincial regulations and effluent quality guidelines.

18.14 Surface Water Management

A site wide water balance will be prepared during the next phase of the Project study to identify the water quality and quantity of all water sources on site and from the surroundings, which will guide the design of all surface water management infrastructure and facilities on site.

For the PEA, cost allowances were included in the cost estimate for:

- Mine dewatering
- Uncontacted water diversion
- Collection, storage, pumping, and treatment of contact water on site, including the raw plant feed wash water from the process plant

Sediment Control Plan

Normal industry practices (such as silt fences, containment ponds, etc.) will be used during the construction of any of the facilities.

Non-contact storm water from above the disturbed areas/facilities will be controlled and diverted before it can become contaminated (i.e. become contact water), thus reducing the sizing of sediment control structures.

All contact water at the plant site will be directed to the sediment control pond located adjacent to each infrastructure pad. Solids will be allowed to settle out, and the water will be removed for site water use, or pumped to the portable water treatment modules or released, based on the water quality. The contact water will be treated prior to releasing, according to applicable regulations and water quality standards.

A modular water treatment plant will treat contact water as required, prior to discharging.

18.15 Accommodation Camp

An existing off-site modular accommodation camp nearby, owned and operated by a third party, will be available for Abasca's use during Project construction and operation.

18.16 Waste Rock and Tailings Co-Deposition Facility (WRTCF)

The WRTCF was designed and developed in consultation with the study team to suit the project setting, design requirements, and project economics. The WRTCF was designed to accommodate 17.9 Mt of tailings and 108.4 Mt of waste rock over the life of the mine. The waste rock was assumed to be 50% Potentially Acid Generating (PAG) and 50% Not-Potentially Acid Generating (NPAG) per information from Abasca. At an assumed average in-place dry density of 2 t/m3 the required waste rock storage volume is 54.2 Mm3 for PAG and NPAG rock combined. The tailings will be filtered and placed in the storage facility. At an assumed average dry density of 1.5 t/m³, the required tailings solids storage volume is 11.9 Mm3. The WRTCF will consist of two stockpiles providing separate storage for PAG and NPAG waste rock as shown on the drawings in Figures 18-3 to 18-5:

- NPAG waste rock storage facility (NPAG-WRSF); and
- PAG waste rock and tailings co-deposition facility (PAG-WRTCF).

18.16.1 Design Criteria

The geotechnical design concept was prepared with reference to:

- CDA Technical Bulletin: Application of Dam Safety Guidelines to Mining Dams (CDA, 2019)
- Mining Association of Canada (MAC): A Guide to the Management of Tailings Facilities (MAC, 2021)
- Guidelines for Mine Waste Dump and Stockpile Design (Hawley and Cunning, 2017)

18.16.2 NPAG Waste Rock Storage Facility

The NPAG rock will be stored in a WRSF approximately 140 m south of the open pit. The facility was designed with 3H:1V side slopes and a maximum height of approximately 68 m. A perimeter berm will be constructed around the facility toe to contain surface water runoff. The perimeter berm and ditch will drain into a contact water collection pond. The footprint of the facility will be cleared, the topsoil stripped, and unsuitable foundation material will be removed.

18.16.3 PAG Waste Rock and Tailings Co-Deposition Facility

PAG waste rock and filtered tailings will be stored in a co-deposition facility with the tailings placed in a central storage cell surrounded by a waste rock berm. The waste rock berm will have a minimum width of 32 m and 3H:1V side slopes. The tailings will be covered with waste rock at a minimum thickness of 3 m. The design maximum facility height will be approximately 80 m.

The footprint of the facility will be cleared, the topsoil stripped, and unsuitable foundation material will be removed. Seepage and surface runoff from the PAG waste rock and filtered tailings will be contained with a linear low-density polyethylene (LLDPE) geomembrane liner. The geomembrane liner will extend 10 m beyond the toe of the facility to include the perimeter collection ditch and berm. Seepage collection below the tailings will be enhanced with a basal drainage system composed of gravel fingers graded to drain to an external geomembrane lined seepage pond.

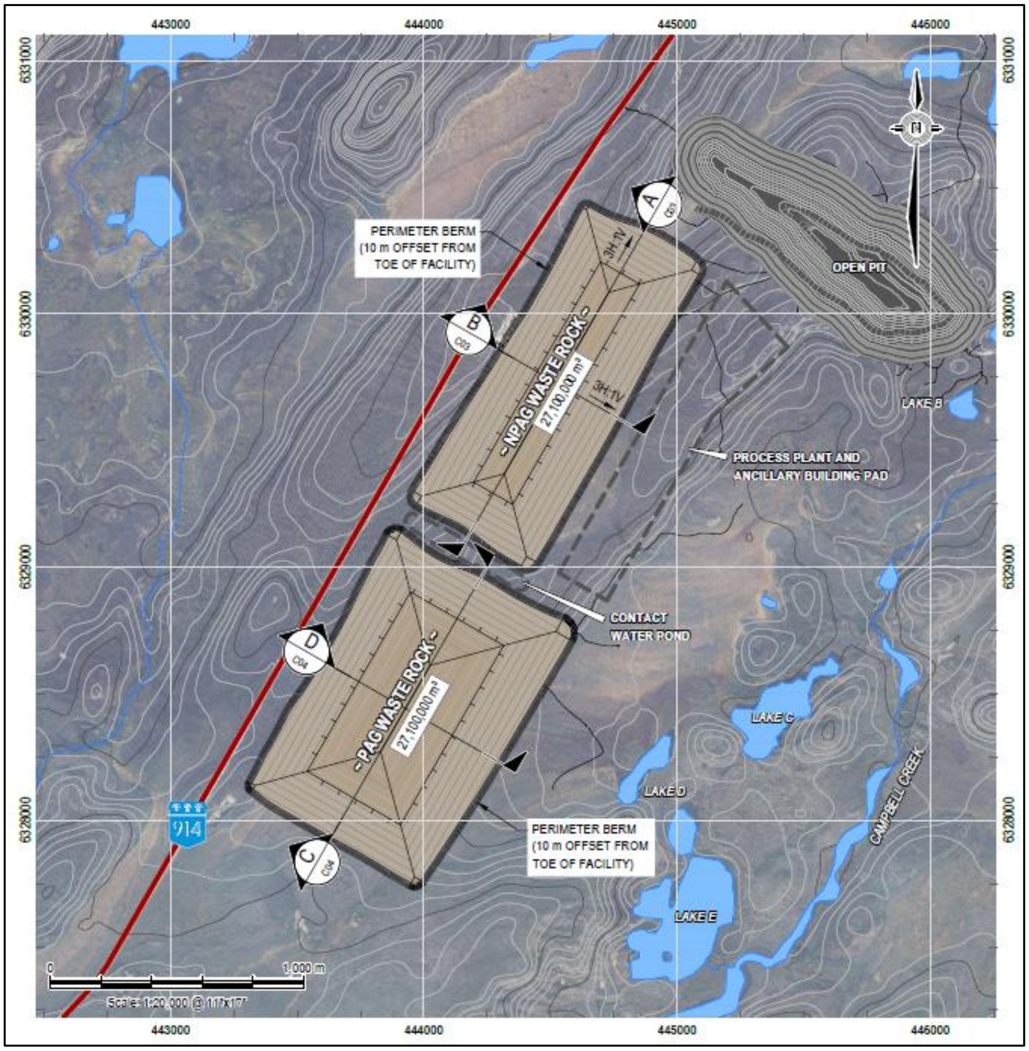


Figure 18-3: WRTCF Plan (Tetra Tech, 2026)

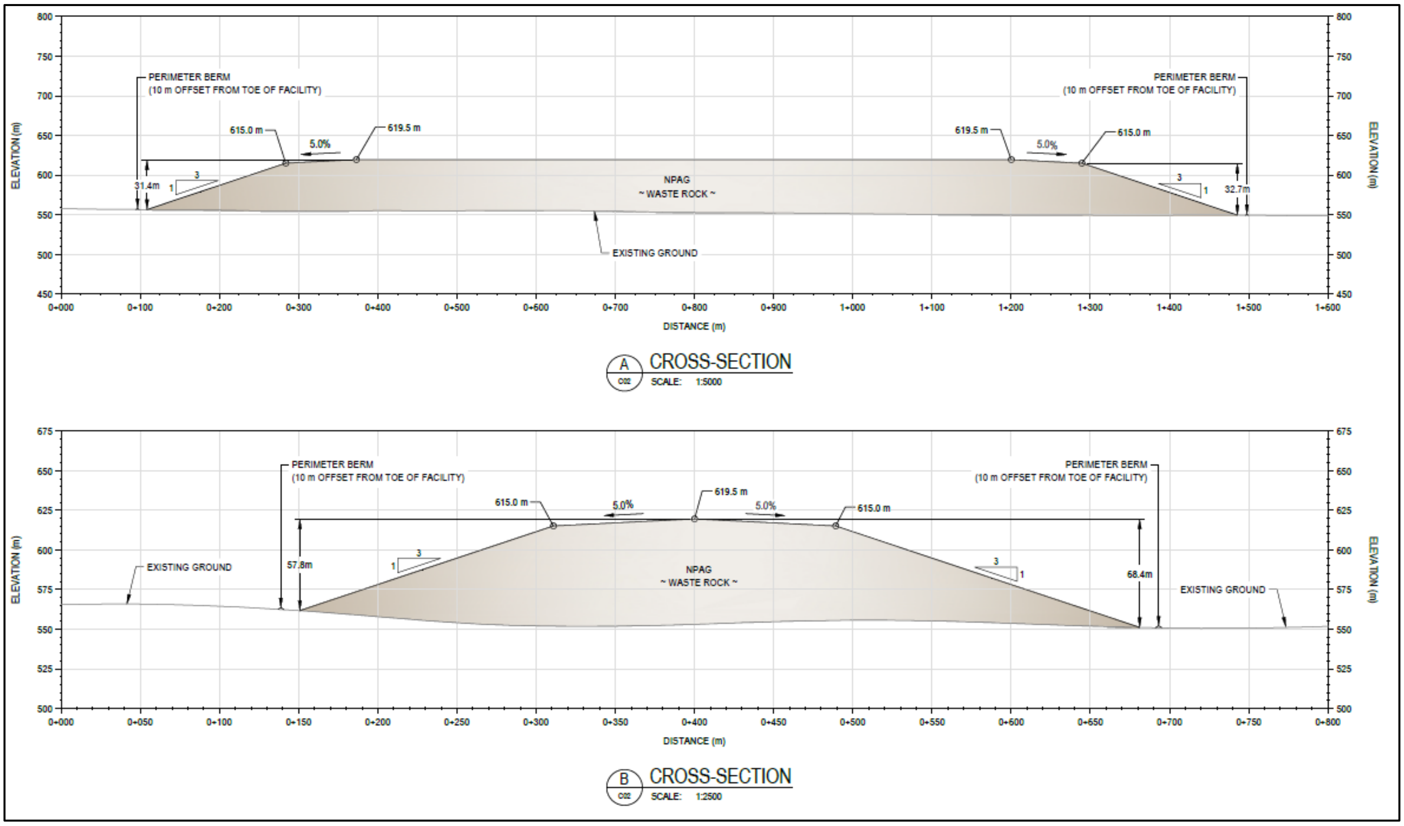


Figure 18-4: WRTCF Cross Sections A-A’ and B-B’ (Tetra Tech, 2026)

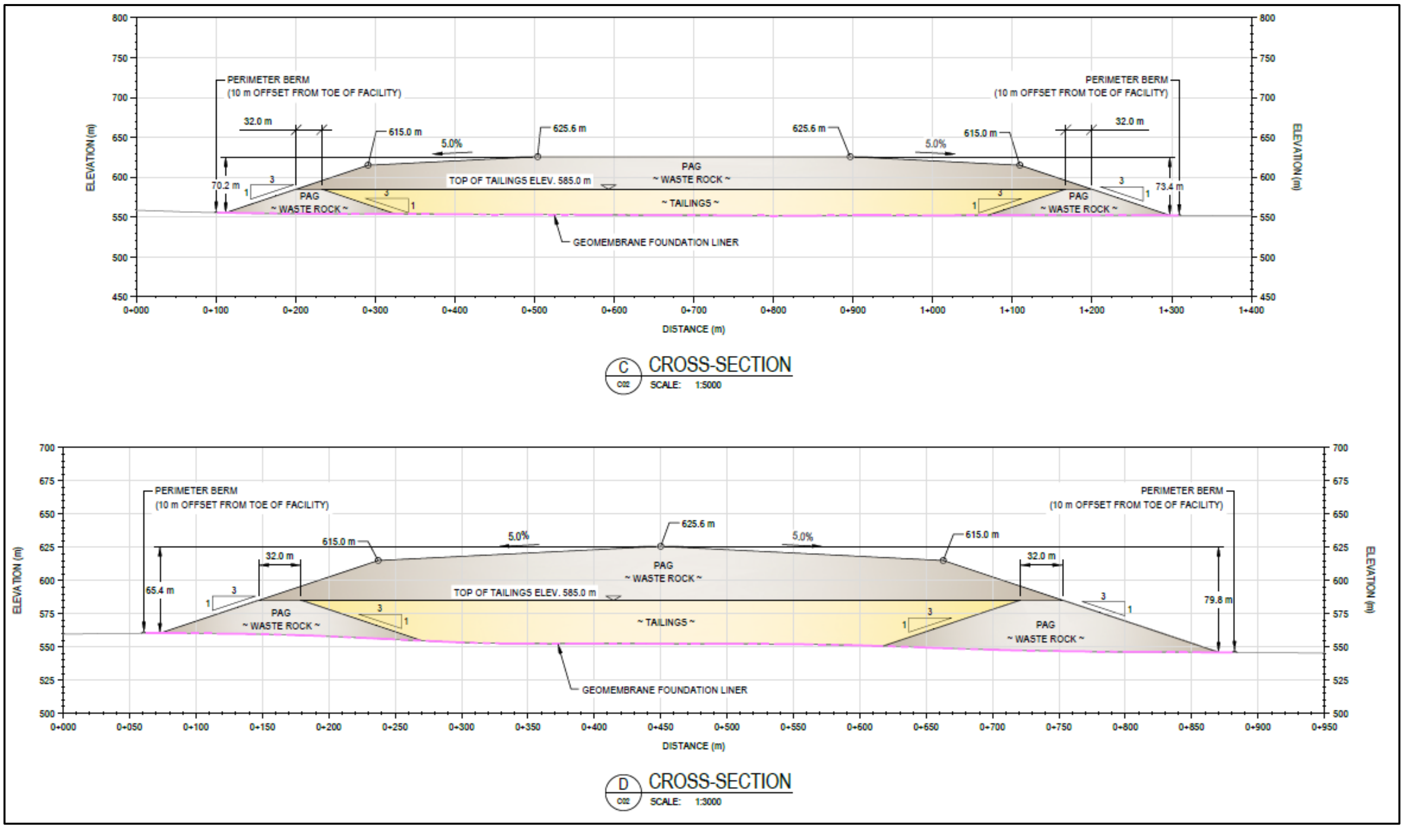


Figure 18-5: WRTCF Cross Sections C-C' and D-D' (Tetra Tech, 2026)

18.16.4 Construction, Operation, and Closure

Foundation preparation for both the NPAG-WRSF and the PAG-WRTCF will include scarification, moisture control, and compaction of in-situ soils. The PAG-WRTCF will require placement of the LLDPE geomembrane liner and basal drains prior to tailings and waste rock placement.

