DOUBLEVIEW GOLD CORP.

National Instrument 43-101 Technical Report
Preliminary Economic Assessment and Updated Mineral Resource Estimate
for the Hat Project, Northwest British Columbia, Canada



QUALIFIED PERSONS:
Shervin Teymouri, P.Eng.
Tomasz Wawruch, FAusIMM.
Andre de Ruijter, P.Eng.
Andrew Carter, C.Eng.
Franky Li, P.Eng.
Jayesh Rami, P.Eng.

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REPORT PREPARED FOR

Client Name	**Doubleview Gold Corp.**
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Contact Name	Farshad Shirvani
Contacts Title	President & CEO
Office Address	**Doubleview Gold Corp** Suite 822-470 Granville St Vancouver, BC, Canada V6C 1V5 T: +1 604 678 9587

REPORT ISSUED BY

Office	**Mineit Consulting Inc.** Suite 1188–1095 West Pender St. Vancouver, BC, Canada V6E 2M6 T: +1 604 620 1300
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Glossary

ABBREVIATIONS AND ACRONYMS

%	percent
3-D	three-dimensional
A3477	sodium dialkyl dithiophosphate
AACE	Association for the Advancement of Cost Engineering
AAS	Atomic absorption spectrophotometer
AES	atomic emission spectroscopy
AIA	Archaeological Impact Assessments
AISC	all-in sustaining cost
ARD	acid-rock drainage
B.C.	British Columbia
BHID	borehole ID
BLK	blank sample
Bureau Veritas	Bureau Veritas Laboratories
CAD	Canadian dollar
CAPEX	capital cost
Cb–Ep	calcite–epidote
CBV	certified best value
CIL	carbon in leach
CIM Definition Standards	CIM Definition Standards for Mineral Resources and Mineral Reserves
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIP	carbon in pulp
COSEWIC	Committee on the Status of Endangered Species in Canada
CR	core recovery
CRM	certified reference material
CV	coefficient of variation
CWP	contact-water pond

DCIP ..... direct current induced polarization
DEHPA ..... di (2-ethylhexylphosphoric acid
DFO ..... Department of Fisheries and Oceans Canada
DMS ..... degrees–minutes–seconds
Dol–Ank ..... dolomite–ankerite
Doubleview ..... Doubleview Gold Corp.
DUP ..... duplicate samples
EAO ..... Environmental Assessment Office
EBITDA ..... earnings before interest, taxes, depreciation, and amortization
EM ..... electromagnetic
EMLI ..... Ministry of Energy, Mines and Low Carbon Innovation
EMS ..... Environmental Monitoring System
ENV ..... Ministry of Environment and Parks
EPCM ..... engineering procurement construction management
FDIQ ..... Fish Inventory Data Queries
Geo4M ..... Geo4M Geological Services Consultant
GPS ..... Global Positioning System
HADD ..... harmful alteration, disruption, or destruction
HCA ..... Heritage Conservation Act
HF ..... hydrogen fluoride
HFSE ..... high-field-strength element
HR ..... human resources
IAA ..... Impact Assessment Act
IAAC ..... Impact Assessment Agency of Canada
ICP AES ..... inductively coupled plasma atomic emission spectroscopy
INAA ..... instrumental neutron activation analysis
IP ..... induced polarization
IRR ..... internal rate of return
IX ..... ion exchange
JA ..... Joint Mines Act and Environment Management Act Application
LOM ..... life of mine
LOP ..... large open pit
masl ..... metres above sea level
MCM ..... Ministry of Mining and Critical Minerals
MDMER ..... Metal and Diamond Mines Effluent Regulations
MIBC ..... methyl-isobutyl carbinol
Micon ..... Micon International Ltd.

Mineit	Mineit Consulting Inc.
MMCM	Ministry of Mining and Critical Minerals
MMO	Major Mines Office
MRE	Mineral Resource estimate
MRMR	Modified Rock Mass Rating
MTO	Mineral Titles Online
NaCN	sodium cyanide
NAG	non-acid-generating
NI 43-101	National Instrument 43-101—Standards of Disclosure for Mineral Projects
NPV	net present value
NSR	net smelter return
O/A	organic phase to aqueous phase
OPEX	operating cost
P_{80}	80% passing
PAG	potentially acid-generating
PEA	preliminary economic assessment
PFS	pre-feasibility study
PGE	platinum group element
Project	Hat Project
QA/QC	quality assurance and quality control
QEMSCAN	quantitative evaluation of materials by scanning electron microscope
QP	Qualified Person
Q-system	rock mass classification system
RF	revenue factor
RMR	Rock Mass Rating
ROM	run-of-mine
RPEE	reasonable prospects for economic extraction
RQD	rock quality designation
RTK	real-time kinematic positioning
RTP	reduced-to-pole
SAG	semi-autonomous grinding
SARA	*Species at Risk Act*
SD	standard deviations
SGS	SGS Lakefield
SII	sequential indicator interpolation
SIPX	sodium isopropyl xanthate
SMU	selective mining unit or small mining unit

STD.....standard samples
SX.....solvent extraction
SX/EW.....solvent extraction and electrowinning
Tahltan.....Tahltan First Nation
TCG.....Tahltan Central Government
TRTFN.....Taku River Tlingit First Nation
TSF.....tailings storage facility
UCS.....uniaxial compressive strength
USD.....United States dollar
USGS.....United States Geological Survey
UTM.....Universal Transverse Mercator
UWR.....Ungulate Winter Range
VMS.....volcanic massive sulphide
WHMIS.....Workplace Hazardous Materials Information Systems
WLRS.....Ministry of Water, Land, and Resource Stewardship
WSF.....waste storage facility
XRF.....X-ray fluorescence spectrometer

UNITS OF MEASURE

"	inch	m	metre
°C	degree Celsius	m^2	square metre
A	ampere	m^3	cubic metre
cm	centimetre	µm	micrometre (micron)
cm^2	square centimetre	masl	metres above sea level
d	day	mg	microgram
d/a	days per annum	m^3/h	cubic metres per hour
dmt	dry metric tonne	ML	million litres
g	gram	Mt	million tonnes
g/L	gram per litre	Mt/a	million tonnes per annum
g/t	gram per tonne	min	minute
g/m^3	grain per cubic metre	mm	millimetre
ha	hectare	MVA	megavolt-amperes
h	hour	MW	megawatt
Hz	hertz	MWh	megawatt-hour
J	joule	oz	troy ounce (31.1035 g)
k	kilo (thousand)	oz Au/a	ounces of gold per annum
kg	kilogram	ppb	parts per billion

km	kilometre	s	second
km^2	square kilometre	t	metric tonne
km/h	kilometres per hour	t/a	metric tonne per year
kPa	kilopascal	t/d	metric tonne per day
kVA	kilovolt-amperes	V	volt
kW	kilowatt	W	watt
kWh	kilowatt-hour	wmt	wet metric tonne
L	litre	wt%	weight percent
L/s	litres per second	Wt	weighted average

1 SUMMARY

1.1 Introduction

Doubleview Gold Corp. ("Doubleview" or the "Issuer") (TSX-V:DBG) requested that Mineit Consulting Inc. (Mineit) prepared a report titled *National Instrument 43-101 Technical Report on the Preliminary Economic Assessment and Updated Mineral Resource Estimate of the Hat Project, Northwest British Columbia, Canada* (this Technical Report or PEA). This Technical Report was prepared in accordance with National Instrument 43-101 (NI 43-101), and evaluates the potential economic performance of the Hat Project.

This PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the results of the PEA will be realized.

The PEA contemplates a conventional open-pit mine and processing operation with a 25-year mine life at a 120,000 t/d (42 Mt/a) plant throughput. Two processing pathways were evaluated, A1 and its alternative, A2, and B: the first alternative, A1, is a Cu–Au–Ag–Co flotation concentrator with two recovery cases based on current metallurgical testwork, and A2, reflecting expected performance; and B, a full circuit that retains the base flowsheet and adds a pyrite-cobalt flotation circuit together with downstream hydrometallurgical recovery circuit for recovery of cobalt, additional gold and silver, and scandium.

The PEA is based on an updated Mineral Resource estimate, which is detailed in this Technical Report. A site visit to the Hat Project was conducted in October 2025 by Tomasz Wawruch, geological and geostatistical QP.

Unless otherwise stated, all monetary values are presented in Canadian dollars ($), all measurements are in metric units, and gold and silver are weighed in troy ounces.

1.2 Property Description and Ownership

The Hat Project is in northwestern British Columbia, within the Atlin Mining Division. The property lies north of Hatchau Lake, approximately 95 km southwest of Dease Lake and 40 km north of Telegraph Creek. The southwest part of the mineral tenures is positioned at approximately UTM Zone 9, 345,211 and 6,453,290 (latitude 58°12' N, longitude 131°35' W), corresponding to National Topographic Service map sheets 104J/12, 104J/13, 104J/22, and 104J/23. The Hat Project comprises 16 mineral tenures totalling 13,823.09 ha.