PAG and NPAG waste rock will be received from the open pit as mining proceeds and placed at their respective facilities in lifts of a predetermined thickness to achieve density requirements. Filtered tailings will be hauled to the PAG-WRTCF by truck for placement in lifts and a roller compactor will be used to achieve density requirements. PAG waste rock will be built up during the pre-production mining phase to provide a structural zone around the perimeter of the facility and tailings that do not meet design specifications will be placed on the inner part of the facility. Quality assurance and quality control (QAQC) construction monitoring will be on-site during lift placement. QAQC activities will include in-situ density tests, moisture-density and grain size tests, and visual inspections of placement areas.

Contact water will be pumped to the process plant for use in mineral processing or treated and released.

Closure construction of the NPAG-WRSF will incorporate a dry soil cover and planting and seeding with native species. Progressive reclamation will be undertaken, when possible. The PAG-WRTCF will require regrading of the waste rock surface and placement of an LLDPE geomembrane cover. The geomembrane will be covered with a nominal 1 m thick soil cover that is seeded and planted with native vegetation.

18.16.5 Instrumentation Monitoring

Geotechnical instrumentation will be installed as part of the tailings and waste rock monitoring system. The instrumentation monitoring program will be designed to measure key performance indicators including displacement, pore pressure, and seepage flow. Geotechnical monitoring incorporated in the facility will include:

- Vibrating Wire and Standpipe Piezometers.
- Change detection LiDAR monitoring.
- V-notch weir flow monitoring devices at the outlets of the contact water ditches and seepage locations.

18.17 2026 Preliminary Geochemical Characterization Program

In 2026, BGC Engineering Inc. (BGC) conducted a preliminary geochemical characterization program for the Project. The scope of work comprises a preliminary geochemical characterization of 40 samples of waste rock (n = 39) and mineralized material (n = 1). The objectives of this program were to provide an initial understanding of the acid rock drainage (ARD) and metal leaching (ML) potential of the sampled materials, identify dominant acid-generating and acid-neutralizing phases, and identify parameters of potential concern (POPCs). Samples were selected from historical drilling core using a drilling database provided to BGC by Abasca, which included interval information on lithology, chemical composition, and alteration patterns (Abasca, March 9, 2026). Lithologic units interpreted to be volumetrically significant based on frequency of drillhole intersection within the drilling database (i.e., percentage of total meterage) were selected for laboratory analysis. Namely, samples were collected of biotite gneiss (n = 21), pegmatite (n = 14), and calc-silicates (n = 5). Sample selection also relied on results from prior drillhole four-acid metal assaying (which were provided in the drilling database) to inform the concentrations of major rock forming elements as well as trace elements. Review of alteration types provided in the drilling database was completed as part of sample selection but was not a deciding factor in identifying intervals for geochemical testing.

Samples were submitted to SRC Laboratories (Saskatoon, SK) for static testing, including modified Sobek acid-base accounting (ABA), and shake flask extraction (SFE) testing. Standard metal analysis was completed on all SFE leachates, with radiological parameters included in a subset of 10 samples. As part of the planned data analysis, eight samples are to be selected for further analytical testing, based on characteristics from ABA and SFE test work. The further testing includes whole rock analysis (WRA) and mineralogy using X-ray diffraction (XRD). At the time of this summary, WRA and XRD had not been initiated.

Results from static testing identified the following from the waste rock dataset:

- Paste pH for the submitted samples ranged from pH 7.3 to pH 9.9, with a mean value of pH 7.8, which suggests no sample contained stored acidity nor would be classified as 'already acid-generating'.
- Total sulphur percentages ranged from <0.01 %S to 0.73 %S across the dataset; although the minimum to maximum ranges differed between the three units tested, the average total sulphur percentages within each unit-specific subset were similar (i.e., 0.12 %S for biotite gneiss and pegmatite, 0.10 %S for calc-silicates).
- Sulphur was present almost entirely as sulphide, based on below detection sulphate sulphur contents in the majority of the dataset. Where sulphate was detected, it generally contributed to a minor proportion of the sulphur species.
- Total carbon ranged from 0.02 %C to 4.65 %C, with total inorganic carbon (TIC; e.g., carbonate minerals), graphitic carbon and organic carbon phases contributing to the carbon speciation.
- The range of modified Sobek neutralization potential (NP; from Mine Environment Neutral Drainage [MEND], 1991) values for biotite gneiss and pegmatite samples were similar (i.e., 5.2 to 174 kg CaCO3/tonne, and 2 to 169 kg CaCO3/tonne, respectively), whereas a tighter range was measured for calc-silicate samples (i.e., 9.3 to 23 kg CaCO3/tonne). Comparison of these values to calculated carbonate NP values (i.e., 0.8 to 187 CaCO3/tonne [biotite gneiss]; 0.8 to 349 CaCO3/tonne [pegmatite]; 0.8 to 11.7 CaCO3/tonne [calc-silicates]) showed all three units sampled varied in their dominant acid neutralization minerals from, mainly carbonates, to variable carbonate and silicate minerals, and mainly silicate minerals. For two pegmatite and two biotite gneiss samples, carbonate NP values were notably greater than their complementary modified Sobek NP values, which suggests the potential for iron or manganese carbonate mineral phases.
- Calculated neutralization potential ratio (NPR) values using either modified Sobek NP or carbonate NP values were generally consistent in the classification of potentially acid-generating (PAG) material (i.e., NPR ≤ 2.0; based on MEND [December 2009]). Initial estimates of the percent PAG material from this preliminary dataset, which were conservatively calculated using carbonate NP values, identified 30% biotite gneiss samples as PAG, 50% of pegmatite samples as PAG, and 40% of calc-silicate samples as PAG. These percentages decreased when considering modified Sobek NP values, which captures both silicate and carbonate mineral buffering capacity; however, further mineralogy and kinetic testing are required to assess the reactivity of these silicate minerals.
- Four-acid metal assay data identified 'enriched' concentrations of some parameters relative to 10x average crustal abundances (from Ronov and Yaroshevsky [1972] and Taylor [1964]), particularly: selenium, uranium, and bismuth. All three parameters were identified in biotite gneiss and pegmatite samples, with selenium identified in calc-silicate (only) samples.
- Shake flask extraction results showed elevated leachate concentrations relative to 10x Saskatchewan Environmental Quality Guidelines (SEQGs)1, primarily: selenium, uranium, iron, aluminum, lead, and manganese. Selenium and uranium were identified in biotite gneiss and pegmatite samples, lead in pegmatite samples (only), manganese in biotite gneiss samples (only), and iron and aluminum in all three units. Furthermore, aluminum concentrations were noted to be elevated in the majority of samples and future planned mineralogical and kinetic testing will help elucidate, for example, whether these elevated conditions are associated with reactive silicate minerals that may have weathered during sample storage, the presence of soluble aluminum-bearing primary mineral phases, or the presence of colloidal sized clay minerals.

- Radiological analysis of the SFE leachates identified activity-based radium-226 concentrations in greater than direct comparison to its SEQG in 5 of 10 samples, all from biotite gneiss and pegmatite samples.
- The single mineralized material sample was classified as PAG, based on a high sulphide percentage, and low NP (i.e., modified Sobek NP was 7 kg CaCO3/tonne; carbonate NP was 17.5 kg CaCO3/tonne). Enriched concentrations from four-acid metal assay testing on the single mineralized material sample were identified for selenium and bismuth, like that of waste rock, as well as copper, cobalt and molybdenum. Leachate concentrations from SFE testing on the mineralized material sample identified selenium (only) as being present at an elevated concentration, relative to 10x the SEQG.

Overall, initial geochemical results suggest that waste rock for the Deposit presents some ARD and ML potential. Key POPCs for the Project include uranium and selenium, as they were identified as having enriched solid-phase concentrations (from four-acid metal assays) and were present in elevated concentrations (in SFE leachates); however, other POPCs could include bismuth, aluminum, iron, lead and manganese. Further testing (i.e., WRA and XRD) is ongoing at the time of this reporting as part of the preliminary geochemical characterization work.

Collectively, these preliminary results will support the development of future recommendations for the Project, potentially including expanding the static testing dataset of the sampled units, including other lithologies not yet geochemically tested, and initiating kinetic testing to assess long-term acid generation and metal leaching rates.

18.18 2025/26 Geotechnical and Hydrogeological Site Investigation Program

Misty Clifton Engineering LP (MCLP) conducted two phases of geotechnical and hydrogeological site investigation for the Project. The preliminary investigation was completed in July and August 2025 (Misty Clifton Engineering Ltd (MCLP 2026a) and the infill investigation in February and March 2026 (MCLP 2026b and 2026c). This report presents the integrated analysis, interpretation, and preliminary recommendations from the combined dataset in support of feasibility mine planning and future Environmental Impact Statement (EIS) preparation.

Field Programs

A total of 26 boreholes were completed across the two programs: 16 sonic boreholes through the glacial overburden (19 m to 47 m) and 10 diamond boreholes in the competent basement rock (to 328 m). Instrumentation comprised 34 monitoring instruments in total, including vibrating wire piezometer nests, standpipe piezometers, and one production well (KLS-26-129PW). Field testing included 213 standard penetration tests (SPT) in the overburden, packer testing in four bedrock intervals, in-situ falling-head slug tests in six standpipes, a stepped-rate and constant-rate pumping test, and acoustic and optical televiewer surveys in three diamond boreholes. Laboratory testing encompassed 288 water contents, 80 hydrometer analyses, 16 Atterberg limit tests, six direct shear tests, 60 uniaxial compressive strength (UCS) tests, and 30 Poisson's ratio measurements.

Hydrostratigraphy

Six hydrostratigraphic units (HSUs) were identified and characterised across the site. The Till (Till) was the most laterally continuous unit, present at 16 of 17 boreholes with combined thicknesses of 5 m to 48 m, and represented the primary overburden HSU with a median hydraulic conductivity (P50) of 3×10^{-5} m/s. Saturated Glaciofluvial Sands (GF) were present at six boreholes, concentrated on the southeastern pit margin and the southern infrastructure corridor, with saturated thicknesses of up to 23 m and a median hydraulic conductivity (P50) of 1×10^{-4} m/s, the highest overburden hydraulic conductivity recorded on site and the most hydraulically significant unit for both dewatering and contaminant pathway assessment. Interglacial Coarse-Grained Sediments (IGS-CG) were laterally discontinuous, present at eight boreholes, with log-normal hydraulic conductivity distribution and a median hydraulic conductivity of 9×10^{-5} m/s that was corroborated by slug testing.

Interglacial Fine-Grained Sediments (IGS-FG) were the least continuous unit, present at five boreholes, with a median hydraulic conductivity of 2×10^{-7} m/s and wide variability spanning five orders of magnitude. The Basal Till and Regolith (BT-R) was present at all boreholes and exhibited a high SPT refusal rate of 65%; slug testing consistently returned hydraulic conductivity values of 6×10^{-6} to 1×10^{-5} m/s, higher than GSD-derived estimates, which was attributed to sample disturbance during sonic drilling. Seven packer tests were conducted in the intact un-weathered Bedrock (BR) across three boreholes. Hydraulic conductivity values ranged from 2×10^{-11} to 3×10^{-10} m/s, with a geometric mean of 9×10^{-11} m/s, consistent with a competent low-permeability crystalline rock mass.

Bedrock Characterisation

The competent basement comprised biotite gneiss (~45%), granitic gneiss (~25%), pegmatite (~15%), calc-silicate (~6%), and a graphite-bearing shear zone (~3%). UCS values averaged 110 MPa for biotite gneiss, 126 MPa for granitic gneiss, 120 MPa for pegmatite, and 115 MPa for calc-silicate. RMR values ranged from Poor to Very Good and generally improved with depth as near-surface weathering diminished. Four discontinuity sets were identified from televiewer surveys and oriented core, with steeply dipping conjugate pairs dominant across all boreholes. The graphitic shear zone, intersected in KLS-26-117D between 160 m and 290 m depth, was characterised by slickensided graphite-infilled surfaces, poor to fair RQD, and elevated fracture density relative to the surrounding intact gneiss.

Groundwater Regime

The shallow overburden water table ranged from 537.5 to 553.7 masl and reflected regional flow from west to east toward Norse Lake and Campbell Creek. Overburden heads were temporally stable, varying by less than 0.5 m over the monitored period. Piezometric heads in the BT-R unit broadly tracked the overlying water table with small downward vertical gradients at most locations, consistent with a recharge-focused shallow flow system. Anomalous upward flowing artesian conditions were confirmed at four locations: KLS-25-079D (BR, 556.75 masl, 2.8 m above ground surface), KLS-26-117D (BR, 548.28 masl, 3.0 m above ground surface), KLS-26-128S (BT-R, 547.30 masl, 5.5 m above ground surface), and KLS-26-126S (GF, 553.73 masl). Bedrock piezometric heads ranged from 499.7 to 556.8 masl, a differential of 57.1 m across approximately 320 m of horizontal separation, consistent with hydraulically isolated pressure compartments rather than a connected flow system.

Structural Controls

The spatial pattern of artesian anomalies, the sharp lateral bedrock head discontinuity, a NE-SW trending regional fault mapped in the Saskatchewan 1:250,000 geological compilation, and cross-cutting NE-SW electromagnetic conductors near the deposit collectively supported the presence of a discrete structural zone. This zone was interpreted to have acted simultaneously as an upward pressure conduit transmitting deep bedrock pressures into the overlying overburden and as a lateral hydraulic barrier within the bedrock separating high-pressure and low-pressure compartments.

Groundwater Chemistry

Eleven groundwater samples were collected across both programs. The dominant overburden water type was calcium-magnesium bicarbonate, consistent with short residence times and active recharge through glacial sediments. Total arsenic exceeded the CCME Freshwater Aquatic Life guideline (5 µg/L) at six locations across both programs; dissolved arsenic was below the guideline everywhere, potentially indicating co-precipitation associated with colloidal iron oxyhydroxides. Selenium was below detection at all sites across both programs. Uranium concentrations increased with depth in the overburden, reaching a maximum of 10 µg/L in BT-R-screened wells, but all values remained below the Saskatchewan Drinking Water Standard of 15 µg/L. The single bedrock sample (KLS-25-076D) was substantially more mineralised (EC 5,620 µS/cm), with chloride and sulphate enrichment consistent with longer groundwater residence time in fractured gneiss, providing chemical confirmation of hydraulic compartmentalisation between the overburden and bedrock systems.

19.0 MARKET STUDIES AND CONTRACTS

Graphite is commonly a crystalline form of carbon but can also take an amorphous form. It is one of the few major naturally occurring forms of carbon but can also be produced synthetically. Diamonds are the other naturally occurring crystalline form of carbon, while coal is a naturally occurring amorphous carbon. As a natural form of carbon, graphite holds many unique characteristics making it an in-demand mineral with high industrial utility and value. Graphite is a stable and soft form of carbon that has both metallic and non-metallic mineral properties, including lubricity, thermal conductivity, electrical conductivity, and chemical inertness, among others. These qualities make graphite suitable for a wide variety of industrial applications including, but not limited to, steelmaking, battery production, brake linings, lubricants, and the production of refractory materials. Notably, graphite represents the largest component by weight of lithium-ion batteries, as it is used to produce anodes. However, the utility and value of natural graphite is dependent on the characteristics and type of mineral being mined from a specific deposit (Figure 19-1).

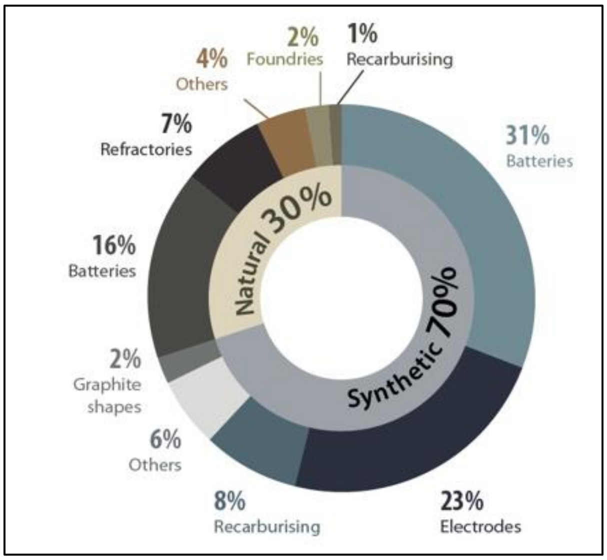


Figure 19-1: Global Graphite Usage (Source: Natural Resources Canada)

Natural graphite typically takes the form of flakes or other types of masses, such as lumps or chips, and is mined from metamorphic deposits. Crystalline flake graphite is typically mined and then processed (often through flotation) to create graphite concentrate, a high-carbon graphite product, typically with a carbon content of 90% or higher and generally marketed commercially as various sized flake or powder products.

Production

According to data from the United States Geological Survey, global mined (natural) graphite production in 2025 was reported to be 1.80 million tons, up 16% from 1.55 million tonnes in 2024. China is the largest producer of graphite, representing 78% of global production, with other major producers including Madagascar, Tanzania, Brazil, and Mozambique. Canada's graphite production was reported to be 8,000 tons in 2025 but is home to 5.9 million tons of reserves (Figure 19-2).

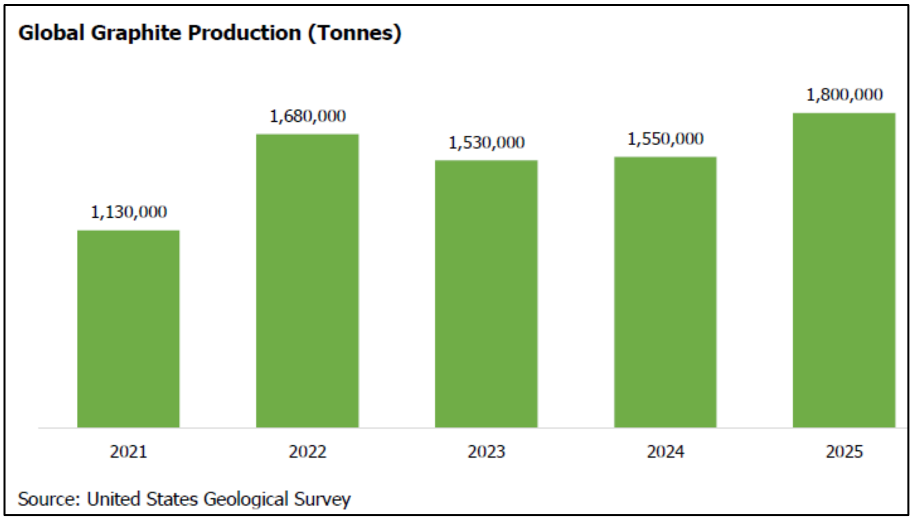


Figure 19-2: Global Graphite Production (Source: United States Geological Survey)

Demand

According to data from Natural Resources Canada, total global natural flake graphite was 1.7 million tonnes in 2024 and demand for synthetic graphite was 4 million tonnes. Combined, total demand for graphite was 5.7 million tonnes in 2024, up 9.62% from 2023. Asia represents the largest market for graphite globally, representing 84.5% of the demand for natural flake graphite (Figure 19-3).

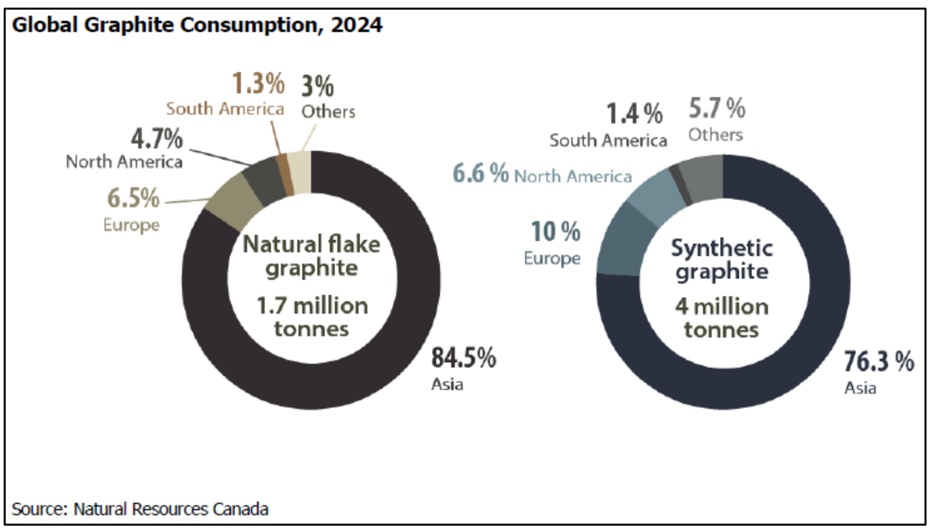


Figure 19-3: Global Graphite Consumption (Source: Natural Resources Canada)

Rising demand for graphite is largely being fueled by growing demand for batteries, which represent approximately half of the total demand for natural graphite and is driving demand for graphite concentrate specifically, as lithium-ion batteries require high grade graphite concentrate. Batteries are in high demand across a wide variety of sectors, including consumer electronics, renewable energy storage, and electric vehicles. One of the primary growth drivers for graphite is the growing adoption of electric vehicles. While growth in recent years of electric vehicles has been strong, looking ahead global growth in demand for electric vehicles is expected to continue. A report from BloomberNEF forecasts that passenger electric vehicle sales will grow 17.6% by 2030 from 2024.

Demand for graphite is also being impacted by increasing steel production. Graphite electrodes are utilized in steelmaking, primarily in steelmaking utilizing electric arc furnaces. The Global Energy Monitor reports that electric arc furnace capacity is projected to increase 24% by 2030 and that half of all new steelmaking projects under development plan to use electric arc furnace technology.

In terms of trade in graphite, global imports of natural graphite (HS 2504) totaled CAD$880 million in 2025 in terms of value and 523 million kgs by volume. Since 2021, the value of global imports of natural graphite have grown at a CAGR of 1.54%, while the volume has decreased at CAGR of -15.06%. Between 2021 and 2025, 93% of all global imports consisted of natural graphite in powder or flakes (HS 250410) (Figure 19-4).

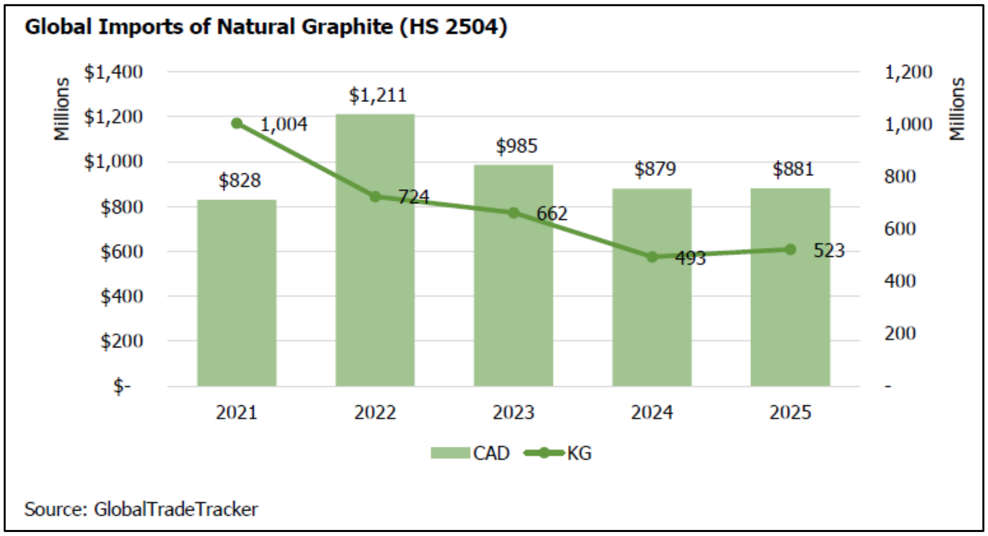


Figure 19-4: Global Imports of Natural Graphite (Source: Global Trade Tracker)

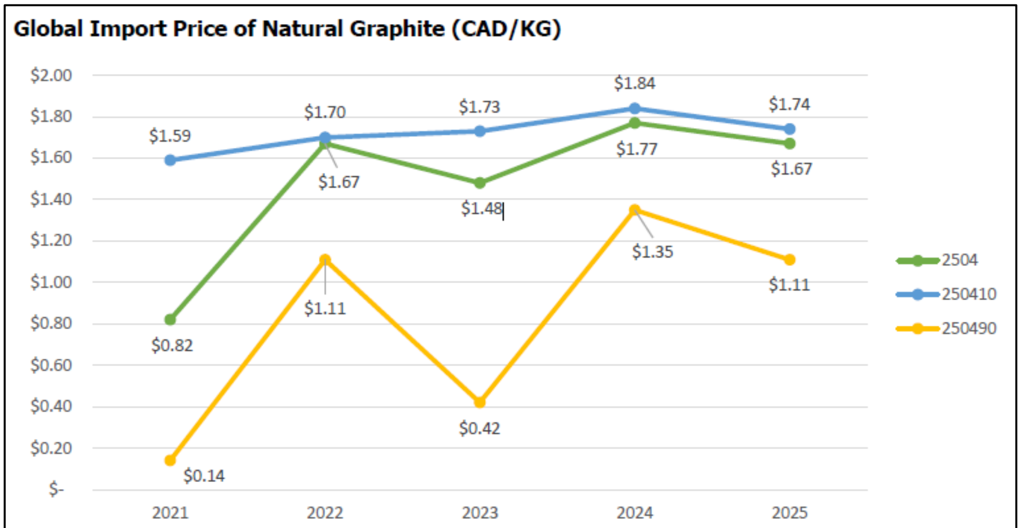
In 2025, the United States (U.S.) was the largest importer of natural graphite accounting for 23% of global imports by value and 16% by volume. Other major importers in 2025 included China, Indonesia, India, Germany, Japan, and South Korea. The U.S. currently has no domestic natural graphite production, but projects are being explored. According to the USGS data, in 2025, 73% of U.S. imports consisted of flake and high-purity graphite.

Pricing

Natural graphite pricing is determined by private commercial sales and not market prices, as there are no spot or futures markets for graphite. Given this, data on natural graphite and graphite products is limited. Import data is a source of reliable data on graphite prices globally and by country. However, it does not account for differences between products contained within a single HS code. For example, most graphite concentrate products would be imported under HS 250140, but this would not differentiate between flake size, powder composition, or quality. Thus, import data should be viewed as representative of average market prices for graphite, as opposed to specific product prices.

Throughout 2025, all natural graphite imports globally were imported at an average price of CAD$1.67/KG (CAD$1,670/tonne). Graphite concentrate is typically imported under HS 250410; products under this code were imported at an average price of CAD$1.74 (CAD$1,740/tonne) in 2025. The price of imports under HS 25410 has increased over the past five years at a CAGR of 2.28% (Figure 19-5).

Looking at import prices within specific countries, there are differences in prices being paid in different markets and for different products. Within the U.S., the largest importer of natural graphite, graphite flakes were imported at an average price of CAD$1.65/kg (CAD$1,650/tonne) in 2025, while natural graphite powder was imported at an average price of CAD$3.15/kg ($3,150/tonne). Similarly, prices varied by type of product in other markets and some prices were higher and lower depending on the destined market. The Graphite concentrate pricing used for PEA economic analysis is discussed in Section 22.0.



Source: GlobalTradeTracker

Notes: HS 2504 – Natural Graphite

HS 2504.10 – Natural Graphite in Powder or Flakes

HS 2504.90 – Natural Graphite, Excluding in Powder or Flakes

Figure 19-5: Global Import Price of Natural Graphite (Source: Global Trade Tracker)

Table 19-1: Price of Natural Graphite Imports by Top Three Import Countries & Product, 2025 (Source: Global TradeTracker)

Price of Natural Graphite Imports by Top Three Import Countries & Product, 2025

Country	HS Code	HS Description	Import Price (CAD/KG)	Import Price (CAD/Tonne)
United States	2504	Natural Graphite	$2.36	$2,360
	2504.10.1000	Natural Graphite Crystalline Flake, Except Flake Dust	$1.65	$1,650
	2504.10.5000	Natural Graphite in Powder or Flakes, nesoi	$3.15	$3,150
	2504.90.0000	Natural Graphite, Except Powder of Flakes	$0.88	$880
China	2504	Natural Graphite	$0.83	$830
	2501.10.10	Phosphorus Flake Natural Graphite	$0.70	$700
	2504.10.91	Spheroidized Graphite	$13.53	$13,530
	2504.10.99	Other Powder or Flake Natural Graphite	$3.63	$3,630
	2504.90.00	Natural Graphite, Except Powder of Flakes	$0.86	$860
Indonesia	2504	Natural Graphite	$1.08	$1,080
	2504.40.00	Natural Graphite in Powder or Flakes	$1.09	$1,090
	2504.90.00	Natural Graphite, Except Powder of Flakes	$0.91	$910

Source: GlobalTradeTracker

Note: Price is based on imported customs value as reported by each country.

The graphite concentrate produced by the Project is expected to yield an average grade of 95% Cg or higher. The current project concept is shipping the concentrate from the Project to Saskatoon, and from there the product would be consolidated for shipment to domestic and foreign processing facilities for purification. The concentrate price used for the economic analysis is US$1,450/t. The concentrate handling and transportation costs are included in the economic analysis. Currently, there have been no concentrate sales contracts established for the Project. Table 19-2 shows the offsite charges and assumptions used in Section 22.0.

Table 19-2: Offsite Charges and Assumptions (Tetra Tech, 2026)

Item	Unit	Value
Payability	%	100.0%
Insurance	% of Product Values Produced	0.125%
Concentrate Transport Loss	% of Product Values Produced	0.15%
Representation	US$/t Dry Concentrate	$2.00
Royalty	% of Product Values Produced	0.00%
Concentrate Transport		
Concentrate Moisture	%	~1%
Site to Saskatoon, Sask.	US$/t Wet Concentrate	43.20

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Setting

The Project is located approximately 15 km southwest of the Key Lake uranium mill and 185 km north of Pinehouse, Saskatchewan (see Figure 4-1 in section 4), on the margin of the Athabasca sedimentary basin region, which is coincident with the Athabasca Plain Ecoregion and Boreal Shield Ecozone. The Athabasca Basin to the north of the project covers about 100,000 square kilometers in northern Saskatchewan and the northeastern corner of Alberta, Canada. The annual temperature of the basin and surrounding region generally ranges from -40°C in the winter to 35°C in the summer. The mean temperatures for January are -22°C and 16°C for July. Annual average rainfall is 225 mm and snowfall 2,150 mm. Lake ice thaws in late April and returns in late October. The obtained monthly data are presented in Table 20-1.

Topography is generally flat lying with low ridges and hills reaching up to several tens of metres in relief. A strong north-easterly structural grain is evident in the topography. The area is characterized by numerous lakes, creeks, and ponds. Soil thickness on the ridges and hills is minimal, and bedrock exposure is scarce but is found within recent burns and along large ridges. Rare outcrops are typically clustered and covered in lichens and moss. The vegetation is characteristic of subarctic tundra, dominated by small coniferous and deciduous trees, as well as shade-tolerant shrubs with grasses and feathermoss carpets at their base. Low-lying areas between hills and ridges consist of poorly drained muskeg swamps with scattered tamarack and black spruce.

The Project is accessible all year round by provincial highway 914, which runs through the project area. The nearest communities are Pinehouse and La Ronge, and are respectively located approximately 185 km and 400 km south of the project area by road. Supplies, groceries, fuel, and accommodations can be found in these communities. Accurate Industries, who have a partnership with English River First Nation operates a commercial camp at kilometer 210 along Highway 914 and can provide accommodations, fuel, and geological logging facilities. The camp is an ATCO trailer style facility, and plans are in place to expand this camp for future operations and may include construction of an airstrip by Accurate at the facility, which is accessible to Abasca.

All infrastructure currently on the Project is non-permanent. The Government of Saskatchewan requires a land disposition to be issued for all permanent structures. There is access to fresh water close to the project and the provincial electrical grid is located within approximately 15 km of the Project, which supplies electricity to the operations at Key Lake. The nearest cellular network service is via the signal from the Key Lake operations. Service is available in the northern area of the project; however, a signal booster can be used in other parts of the project area.

Table 20-1: Climate Normals for Average Air Temperature and Total Precipitation at Key Lake, SK (15 km northeast of the Project)

Period	Average Air Temperature (°C)	Total Precipitation (mm)
January	-22	10.1
February	-19	10.3
March	-12	14.1
April	-2	24.4
May	7	40.3

Period	Average Air Temperature (°C)	Total Precipitation (mm)
June	14	68.0
July	16	75.8
August	15	60.3
September	8	46.1
October	0	25.9
November	-11	15.1
December	-20	9.4

20.2 Environmental Baseline Program

In 2025, Abasca engaged CanNorth Environmental Services to conduct environmental studies on the Project. The purpose of the studies was to initiate data collection for components that require baseline datasets, specifically hydrology, water chemistry, and bathymetry to support a future Environmental Impact Assessment. Studies and data collection will continue into 2026 and 2027, including further work on aquatic, terrestrial and heritage resources as well as work on meteorological, hydrogeological and geochemical conditions. This work will advance project development and support future submissions to meet regulatory requirements.

20.3 Environmental Assessment Process

The Environmental Assessment Process follows a structured path (Figure 20-1) managed by the Environmental Assessment Branch (EAB):

- Self-Screening & Technical Proposal: Proponents evaluate their project using the Technical Proposal Guidelines (Saskatchewan EA, 2021). If it triggers development criteria, they submit a Technical Proposal to the EAB.
- Ministerial Determination: The Minister of Environment issues a decision confirming whether the project is classified as a development. If no, the project proceeds directly to standard permitting.
- Terms of Reference (TOR): If an EIA is required, the proponent prepares a draft TOR outlining the scope of required baseline studies. The EAB and the Saskatchewan Environmental Assessment Review Panel (SEARP) review and approve the final TOR.
- Conducting the EIA & Creating the EIS: The proponent conducts fieldwork to assess Valued Components (e.g., air, water, wildlife, and Indigenous land use). They compile these findings into an Environmental Impact Statement (EIS).
- Technical and Public Review: SEARP evaluates the EIS technical merits while the public is given a statutory review period to provide feedback.
- Ministerial Decision: The Minister reviews all technical findings and public inputs before granting or withholding approval. Legally binding conditions are attached to any approved project.

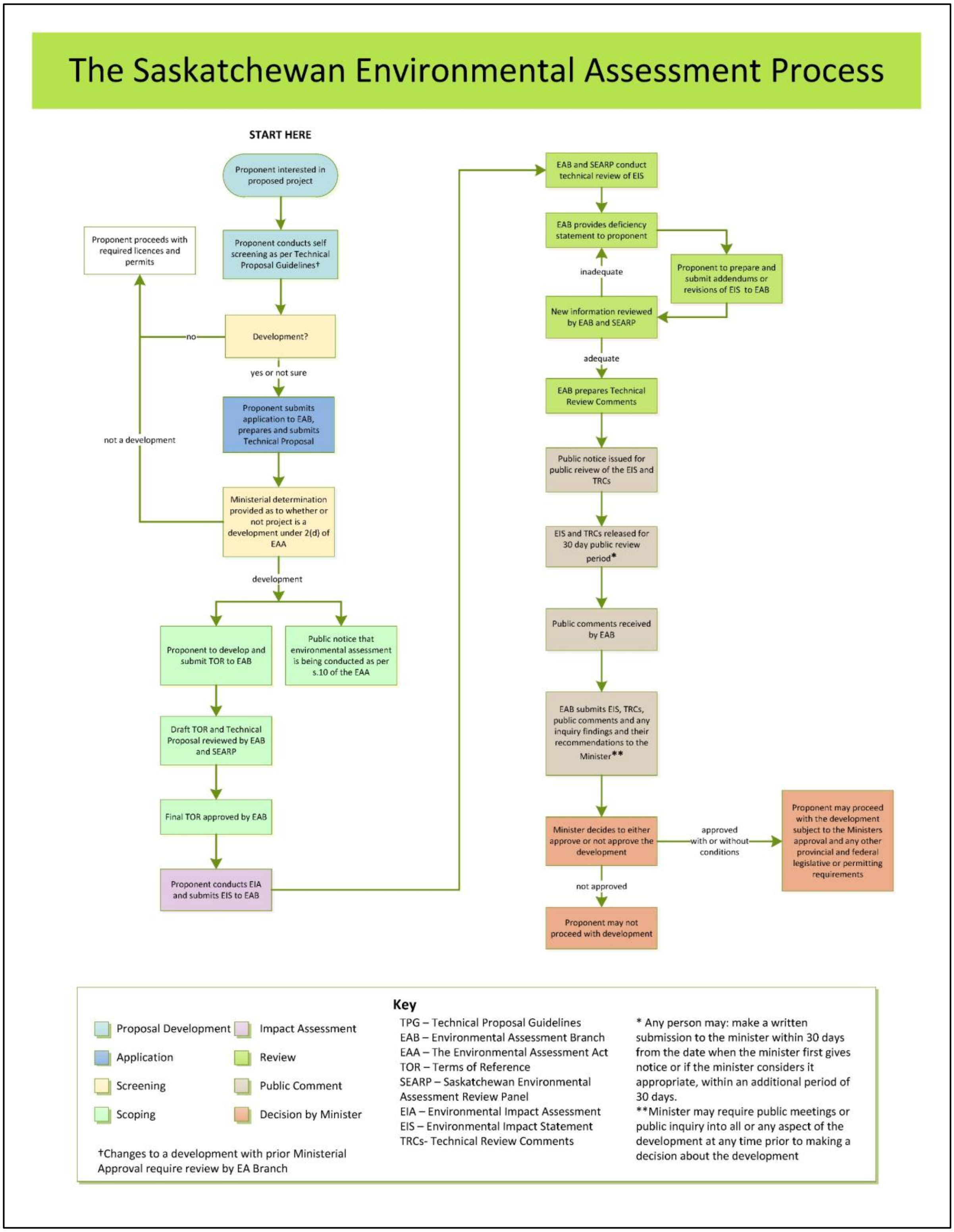


Figure 20-1: The Saskatchewan Environmental Assessment Process (Gov't of Sask., 2026)

In 2019, the Government of Canada enacted the Impact Assessment Act (IAA) outlining the new federal assessment requirements for projects listed as a Designated Activity within the Physical Activities Regulations. The Project does not currently fulfil the criteria to be a designated project under the IAA, therefore, a federal EA is not anticipated. A list of potential permits, approvals, and/or authorizations for the Project is presented in Table 20-2.

Table 20-2: Potential Permits or Approvals for the Project

Permit or Approval	Responsible Agency or Department
Mineral Surface Lease Agreement, including a human resources development agreement	Saskatchewan Ministry of Government Relations, Saskatchewan Ministry of Environment, and Ministry of Immigration and Career Training
Approval to Construct a Pollutant Control Facility	Saskatchewan Ministry of Environment
Approval to Operate a Pollutant Control Facility	Saskatchewan Ministry of Environment
Forest Product Permit	Saskatchewan Ministry of Environment
Miscellaneous Use Permit	Saskatchewan Ministry of Environment
Industrial Air Source Environmental Protection Plan	Saskatchewan Ministry of Environment
Approval to Construct and Operate an Industrial Effluent Works	Saskatchewan Ministry of Environment
Approval to Construct, Upgrade, Operate, or Decommission a Storage Facility (Hazardous Materials and Waste Dangerous Goods)	Saskatchewan Ministry of Environment
Approval to Decommission & Reclaim a Pollutant Control Facilities	Saskatchewan Ministry of Environment
Approval to Release from Decommissioning and Reclamation Requirements	Saskatchewan Ministry of Environment
Provincial Acceptance of Decommissioned and Reclaimed Site into Institutional Control Program	Saskatchewan Ministry of Environment
Wildfire Preparedness Plan	Saskatchewan Ministry of Environment
Approval to Construct and Operate Waterworks (for potable water and/or sewage)	Saskatchewan Water Security Agency
Approval to Construct and Operate Drainage Works	Saskatchewan Water Security Agency
Approval to Construct and Operate Sewage Works	Saskatchewan Water Security Agency
Aquatic Habitat Protection Permit	Saskatchewan Water Security Agency
Ground Water Investigation Permit	Saskatchewan Water Security Agency
Water Rights License	Saskatchewan Water Security Agency
License to Store, Manufacture, or Handle Explosives	Natural Resources Canada

20.4 Indigenous Nations and Community Engagement

Indigenous Nations and Rights Holders

The Project is located within Treaty 10 territory in northern Saskatchewan, in an area that intersects with the traditional lands, asserted rights, and interests of the English River First Nation (with its principal community at Patuanak), the Lac La Ronge Indian Band, and the Kineepik Métis Local of Pinehouse (Métis Nation – Saskatchewan). These Indigenous rights holders have constitutionally protected treaty and Aboriginal rights and continue to exercise traditional land use activities, including hunting, fishing, tapping, and gathering, within the broader Project area.

Abasca has been actively communicating with each of these communities to support meaningful and ongoing engagement throughout the exploration phase of the Project. Engagement to date has included community visits, meetings with First Nation and Métis leadership, and information sharing on the Project's exploration activities. In support of these relationships, Abasca has executed an Exploration Agreement with the English River First Nation in May 2025 that establishes a framework for information sharing, consultation, capacity building & community investment, and access to employment and business opportunities, and that acknowledges the Nation's interests within its traditional use of land. Engagement with the Lac La Ronge Indian Band and the Kineepik Métis Local is ongoing, with discussions to date being positive and focused on building constructive, long-term relationships. Abasca is committed to supporting Saskatchewan's northern communities, including La Ronge. As part of this commitment, the Company has previously provided a geologist to review the geology curriculum for the Mining Engineering Technician Program at Northlands College, as well as geological support to assist the Saskatchewan Precambrian Core Laboratory - both of which are key supports of the annual Core Days Conferences.

As the Project advances, Abasca intends to formalize its approach through an Indigenous engagement framework developed in accordance with applicable provincial and federal Crown consultation guidance. Engagement activities will be coordinated with the scoping and execution of the environmental baseline data collection program, so that Indigenous Knowledge and Traditional Land Use information shared by participating communities can be considered in baseline study design, project planning, and the development of mitigation measures.

Local Government and Public Engagement

Initial outreach has been undertaken with the Northern Village of Pinehouse and other local interest holders along the access corridor. Key interests identified to date include:

- Local economic development and employment opportunities
- Training and skills development for residents of nearby communities
- Use of provincial Highway 914 and associated traffic impacts
- Traditional land use, hunting, fishing, and trapping activities in the area

As the Project progresses and additional information becomes available, future engagement will be expanded to include additional local stakeholders, land users, and community organizations.

Social Considerations

The Project has the potential to provide meaningful employment, training, and business opportunities for residents of nearby Indigenous and non-Indigenous communities in a remote region of northern Saskatchewan. Early planning includes local hiring and procurement strategies, and discussions with rights holders to date have been positive and supportive of these objectives. No known non-Indigenous land users have raised concerns to date; this will be confirmed through expanded engagement as the Project advances.

Beyond direct employment and procurement, the Project is expected to generate broader economic benefits for the Province of Saskatchewan over its operating life through Crown royalties, corporate and payroll taxes, and ancillary spending across the regional supply chain. These contributions would support provincial revenues that fund public services in northern Saskatchewan and across the Province, while strengthening the long-term economic base of the surrounding region.

The Project aligns with both the Saskatchewan Critical Minerals Strategy and the Canadian Critical Minerals Strategy, each of which identifies natural graphite as a strategic mineral for the clean-energy transition and for the development of secure North American battery and advanced-materials supply chains. The Province of Saskatchewan has signalled strong policy and program support for critical mineral exploration and development through measures such as the Saskatchewan Mineral Exploration Tax Credit, the Targeted Mineral Exploration Incentive, and the Saskatchewan Critical Minerals Innovation Incentive, each of which is relevant to projects of this nature. At the federal level, natural graphite is included on the Government of Canada's Critical Minerals List, and the Project is positioned to benefit from federal funding, and policy supports advanced under the Canadian Critical Minerals Strategy. Abasca Resources continues to engage with both levels of government, and the Project has received supportive engagement from the Province of Saskatchewan and the Government of Canada in connection with their respective critical minerals priorities.

20.5 Conceptual Mine Closure and Reclamation Plan

The Conceptual Closure and Reclamation Plan for the project includes the following activities:

- Decommissioning and removal of the surface facilities from the project site, including all buildings and equipment, such as the processing plant, power plant, camp, maintenance shops, warehouse, storage, tanks, pumps, assay laboratory, power distribution, mine ventilation, explosive magazines, and utilities. Concrete slabs and footings will be broken and placed appropriately. Certain equipment packages, such as the water treatment plant and power supply, will be retained on site to provide post-closure water treatment service.
- Collection and removal of contaminated soils.
- Stockpiles and earthworks, including WRTCF, building pads, laydown areas and certain internal roads, will be re-contoured for geotechnical stability, covered and capped with appropriate fill, top soiled and revegetated. Engineered cover systems will be applied where PAG material may be present for ARD mitigation.
- Pits and wells will be backfilled with appropriate NAG fill material, top soiled and revegetated.
- Water will be treated to meet discharge criteria prior to discharge, if treatment is required.
- Appropriate seepage collection and piping systems will be installed to collect and transport any seepage to the water treatment plant.

The estimated Closure and Reclamation costs are discussed in Section 22.0.

20.6 Environmental Management and Monitoring Plans

During the Feasibility Study (FS) and Environmental Assessment (EA) permitting phases, several environmental management and monitoring plans will be established for the project. These plans will complement the engineered designs required for waste rock management, tailings management, and the storage and conveyance of the mineralized material (see Section 18.0 of this report).

21.0 CAPITAL AND OPERATING COSTS

21.1 Summary

The capital and operating costs for the Project have been estimated and are summarized in **Table 21-1**.

Table 21-1: Summary of Capital and Operating Costs

Description	Total Capital Cost (US$ M)	Average Unit Operating Cost (US$/t processed)
Initial Capital Costs	216.5	-
Sustaining Capital and Closure Costs	91.0	-
Life of Mine (LOM) On-site Operating Costs	-	46.34

All costs are reflected in 2026 Q2 US Dollars unless otherwise specified. The expected accuracy range of the cost estimates is within ±35%. Where applicable, costs in this report have been converted from Canadian Dollars to US Dollars using a currency exchange rate of CAD1.00:USD0.72.

21.2 Capital Cost Estimate

The total estimated initial capital cost for the design, construction, installation, and commissioning of the Project is US$216.5 M. A summary breakdown of the initial capital cost is provided in Table 21-2. This total includes all direct costs, indirect costs, Owner's costs, and contingency.

Table 21-2: Initial Capital Cost Summary

Description	Initial Capital Cost (US$ M)
Direct Costs	
Mining Pre-Production Capex	16.6
Mining Equipment Fleet Down Payment	5.9
Site Preparation & Surface Mobile Equipment	7.7
Processing Plant	73.2
Waste Management	5.2
Utilities	15.7
Ancillary Service Buildings	15.8
Offsite Infrastructure	10.1
Subtotal – Direct Costs	**150.2**
Indirect Costs	
Project Indirect Costs	39.6
Owner's Costs	2.4

Description	Initial Capital Cost (US$ M)
Contingencies	24.3
Total	**216.5**

21.2.1 Class of Estimate

This Class 4 cost estimate has been prepared in accordance with the standards of AACE International. There is no deviation from AACE International's recommended practices in the preparation of this estimate. The expected accuracy range of this cost estimate is within ±35%.

21.2.2 Estimate Base Date and Validity Period

This cost estimate was prepared with a base date of Q2 2026 and does not include any escalation beyond this date. For any vendor quotations used for this PEA cost estimate, they were obtained in Q2 2026 and have a validity period of 30 calendar days or less.

21.2.3 Estimate Approach

Currency and Foreign Exchange

The capital cost estimate uses US Dollars as the base currency. Where applicable, quotations received from vendors were converted to US Dollars using a currency exchange rate of CAD1.00:USD0.72. There are no provisions for foreign exchange fluctuations.

Duties and Taxes

Duties and taxes are not included in the estimate.

Measurement System

The International System of Units (SI) is used in this estimate.

Work Breakdown Structure

The estimate is organized according to the following hierarchical work breakdown structure (WBS):

- 1000 Mining
- 2000 Site Preparation/Surface General
- 3000 Processing Plant
- 4000 Waste Management
- 5000 Utilities
- 6000 Ancillary Service Buildings
- 7000 Offsite Infrastructure
- 8000 Project Indirects

- 9100 Owner's Costs
- 9200 Contingency

21.2.4 Elements of Cost

This capital cost estimate consists of the four main parts: direct costs, indirect costs, Owner's costs, and contingency.

Direct Costs

AACE International defines direct costs as:

...costs of completing work that are directly attributable to its performance and are necessary for its completion. In construction, it is considered to be the cost of installed equipment, material, labour and supervision directly or immediately involved in the physical construction of the permanent facility.

Examples of direct costs include mining equipment, processing equipment and permanent buildings.

The total direct cost for the Project is estimated to be US$150.2 M.

Indirect Costs

AACE International defines indirect costs as:

...costs not directly attributable to the completion of an activity, which are typically allocated or spread across all activities on a predetermined basis. In construction, (field) indirects are costs which do not become a final part of the installation, but which are required for the orderly completion of the installation and may include, but are not limited to, field administration, direct supervision, capital tools, start-up costs, contractor's fees, insurance, taxes, etc.

The total indirect cost for the Project is estimated to be US$39.6 M.

Owner's Costs

Owner's costs are costs provided by the Owner to support and execute the Project.

The Project execution strategy, in particular for construction management, involves the Owner working with an engineering, procurement, and construction management (EPCM) organization and supervising the general contractor(s). The Owner's costs include home office staffing, home office travel, home office general expenses, field staffing, field travel, general field expenses, community relations, and Owner's contingency.

The total Owner's cost allowance for the Project is estimated to be US$2.4 M.

Contingency

Tetra Tech estimated a contingency for each activity or discipline based on the level of engineering effort as well as experience on past projects.

The total contingency allowance for the Project is US$24.3 M.

21.2.5 Capital Cost Exclusions

The following items have been excluded from this capital cost estimate:

- Working or deferred capital

- Financing costs
- Taxes and duties
- Land acquisition
- Currency fluctuations
- Lost time due to severe weather conditions
- Lost time due to force majeure
- Additional costs for accelerated or decelerated deliveries of equipment, materials, or services resultant from a change in project schedule
- Warehouse inventories, other than those supplied in initial fills, capital spares, or commissioning spares
- Any project sunk costs (studies, exploration programs, etc.)
- Mine reclamation costs (included in the financial model)
- Mine closure costs (included in the financial model)
- Escalation costs beyond Q2 2026
- Any other unforeseeable events or anomalies.

21.3 Operating Cost Estimate

21.3.1 Summary

On average, the LOM on-site operating costs for the Project were estimated to be $46.34/t of material processed. The operating costs are defined as the direct operating costs, including mining, processing, site servicing, and G&A costs, including related freight costs. Table 21-3 and Figure 21-1 show the cost breakdown for various areas.

The cost estimates in this section are based upon the consumable prices and labour salaries/wages in Q2 2026 or based on the information from Tetra Tech's database. Where applicable, the currency exchange rate of CAD1.00:US0.72 is used. The expected accuracy range of the operating cost estimate is within ±35%.

Table 21-3: LOM Average Operating Cost Summary

Function	Operating Cost* (US$/t processed)
Mining Cost	20.42
Processing Cost	12.81
G&A and Site Services	12.62
Interest (Mining Fleet Lease)	0.49
Total Operating Cost	**46.34**

Note: *LOM average operating, which is slightly different from the unit cost at the nominal processing rate of 2,750 t/d.

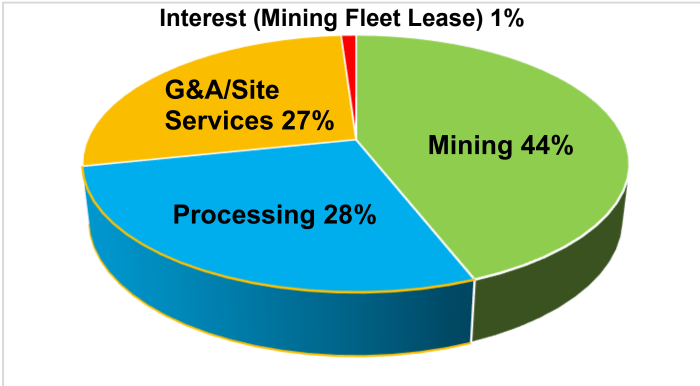


Figure 21-1: LOM Average Operating Cost Distribution by Operation Unit

21.3.2 Mining Operating Cost

The estimated mining operating cost is based on owner-operated mining operations. Table 21-4 outlines the Life of Mine (LOM) average cost per tonne mined by mining activity.

Table 21-4: Life of Mine Average Mining Operating Cost

Activity	LOM Average Unit Cost, US$/t Mined	LOM Average Unit Cost, US/t Processed
Blasting	0.28	1.87
Primary Equipment	1.21	8.20
Secondary Equipment	0.19	1.27
Hourly Labour	1.10	7.47
Salaried Labour	0.24	1.61
Total	**3.02**	**20.42**

21.3.3 Processing Operating Cost

The LOM average annual operating cost for the nominal processing rate of 2,750 t/d is estimated to be US$12.0 million per year, or US$12.81/t processed. This estimate includes:

- Staffing and salary/wage level estimates, based on the 2026 mining industry labour rates in Western Canada.
- Power consumption estimates are based on power draw estimates from preliminary equipment load list.
- Unit power cost estimate of US$0.0451/kWh based on SaskPower's rate schedule.
- Operating and maintenance supply cost estimate, based on approximately 5% of major equipment capital costs, benchmarked against comparable operations in Western Canada and worldwide.

Table 21-5 summarizes the estimated processing costs at the nominal processing rate. Figure 21-2 shows the average operating cost distribution by area at the nominal processing rate. No taxes have been accounted for in the estimates, unless specified.

Table 21-5: Summary of Processing Operating Costs

Description	Humanpower	Annual Cost (US$ '000)	Unit Cost (US$/t processed)
Humanpower	78	5,548	5.53
Consumables		3,091	3.08
Maintenance Supplies		1,223	1.22
Operating Supplies		938	0.93
Power Supply		1,284	1.28
Others		604	0.60
Total		**12,690**	**12.64**

Note: Dollar figures are either rounded to the nearest thousand dollars or the nearest cent. Sums may not add up due to rounding.

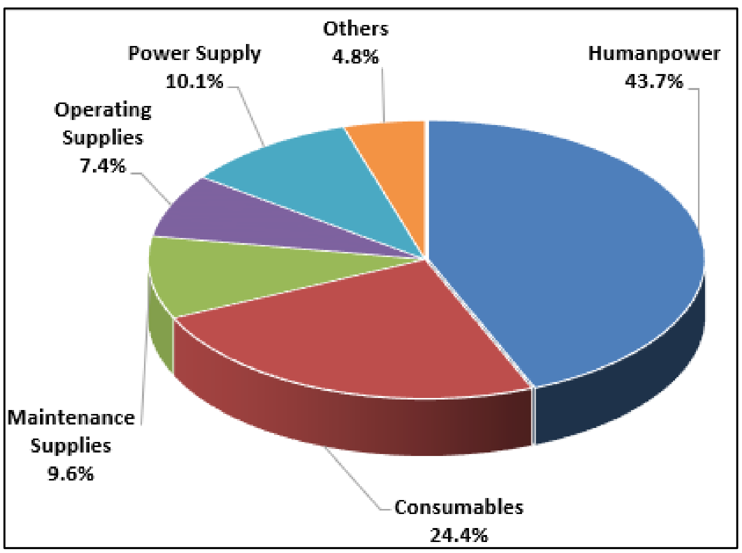


Figure 21-2: Average Processing Operating Cost Distribution by Area

Power, consumable and supply costs constitute approximately 52% of total processing costs. The cost estimates are detailed in the following sections.

Processing Operating Cost

Labour Cost

Total processing labour costs at the nominal processing rate of 2,750 t/d are estimated to be US$5.55 M per year, or US$5.53/t processed. Salary estimates are based on the 2026 mining industry labour rates in western Canada. The processing plant will be staffed with 78 personnel, including 9 in general supervisory and technical services, 34 in operational roles, 28 in maintenance and 7 in metallurgical and assay laboratory roles.

Loaded salary includes base salary and burden, which on average is estimated to be approximately 30% of the base payment. The burden includes holiday and vacation payments, overtime payments, pension plan, Medical Services

Plan, Canada Pension Plan, Employment Insurance, Workers' Compensation Board insurance and tool allowance costs.

Power Cost

The average annual processing power consumption, based on the nominal processing rate of 2,750 t/d, is estimated to be 28.5 MWh/a. At an average power unit cost of US$0.0451/kWh, the annual power cost is estimated to be US$1.28 M, or US$1.28/t processed.

Maintenance and Operating Supplies Cost

Maintenance and operating supplies are estimated to cost US$2.16 M per year, or US$2.15/t of processed. Maintenance supplies are estimated to cost US$1.22 M per year, or US $1.22/t of processed. Operating supplies are estimated to be US$0.94 M per year, or US$0.93/t of processed.

Major Consumable Costs

Major consumable costs are estimated to be US$3.09 M per year at the nominal processing rate of 2,750 t/d, or US $3.08/t of processed.

The estimates of major consumable costs are based on the following:

- Consumption rates for liners are based on the data provided by the potential crusher and ball mill suppliers, or in-house data.
- Grinding ball consumptions are estimated based the test results and in-house experience and unit ball prices are based on in-house database.
- Consumption rates for reagents are based on the data provided in the metallurgical test results.
- Maintenance supplies consumption is based on experience or similar projects.
- Fuel consumptions for mobile equipment, including tailings cake loaders, are based on in-house experience and the diesel price projected for this project.
- Other operating supplies include filter cloths and concentrate bags.
- No taxes have been accounted in the estimates, unless specified.

Tailings cake transport and cake placement and compaction costs are included in the mining operating costs.

21.3.4 General and Administrative

At the nominal processing rate, total on-site and off-site G&A expenses are estimated at approximately US$11.49 million per year, or US $11.45/t processed. The costs include:

- Personnel – supervisors and staffing in accounting, purchasing, environmental departments, and other G&A departments.
- G&A expenses including insurance, administrative supplies, medical services, legal services, human resources related expenses, travelling, electricity power demand charge, accommodation/camp costs, air and bus crew transportation, and external assay/testing.

At the nominal process rate, the humanpower cost and general expenses are estimated to be approximately US $2.70/t processed and US $8.75/t processed, respectively. The major costs are accommodation and crew air and bus transportation, estimated at approximately US $6.02 million per year.

21.3.5 Surface Services

The LOM average site service cost is estimated at US $1.17/t processed or approximately US $1.17 million per year. The estimate includes:

- Surface mobile equipment and light vehicle operations
- Potable water and waste management
- General maintenance including yards, roads, fences, and building maintenance
- Building heating
- General site water treatment (allowance), the cost estimate is preliminary and should be reviewed and revised during the next phase of the study and after the completion of baseline data collection, hydrogeological drilling and water quality modelling.

The costs for the site service operators are included in the process department.

22.0 ECONOMIC ANALYSIS

The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the results of the PEA will be realized. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

22.1 Introduction

The economic analysis of the Project has been derived from the inputs described in previous chapters. The economic analysis has been performed on a 100% basis using Q2 2026 US dollars, and its unlevered post-tax free cash flow (FCF) has been discounted at a rate of 8% per annum. Capital, onsite and off-site operating, sustaining and closure costs and Federal and Provincial income taxes are included in the financial analysis. The economic analysis results are presented in US dollars. As specified in the Marketing Section (Section 19.0 of this report), the base case graphite concentrate price applied in the economic analysis is US$1,450/t delivered to Saskatoon and the base case foreign exchange rate is CAD1.00:USD 0.72.

For the 19-year mine life and 17.9 Mt processing plant feed tonnage, LOM average feed grade of 7.27% Cg, the following investment returns and select financial metrics were assessed:

Table 22-1: Summary of the Before-Tax and After-Tax Economic Analysis Results

Graphite Concentrate Price (US$/t)	US$1,450/t
Exchange rate CAD:USD	0.72
Pre-Tax Net Present Value (NPV) @8%	US$161 M
Pre-Tax Internal Rate of Return (IRR)	16.6%
Pre-Tax Payback Period	6.2 Years
After-Tax Net Present Value (NPV) @ 8%	US$130 M
After-Tax Internal Rate of Return (IRR)	16.7%
After-Tax Payback Period*	4.7 Years

* After-tax payback period is shorter due tax refunds that are estimated by Abasca's tax expert to be received during the earlier project years.

22.2 Inputs/Assumptions to the Preliminary Financial Analysis

The inputs to the preliminary economic assessment include:

- Mine production schedule, including dilution and waste rock handling.
- Process plant feed recovery rates to develop the annual recovered concentrate tonnages based on metallurgical test work results.
- Initial capital expenditures, including required construction and development before commercial production, including mining development and overall project construction expenditures as specified in Section 21.0.

- Sustaining capital costs on a year-by-year basis over the LOM, including the US$27 M mine closure and reclamation costs.
- OPEX, including the costs for mining, processing, general administration, overall site services, and site water management.
- Offsite expenditures, such as concentrate transportation and insurance costs (the offsite charges are detailed in Section 22.3).
- Concentrate price and the foreign exchange rate, as specified in Section 19.0.
- Net proceeds interests and royalties detailed in Section 22.4.

Annual at-mine revenues were determined by deducting the off-site charges from gross revenue. The financial model also includes working capital that will be recovered during the LOM.

Concentrate prices and the foreign exchange rate are assumed to be flat throughout the project life.

All the cash flows in the financial model were based on Q2 2026 US dollars without any adjustment for inflation.

22.3 Offsite Charges

The offsite charges, including the concentrate transportation costs, were specified in Section 19. The assumed Project offsite charges are shown inTable 22-2 (details are in Section 19.0).

Table 22-2Table 22-2 (details are in Section 19.0).

Table 22-2: Offsite Charges and Assumptions

Item	Unit	Value
Payability	%	100.0%
Insurance	% of Product Values Produced	0.125%
Concentrate Transport Loss	% of Product Values Produced	0.15%
Representation	US$/t Dry Concentrate	$2.00
Royalty	% of Product Values Produced	0.00%
Concentrate Transport		
Concentrate Moisture	%	~1%
Site to Saskatoon, Sask.	US$/t Wet Concentrate	43.20

22.4 Royalties and Other Agreements

As advised by Abasca management, Abasca holds a 100% interest in the Project. There are no additional royalties, back-in rights, or encumbrances on the project or potential production, other than the standard royalties due to the Government of Saskatchewan. Currently, there are no concentrate sale agreements or contracts established for the Project.

22.5 Assumptions on Taxes

The Project has been evaluated on an after-tax basis to provide an approximate value of the potential economics. The tax model was prepared by an independent tax consultant. As of the effective date of this report, the project was assumed to be subject to the following tax regime:

22.5.1 Income Taxes

- The Canadian Corporate Income Tax system consists of the federal income tax (15%) and the provincial income tax (12%)

22.5.2 Saskatchewan Crown Mineral Royalty (SCMR)

- Tax Holiday
  - In the first 10 years after commercial production, the net profit is deemed to be zero
- Royalty Rate:
  - Tier 1 – 5% of net profit up to 1,000,000 cumulative metric tonnes of all minerals
  - Tier 2 – 10% of net profit: on volumes above the 1,000,000 threshold
  - Negative amounts in the calculation of SCMR are treated as zero

22.5.3 Clean Technology Manufacturing Investment Tax Credit

The tax model also includes the application of the Clean Technology Manufacturing Investment Tax Credit (CTMITC), which is a 30% refundable income tax ITC on qualifying capital costs. It is assumed that, as with other refundable ITCs, the CRA will follow a protocol of auditing claims before paying any refund, and that refunds will be paid 12-18 months following the due date for filing the claim (i.e. in the 2nd taxation year following the taxation year in which the claim arises). The amount of the ITC is deducted from the undepreciated capital cost of the assets in the year following the year of claim. It is assumed that all depreciable capital costs associated with the project will qualify for the CTMITC.

22.6 Financial Model Summary

The production schedule has been incorporated into the 100% equity, pre-tax financial model to develop annual production based on tonnage processed, head grades, and recoveries.

Graphite concentrate revenues are derived from the concentrate sales delivered to Saskatoon. Operating costs for mining, processing, site services, G&A, as well as off-site charges 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 infrastructure expansions.

Initial and sustaining capital costs (including closure and reclamation) applied in the economic analysis are US$216.5 M and US$63.8 M, respectively. The pre-production construction period is expected to be two years. NPV and IRR are estimated at the start of this two-year period.

Working capital is included in the financial analysis and varies from year to year. The working capital is recovered at the end of the mine life.

Table 22-3 shows the key results of the financial model.

Table 22-3: Financial Model Key Results

Description	Unit	Value
Product Price		
Graphite Concentrate Price (Average)	**US$/t**	**1,450**
Production		
Mine Life	Year	19
Mill Feed Tonnage, Life of Mine (LOM)	Mt	17.9
Mill Feed Grade, Life of Mine (LOM)	% Cg	7.27
Concentrate Grade, Life-of-mine Average	% Cg	95.0
Graphite Recovery to Concentrate	%	92.3
Graphite Produced, Life of Mine (LOM)	dry tonnes	1,263,000
Annual	dry tonnes	66,500
Operating Costs, Life of Mine (LOM)	US $M	828
Unit	US$/t proc.	46.34
Operational Revenue	US $M	940
Capital Costs		
Initial Capex	US $M	216
Sustaining	US $M	64
Closure & Reclamation	US $M	27
Total Capital Costs	US $M	307
Economic Results		
Discount Rate	%	8%
Pre-Tax Unlevered Free Cash Flow (UCF)	**US $M**	**633**
Pre-Tax Net Present Value (NPV) @ 8%	**US $M**	**161**
Pre-Tax Internal Rate of Return (IRR)	**%**	**16.6%**
Pre-Tax Simple Payback Period	**Year**	**6.2**

Description	Unit	Value
After-Tax Unlevered Free Cash Flow (UCF)	**US $M**	**477**
After-Tax Net Present Value (NPV) @ 8%	**US $M**	**130**
After-Tax Internal Rate of Return IRR	**%**	**16.7%**
After-Tax Simple Payback Period	**Year**	**4.7**

The project's post-tax net cash flows summary is shown in Table 22-1. The cash flows decline in years 18 and 19 since the process plant feed is mainly from low grade stockpile materials. The negative cash flows in years 19 to 23 are due to closure and reclamation expenditures.

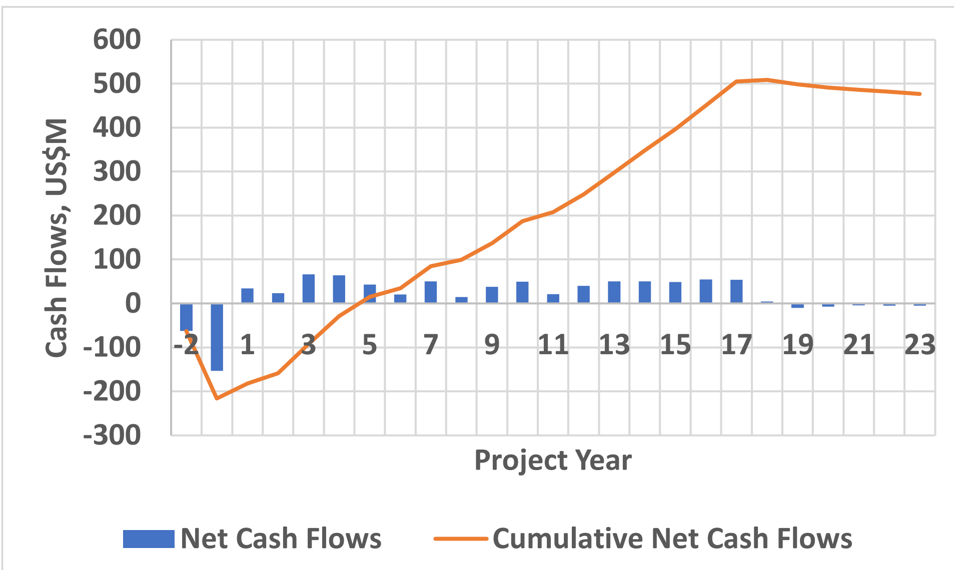


Figure 22-1: Post-Tax Annual and Cumulative FCFs

Federal, Provincial Tax, and Provincial Resource Tax payables based on the PEA financial model are calculated and shown below in Table 22-4.

Table 22-4: Project Tax Payables Summary

Tax	Value, US$ M
Total Federal and Provincial Corporate Income Tax Payable	120.52
Provincial Resource Tax Payable	35.71
Total	**156.23**

22.7 Sensitivity Analysis

Sensitivity analysis for after-tax financial parameters for both NPV at a discount rate of 8%, IRR and payback period was conducted and is presented in Figure 22-2 to Figure 22-4. The financial parameters include:

- Product price
- Exchange rate
- Operating costs
- Capital costs

The Project NPV is most sensitive to product price, followed by exchange rate, operating costs, and capital costs. IRR is most sensitive to product price and exchange rate, followed by capital costs and operating costs. Payback period is most sensitive to product price, followed by the exchange rate, operating costs and capital costs.

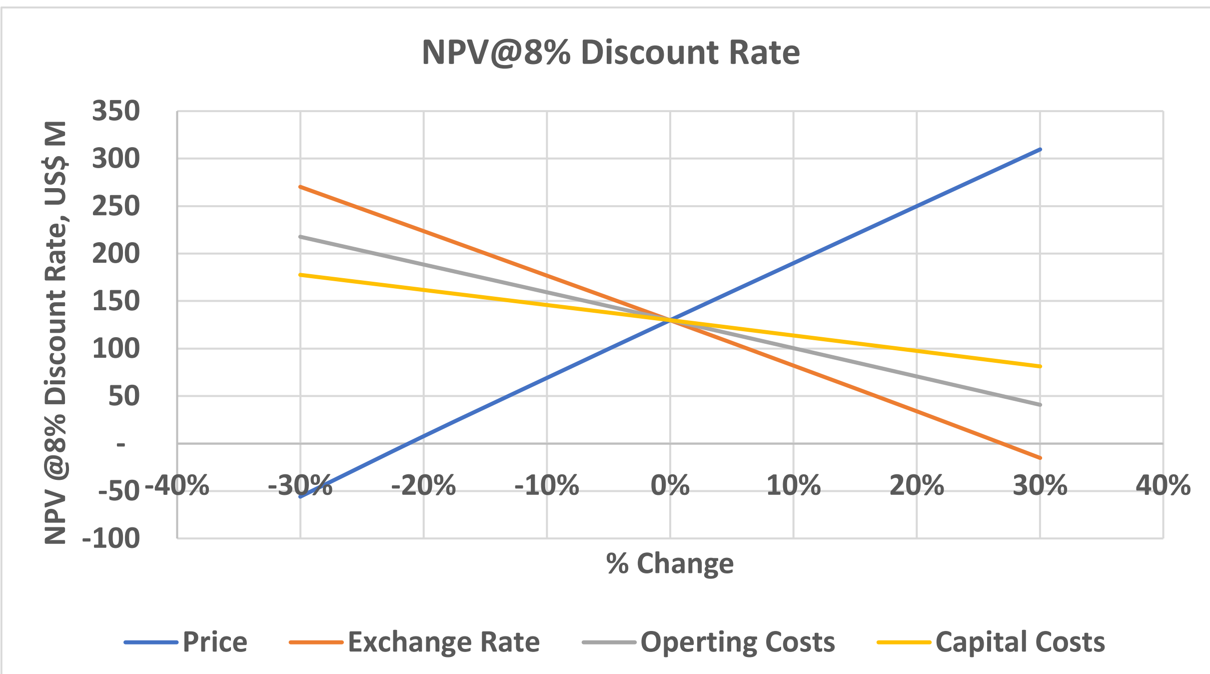


Figure 22-2: After-tax NPV Sensitivity (+/-30%)

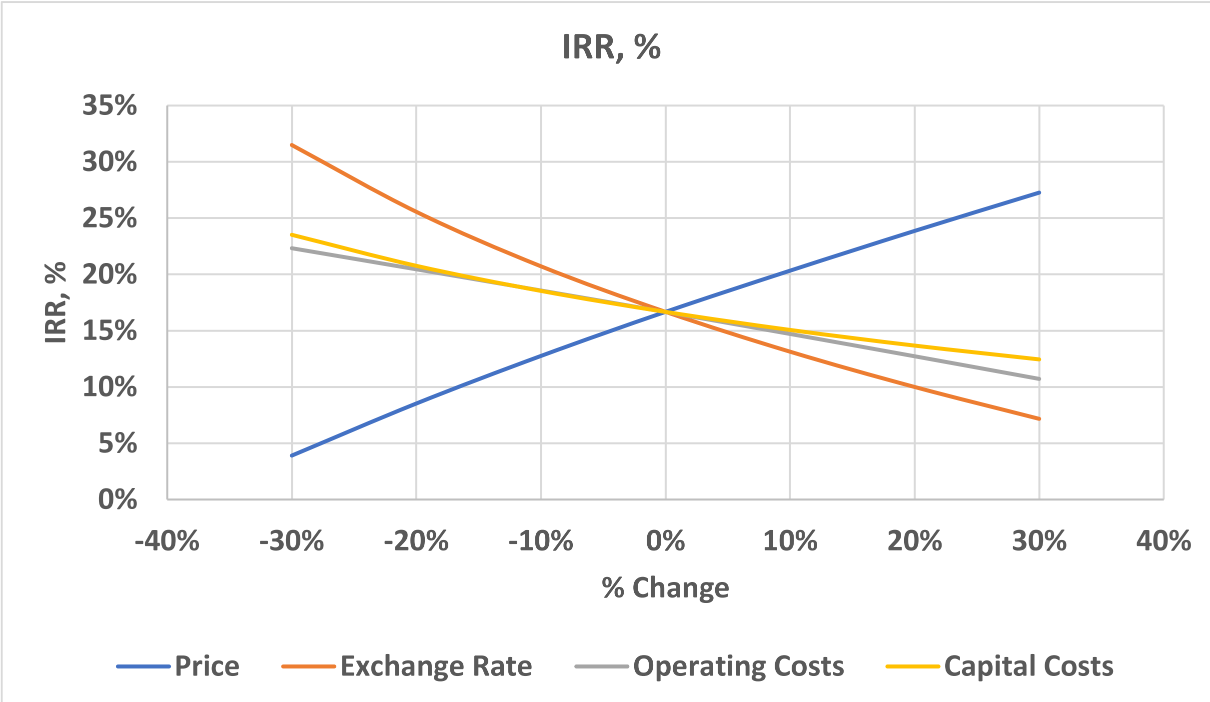


Figure 22-3: After-tax IRR Sensitivity (+/-30%)

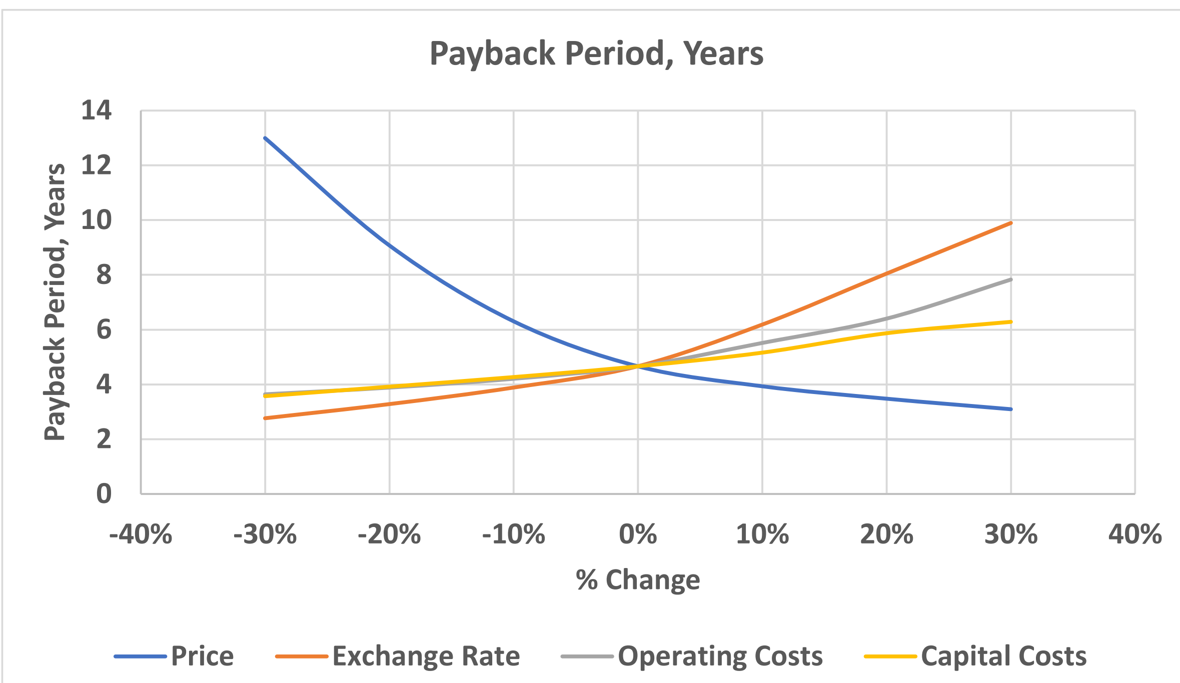


Figure 22-4: After-tax Payback Period Sensitivity (+/-30%)

23.0 ADJACENT PROPERTIES

The qualified person has not verified the information of the adjacent properties, and that the information of the adjacent properties is not necessarily indicative of the mineralization on the property that is the subject of the technical report.

There are several adjacent properties to the Project (Figure 23-1) that conduct greenfields exploration programs, although there is only one notable property with extensive work and operations being conducted, which is Cameco Corporation's Key Lake Operation (Key Lake), located toward the northeast of the Project. Information reported here is referenced from Cameco Corporation's website (https://www.cameco.com/businesses/uranium-operations/canada/mcarthur-river-key-lake and https://www.cameconorth.com). Key Lake is a past producing open pit mine that produced 535 million pounds of uranium concentrate since 1983. Although the mine is not producing anymore, the mill is used to process the ore from McArthur River, the world's largest, high-grade uranium mine. Since the year 2000, more than 325 million pounds of uranium have been mined at McArthur River and milled at Key Lake. In October of 2023, the Canadian Nuclear Safety Commission (CNSC) granted a 20-year renewal licence for both McArthur River and the Key Lake facilities. They are now expected to continue to operate until October 2043.

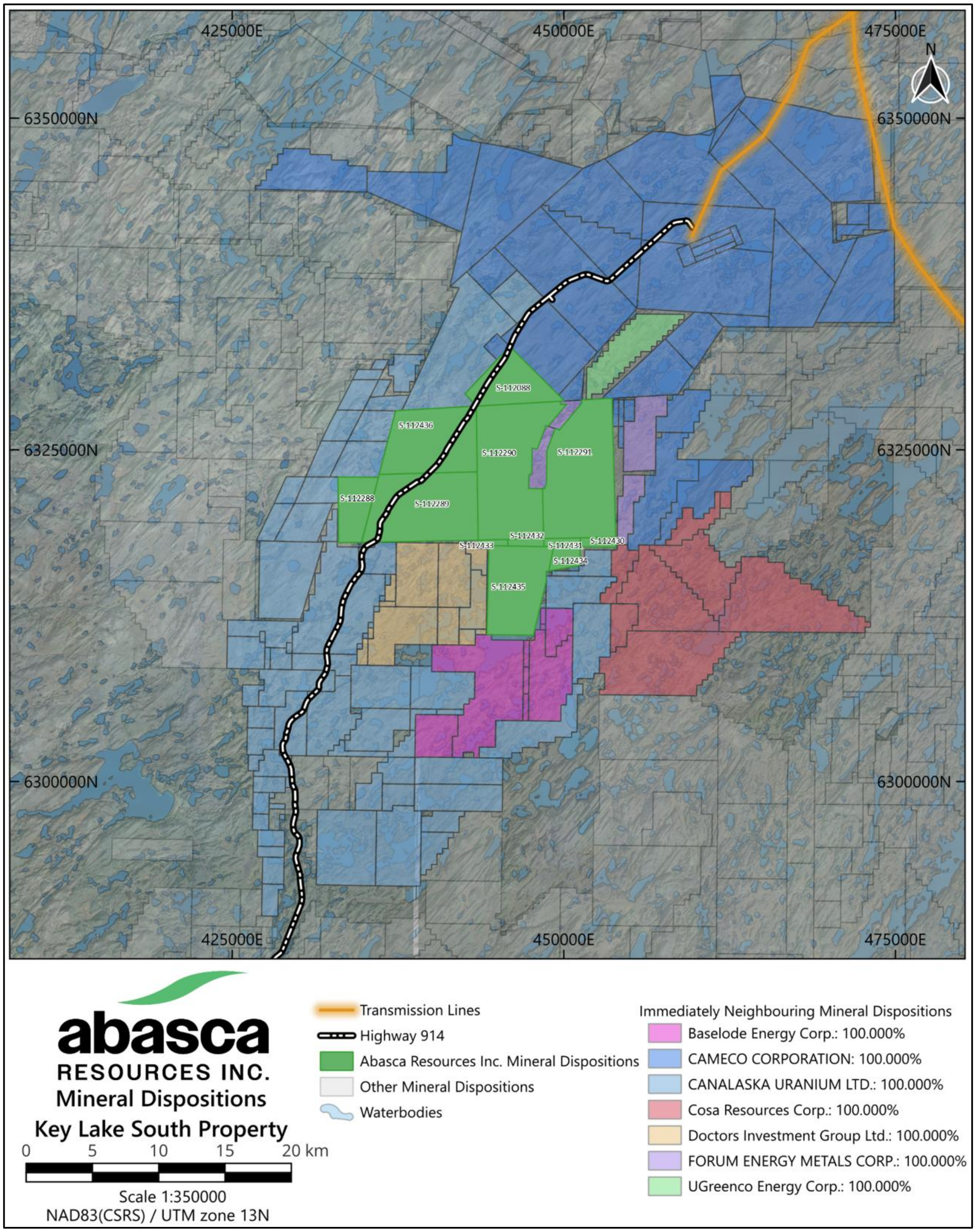


Figure 23-1: Map of the Adjacent Properties to the Project

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24.0 OTHER RELEVANT DATA AND INFORMATION

A preliminary project execution plan has been prepared for the Project, which describes a logical sequence by which the Project may advance from its current status to the start of mine production.

Environmental studies and permitting (discussed in Section 20.0) will accompany and enable the engineering and construction work described in this section.

Abasca's future activities at the Project site will demonstrate full compliance with all applicable regulations, maintenance of a safe and healthy working environment, care and concern for the natural environment, and responsiveness to the needs and ambitions of the Indigenous and northern communities. Abasca will maintain and enhance these standards as the Project develops towards mine production. The planned construction and commissioning timeline of the Project is summarized in Figure 24-1.

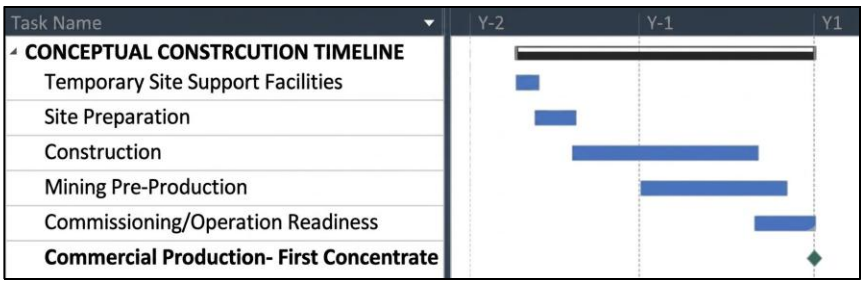

Figure 24-1: Conceptual Project Construction and Commissioning Timeline

The preliminary project execution schedule (the Schedule) was developed to provide a high-level overview of all activities required to complete the project.

Upon receipt of construction and operating permits, the project is expected to take approximately 1.5 years to complete, from the time board approval is received, through construction, commissioning and commencement of production.

The critical path of the project development is composed of activities related to:

1. Feasibility Study and Detailed Engineering
2. Permitting and Licensing
3. Detailed Engineering
4. Procurement of Long Lead Items
5. Construction
6. Commissioning

Additional activities such as trade-off studies, additional drilling programs, metallurgical testing, as well as major equipment fabrication can proceed in parallel to the critical path activities.

25.0 INTERPRETATION AND CONCLUSIONS

Mineral Resource Estimate

The QP's related interpretation and conclusions are summarized below.

- The Loki Deposit is a broad graphite bearing shear-zone ranging in thickness along an orientation of 127° azimuth and -45° dip to the southwest. The QP created one vein wireframe to constrain the estimate in the predominant orientation of mineralization. The wireframe is approximately 1,700 m long in the strike direction with an upper contact 55 m below surface and extends to 335 m below surface. The thickness of the modelled graphite ranges between 10 and 55 m, with an approximate average of 35 m.
- The 2026 Mineral Resource Estimate has an effective date of April 23, 2026, coincident with the date of the last assay result received from the analytical laboratory.
- The Mineral Resource is composed of Indicated and Inferred Mineral Resources. The Indicated Mineral Resource totals 6.99 million tonnes at an average grade of 8.27 % Cg for a total of 0.58 million tonnes Cg, and the Inferred Mineral Resource totals 15.83 million tonnes at an average grade of 6.93 % Cg for a total of 1.10 million tonnes Cg.
- The resource was reported at a cut-off grade of 2.30 % Cg within a conceptual open pit design. No Mineral Reserves have been estimated at the Project.
- In conjunction with infill drilling, the testing of uranium contamination and graphite flake quality will be important for upgrading portions of the deposit from Inferred to Indicated or Indicated to Measured Mineral Resources.
- The lithology domains were assigned a density based on specific gravity measurements observed within the domains or observations from analogous deposits. A linear regression was used to assign density to blocks with estimated graphite grades.
- In the QP's opinion, the Project exploration data are free of any material or systematic errors, well validated and of sufficient quality for use in this Technical Report.
- The QP identified uranium contamination of graphite mineralization as a potential risk. The QP believes that the likelihood of realizing this risk in a material sense is minimal given that the graphitic shear does not appear to host significant uranium grade, but recognizes the risk remains until confirmed otherwise. Beyond this risk, the QP has not identified any other significant risks or uncertainties that could reasonably be expected to affect the reliability or confidence in the Mineral Resource.
- The QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource Estimate that are not discussed in this Technical Report.

Mining

The Loki Flake Graphite Project is planned to be developed using conventional open-pit mining methods, including drilling, blasting, loading, and hauling with large-scale mining equipment. Material will be drilled and blasted, then loaded into diesel-powered haul trucks using a fleet of hydraulic shovels and front-end wheel loaders.

To identify the optimal pit size and mining sequence, Tetra Tech employs the Lerchs–Grossmann (LG) algorithm to evaluate the net value of individual blocks within the block model. An assumed graphite concentrate price of US$1,450 per tonne has been applied. Life of mine (LOM) operating costs were estimated based on similar projects or operations.

The table and figure below show the PEA mine production plan.

Table 25-1: Mine Production Plan

Production Year	Total Mined (Mt)	Waste (Mt)	Strip Ratio	Mill Feed (Mt)	Grade (%)
-1	7,499,919	7,499,919	-	-	0.00
1	9,999,901	9,149,202	10.8	763,616	8.50
2	9,999,840	9,243,484	12.2	843,640	7.05
3	9,999,980	8,804,283	7.4	1,002,839	7.91
4	9,999,970	9,012,357	9.1	1,001,241	6.64
5	9,999,913	8,643,356	6.4	1,002,531	7.76
6	9,999,803	9,034,431	9.4	1,001,241	6.54
7	9,999,978	8,628,075	6.3	1,001,241	7.92
8	8,999,978	8,348,799	12.8	1,001,241	6.22
9	8,499,832	7,413,899	6.8	1,001,241	7.03
10	7,999,916	6,638,336	4.9	1,001,199	8.03
11	5,999,761	5,056,954	5.4	1,001,241	6.52
12	4,499,942	3,498,543	3.5	1,001,241	6.96
13	3,499,819	2,461,676	2.4	1,001,241	7.58
14	2,499,840	1,532,322	1.6	1,001,241	7.79
15	2,499,899	1,430,300	1.3	1,001,281	7.91
16	2,499,914	1,372,698	1.2	1,002,552	8.13
17	1,816,662	676,773	0.6	1,001,241	8.30
18	-	-	-	1,001,241	5.11
19	-	-	-	240,396	4.04
Total	**126,314,868**	**108,445,407**	**6.1**	**17,871,701**	**7.27**

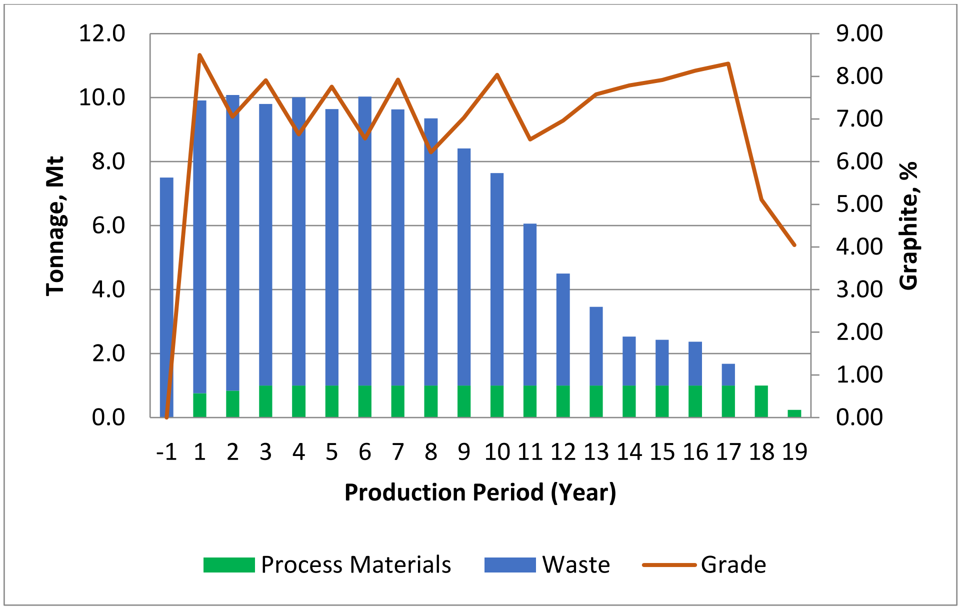

Figure 25-1: KLS Mine Production Plan

The life-of-mine plan comprises 17 years of active mining operations followed by two years of stockpile processing. Mining is conducted at a consistent rate of 10.0 Mt of material annually during the first seven years and slowly diminishing from Y8 to Y17, with a LOM average strip ratio of 6.1:1. The process plant is designed for a nominal throughput of 2,750 tonnes per day (approximately 1.0 million tonnes per year). Average payable graphite production over the 19-year mine life is estimated at approximately 1.23 million dry tonnes per year. Mill throughput is planned at 75% of nameplate capacity in Year 1, 85% in Year 2, and 100% from Year 3 onward, allowing for a controlled start-up and optimization of plant performance. The mill feed grade fluctuates between 6.2% and 8.5% from Year 1 to Year 17, before reaching 4% to 5% during the processing of the stockpiled material from Year 18 to Year 19.

Processing

A preliminary test program for recovery of the graphite from the Loki Deposit was conducted by SGS Lakefield during 2025 and 2026. The test program included chemical characterization, mineralogical analyses, grindability testing, flotation testing, graphite concentrate, and flotation tailings characterisation.

The test results indicate that the samples respond well to conventional flotation concentration. The flowsheet uses separate flotation with staged regrinding for coarse and fine fractions, respectively. The final concentrates are expected to be approximately 95% Cg or higher.

The impurities in the flotation concentrate are silicon (Si), aluminum (Al), iron (Fe), potassium (K), magnesium (Mg), calcium (Ca), and sulphur (S) and other minor elements, which are expected to be removed or reduced by further purification treatments. The main mineral components in the flotation tailings samples are SiO_2, Al_2O_3, Fe_2O_3, K_2O and MgO. The ABA determination results indicate that the graphite flotation tailings are expected to be potential acid generators.

Based on the test work results, the optimized flowsheet was developed for this study. The process flowsheet is designed for 2,750 t/d and follows a conventional flotation circuit to produce a high-grade graphite concentrate, which will be further processed at an offsite purification facility. The processing plant will consist of the following:

- A run-of-Mine (ROM) mill feed stockpile
- A primary crusher operating in open circuit

- A secondary crusher operating in closed circuit with a vibrating screen
- A crushed mill feed stockpile with reclaim feeders
- A ball mill grinding circuit operating with flash flotation and a classification hydrocyclone cluster
- A rougher flotation circuit comprising rougher and rougher-scavenger flotation, followed by a rougher-scavenger tailings dewatering circuit, including thickening and filtration processes to generate tailings filter cakes for tailings dry stacking at the lined WRTCF
- A rougher concentrate upgrading circuit comprising:
  - One-stage polishing regrinding followed by one stage of cleaner flotation
  - Second-stage polishing mill followed by three stages of cleaner flotation
- A fourth-cleaner concentrate sizing circuit to separate the upgraded rougher concentrate into coarse and fine graphite concentrate streams
- A coarse concentrate regrind mill, followed by four additional stages of cleaner flotation (5th to 8th Cleaners) to produce a final coarse graphite concentrate
- A fine concentrate regrind mill, followed by five additional stages of cleaner flotation (5th to 9th Cleaners) to produce a final fine graphite concentrate
- A concentrate dewatering and product handling circuit, including concentrate thickening, filtration, drying, final product sizing, and packaging

The simplified process flowsheet is shown in Figure 17-1.

Infrastructure

The Project benefits from existing transportation infrastructure, proximity to provincial power network and favourable site characteristics that support future development, including:

- Road Access: The Project is located approximately 15 kilometres south of the Key Lake mill, which processes ore from Cameco Corp.'s McArthur River uranium mine. Provincial Highway 914, a north-south all-weather highway in Saskatchewan, crosses the project, providing ideal access.
- Power: The provincial power grid is merely 15 km from the Project, which can provide long-term cheap and reliable electrical power for the Project.
- Water: Several surface water bodies are located near the project site and are expected to provide suitable water sources for future operations, subject to detailed engineering and permitting.
- Waste Rock Tailings Co-deposition Facility (WRTCF): The WRTCF was designed to accommodate 17.9 Mt of tailings and 108.4 Mt of waste rock over the life of the mine. The WRTCF will consist of commingled dry stacked tailings and waste rock.
- Camp and Services: Existing accommodation camps nearby have the potential to support construction and operations. Camp services are expected to be provided by third-party contractors, creating potential business and employment opportunities for nearby Indigenous communities.

Environmental, Social, and Permitting

In 2025, Abasca engaged CanNorth Environmental Services to conduct environmental studies on the Project. The purpose of the studies was to initiate data collection for components that require baseline datasets, specifically hydrology, water chemistry, and bathymetry to support a future Environmental Impact Assessment. Studies and data collection will continue into 2026 and 2027, including further work on aquatic, terrestrial and heritage resources as well as work on meteorological, hydrogeological and geochemical conditions. This work will advance project development and support future submissions to meet regulatory requirements.

26.0 RECOMMENDATIONS

The completion of the PEA establishes a strong technical foundation for the next stage of engineering. The Company believes the Project is well positioned to advance to feasibility-level engineering.

Future work is expected to focus on metallurgical optimization, geotechnical and hydrogeological investigations, detailed mine, infrastructure and tailings engineering, environmental studies, and continued refinement of the Project's capital and operating cost estimates.

Mineral Resource Estimate

The QP's Mineral Resource related recommendations are summarized below.

- Mineral resources are uncertain because of variability at all scales and sparse sampling. Geostatistical techniques can be used to quantify the uncertainty and the expected reduction of uncertainty in resources as a function of data spacing. The QP recommends that a drill hole spacing study be completed on the deposit to inform drill hole spacing for Indicated Mineral Resource classification. After completion of the drill hole study, definition drilling should be planned and executed accordingly. The drill hole spacing study is estimated to cost US$45k.
- The QP recommends future sample testing include impurity removal testing.
- Customer specifications for flake graphite are typically based on physical properties, particularly flake size, in addition to chemical characteristics. It is recommended that Abasca completes more comprehensive testing for graphite quality. The physical project testing is estimated to cost US$150k.
- Additional delineation drilling is recommended to further upgrade Inferred to Indicated Mineral Resources. Drilling required is estimated to include approximately 6,000 m for US$2.5M.

Mining

Several cost saving opportunities will be studied in the future. They include autonomous (driverless) haulage trucks, trolley assist (using electrical power to drive haul trucks up the pit ramp) and regenerative braking (haul trucks generating electricity and charging the site power grid while travelling downhill). Estimated cost for the studies is US$50k.

Metallurgy and Processing

Further test work should be conducted to further optimize processing conditions, in particular, to conserve graphite flake size. Variability test work should be conducted to better understand metallurgical performances for the mineralization samples collected from different lithological and alteration zones and spatial locations. The costs associated with the recommended metallurgical test work are estimated to be approximately US$450,000, excluding sample collection which can be part of geological drill programs and sample shipment. Large scale pilot testing should be conducted to verify the processing flowsheet developed from the bench scale test work. The tests should produce the concentrate samples for further purification tests and for potential customer off-take tests. The large scale testing is also expected to produce the samples for determining engineering design related data for the next stage of engineering work. The costs for the testing are estimated to be approximately US$400k.

Further optimization for the process flowsheet and engineering design should be conducted, including processing equipment type and size optimizations. The costs should part of the next phase of engineering design.

Graphite purification test work is on-going and the results will be incorporated into the next stage of the process design.

A small scale demonstration flotation process plant is planned to be engineered and built to demonstrate the forecasted economical and technical viability of the Project. The cost for the demonstration plant is estimated to be approximately US$5.0M.

Environmental

In parallel with the Feasibility Study, Abasca intends to advance the environmental assessment process and obtain the regulatory approvals required to support future Project development, building on the substantial environmental studies and technical work completed during the previous Environmental Assessment process. The Company will continue to engage with the Indigenous Groups, regulators and local stakeholders throughout this process.

The PEA applies a conservative estimate for closure costs. Additional geochemical, hydrological, and site-specific engineering studies may allow estimates to be further refined in subsequent technical studies. (Estimated cost for the relevant consultation work and field work: US$1.5M)

Infrastructure

SaskPower funds community initiatives, educational programs, and clean energy development across Saskatchewan to align with its strategic utility goals. As the province's principal electric utility, the Crown corporation provides financial backing through corporate sponsorships, capital grants, and energy-efficiency programs. Abasca will initiate business opportunity discussions with SaskPower for funding power supply infrastructure for the project. (Estimated cost for the relevant consultation work: US$100k)

The Company anticipates evaluating a targeted drilling program to support feasibility-level engineering. The program would be expected to focus primarily on infill drilling, geotechnical investigations and metallurgical sample collection, and mineral resource expansion. (Estimated cost to complete the on-going site investigation program: US$500k)

Government and Critical Minerals Funding Opportunities

Graphite is included on Canada’s critical minerals list. Abasca intends to evaluate available federal and provincial funding, infrastructure and strategic investment programs that may support future engineering, infrastructure development and project advancement. (Estimated cost for the relevant professional work: US$50k)

Strategic Development

The Company will continue evaluating opportunities to advance the Project through strategic partnerships, government-supported critical mineral initiatives and engagement with potential customers and other industry participants.

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28.0 CERTIFICATES OF QUALIFIED PERSONS

I, Matt Batty, MSc, P. Geo, of Saskatoon, Saskatchewan, do hereby certify:

1. I am a Geologist with and owner of Understood Mineral Resources Ltd. of 22 Middleton Crescent, Saskatoon, Canada.
2. I am a graduate of the University of Saskatchewan in 2012 with a B.Sc. degree in Geology and a graduate of the University of Alberta in 2022 with a M.Sc. degree in Mining Engineering (Geostatistics).
3. I am a Registered Professional Geologist (Member No. 25595) with the Association of Professional Engineers and Geoscientists of Saskatchewan (APEGS). I have worked as a geologist for a total of 14 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.
   - Over 10 years of experience in modeling and estimating structurally controlled Mineral Resources.
   - 14 years of experience working in the Athabasca Basin in Northern Saskatchewan where the Loki Deposit it located.
   - The initial Mineral Resource and Technical Report for the Loki Deposit.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "qualified person" for the purposes of NI 43-101.
5. I visited the Key Lake South Project from January 28 to 29, 2025.
6. I am responsible for the Sections 1.2 to 1.7, 1.9, 1.18, 4 to 12, 14, 23, as well as the Mineral Resource sub-sections of 1.19, 1.20, 25.0, 26.0, and 27.0.
7. I am independent of Abasca Resources Inc.
8. I have had no prior involvement with the Project that is the subject of the Technical Report.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 14th day of September 2026, in Saskatoon, Saskatchewan.

(Signed & Sealed) "M.D. Batty"

Matt Batty, Geostatistician, Understood Mineral Resources Ltd.

I, Hassan Ghaffari, P.Eng., M.A.Sc. of Vancouver, British Columbia, 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 "Preliminary Economic Assessment, NI 43-101 Technical Report, For The Loki Flake Graphite Deposit, Key Lake South Project, Saskatchewan, Canada" with effective date of August 19, 2026 (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 30 years of experience in mining and mineral processing plant operation, engineering, project studies and management of various types of mineral processing, including hydrometallurgical mineral processing for porphyry 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 the sections of the Technical Report that I am responsible for preparing.
- I have not conducted a personal inspection of the Project.
- I am independent of Abasca Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Project that is the subject of the Technical Report.
- I am responsible for sections 1.1, 1.13 to 1.16, 2.0, 3.0, 18.0 (except 18.16), 19.0, 20.0, 21.1,21.2, 24.0 and the relevant sections in 1.19, 1.20, 25.0, 26.0 and 27.0.
- I have read the Instrument and the Sections of the Technical Report that I am responsible for has been prepared in compliance with the Instrument.
- 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.

Signed and dated this 14th day of September 2026, in Vancouver, British Columbia.

(Signed and Sealed) *"Hassan Ghaffari"*

Hassan Ghaffari, P.Eng., M.A.Sc.

Director of Metallurgy, Tetra Tech Canada Inc.

I, Jianhui (John) Huang, Ph.D., P.Eng. of Vancouver, British Columbia, do hereby certify:

- I am a Senior Metallurgist with Tetra Tech Canada Inc. with a business address at Suite 1000, 10th Fl., 885 Dunsmuir St., Vancouver, BC, V6B 1N5.
- This certificate applies to the technical report entitled "Preliminary Economic Assessment, NI 43-101 Technical Report, For The Loki Flake Graphite Deposit, Key Lake South Project, Saskatchewan, Canada" with effective date of August 19, 2026 (the "Technical Report"). (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 members in good standing of with Engineers and Geoscientists of British Columbia (#30898).
- My relevant experience includes over 40 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 the purposes of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for the sections of the Technical Report that I am responsible for preparing.
- I have not conducted a personal inspection of the Project.
- I am independent of Abasca Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Project that is the subject of the Technical Report.
- I am responsible for sections 1.8, 1.12, 13.0, 17.0, 21.3.1, 21.3.3 to 21.3.5, and the relevant sections in 1.19, 1.20, 25.0, 26.0 and 27.0.
- I have read the Instrument and the sections of the Technical Report that I am responsible for has been prepared in compliance with the Instrument.
- 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.

Signed and dated this 14th day of September 2026, in Vancouver, British Columbia.

(Signed and Sealed) "*Jianhui Huang*"

Jianhui (John) Huang, Ph.D., P.Eng.
Senior Metallurgist, Tetra Tech Canada Inc.

I, Sabry Hafez, Ph.D., P.Eng. of Vancouver, British Columbia, do hereby certify:

- I am a Senior Mining Engineer with Tetra Tech Canada Inc. with a business address at Suite 1000, 10th Fl., 885 Dunsmuir St., Vancouver, BC, V6B 1N5.
- This certificate applies to the technical report entitled "Preliminary Economic Assessment, NI 43-101 Technical Report, For The Loki Flake Graphite Deposit, Key Lake South Project, Saskatchewan, Canada" with effective date of August 19, 2026 (the "Technical Report"). (the "Technical Report").
- I am a graduate of Assiut University (B.Sc Mining Engineering, 1991; M.Sc. in Mining Engineering, 1996; Ph.D. in Mineral Economics, 2000).
- I am members in good standing of with Engineers and Geoscientists of British Columbia (#34975).
- My relevant experience is in mine evaluation. I have more than 31 years of experience in the evaluation of mining projects, advanced financial analysis, and mine planning and optimization. My capabilities range from the conventional mine planning and evaluation to the advanced simulation-based techniques that incorporate both market and geological uncertainties.
- I am a "Qualified Person" for the purposes of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for the sections of the Technical Report that I am responsible for preparing.
- I have conducted a personal inspection of the Project on June 18, 2026.
- I am independent of Abasca Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Project that is the subject of the Technical Report.
- I am responsible for sections 1.11, 1.17, 16.0, 21.3.2, 22.0 and the relevant sections in 1.19, 1.20, 25.0, 26.0 and 27.0.
- I have read the Instrument and the sections of the Technical Report that I am responsible for has been prepared in compliance with the Instrument.
- 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.

Signed and dated this 14th day of September 2026, in Vancouver, British Columbia.

(Signed and Sealed) "*Sabry Hafez*"

Sabry Hafez, Ph.D., P.Eng.

Senior Mining Engineer, Tetra Tech Canada Inc.

I, Chris Johns, P.Eng., of Kelowna, British Columbia, do hereby certify:

- I am a Geological Engineer with Tetra Tech Canada Inc. with a business address at 150-1715 Dickson Avenue, Kelowna, British Columbia, V1Y 9G6.
- This certificate applies to the technical report entitled "Preliminary Economic Assessment, NI 43-101 Technical Report, For The Loki Flake Graphite Deposit, Key Lake South Project, Saskatchewan, Canada" with effective date of August 19, 2026 (the "Technical Report"). (the "Technical Report").
- I am a graduate of Queen's University, Ontario with a Bachelor of Science degree in Geological Engineering (1994) and of the University of Alberta with a Master of Science degree in Environmental Engineering (1999). My relevant experience includes over 25 years of geological engineering on projects involving design of waste containment facilities. I have been involved with tailings storage facility design from scoping study through feasibility and construction stage. I am a registered Professional Engineer in the Provinces of Alberta and British Columbia.
- I am a "Qualified Person" for the purposes of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for the sections of the Technical Report that I am responsible for preparing.
- I have not conducted a personal inspection of the Project.
- I am independent of Abasca Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Project that is the subject of the Technical Report.
- I am responsible for Sections 18.16 and the relevant sections in 1.19, 1.20, 25.0, 26.0 and 27.0 of the Technical Report.
- I confirm that I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- 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.

Signed and dated this 14th day of September 2026, in Vancouver, British Columbia.

(Signed and Sealed) *"Chris Johns"*

Chris Johns, P.Eng.
Geological Engineer, Tetra Tech Canada Inc.