1.3 Geological Setting and Mineralization

The Hat Project is situated within the Stikine Terrane of northwestern British Columbia, a region underlain by Late Triassic volcanic and sedimentary rocks of the Stuhini Group and intruded by multiphase dioritic to gabbroic bodies. These rocks form part of the broader Stuhini–Hazelton arc succession that hosts

several major porphyry and intrusion related deposits across the Golden Triangle. At Hat, the Stuhini Group comprises andesitic to basaltic flows, tuffs, and volcaniclastic strata interlayered with argillite and wacke, reflecting deposition in an active island arc to arc proximal marine basin. This stratigraphy is intruded by a heterogeneous suite of diorite, monzonite, gabbro, and pyroxenite phases that collectively define the intrusive architecture of the Lisle Zone.

Mineralization is dominated by chalcopyrite, pyrite, and magnetite occurring within both volcanic and intrusive host rocks. Copper and gold form the core of the system, with silver and cobalt increasing outward into more distal or structurally influenced domains. Scandium is present at background levels and is primarily controlled by mafic to intermediate lithologies and propylitic alteration. The mineralizing system is closely associated with potassic and inner propylitic alteration assemblages, which overprint earlier sodic–calcic halos. Magnetite is abundant and locally forms semi massive pods, reflecting a strong magmatic–hydrothermal overprint.

Four principal mineralized domains: copper, gold, silver, and cobalt form steep, laterally continuous corridors aligned with the northwest–southeast structural grain (azimuth 150–160°). Each domain exhibits a centrally thickened core and progressively narrowing margins along strike. The copper and gold domains are best developed between 347600 E and 348400 E, where overlapping steeply dipping lenses create vertically persistent mineralized volumes. The silver domain strengthens rapidly eastward into a coherent central corridor, while the cobalt domain is the most laterally extensive, forming a broad lens like body with exceptional vertical continuity.

Collectively, the geological, structural, and alteration characteristics of the Hat system define an intrusion centered mineralizing environment that departs from classic porphyry models, instead reflecting a hybrid magmatic–hydrothermal system with strong structural control, pervasive magnetite development, and a distinctive metal zonation pattern. These features provide a robust framework for ongoing exploration and resource evaluation.

1.4 Exploration Status

Exploration at the Hat Project has advanced from early reconnaissance work in 2013 to a fully integrated program combining geochemistry, geophysics, and systematic drilling. Initial prospecting, mapping, and early drill testing were followed by targeted soil and rock-geochemistry surveys, including a 2018 B-horizon soil program that collected 433 samples and successfully outlined copper–gold anomalies coincident with the Lisle Zone and its extensions. These geochemical datasets, together with airborne magnetics and modern 3-D direct current induced polarization surveys, established clear lithogeochemical controls on mineralization and guided subsequent drill targeting.

Between 2013 and 30 September 2025, a total of 101 diamond drill holes were completed, for a cumulative drilled length of 49,885.6 m. Of these, 97 drill holes, totaling 49,548 m, were accepted and used in the mineral resource update presented in this report. Subsequently, during spatial analytical processing, all assay intervals were regularized to 10 m long, forming the analytical foundation for mineral resource domain modelling and mineral resource estimation.

Drilling has confirmed the presence of a large alkalic porphyry copper–gold system with pervasive silver and cobalt mineralization, extending laterally over approximately 2 × 2 km and to depths approaching 900 m.

The integrated exploration history provides a mature geological framework and a high-confidence dataset that underpins the current Mineral Resource estimate and ongoing project development.

1.5 Mineral Resource Estimates

The Mineral Resource estimate presented in this PEA represents an update to the 2024 Micon estimate and incorporates all drilling completed at the Hat Project up to September 2025. The estimate was prepared by Tomasz Wawruch, acting as the Qualified Person for the Mineral Resource, in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) *CIM Definition Standards for Mineral Resources & Mineral Reserves* (CIM Definition Standards) (CIM, 2014), as required under NI 43-101.

The MRE is supported by a validated drill-hole database comprising 97 sampled diamond drill holes totaling 49,548 m, containing 23,526 primary samples for copper–gold–silver–cobalt and 23,131 samples for scandium. All analytical intervals were regularized into 10 m composites (4,882 composites for Cu–Au–Ag–Co and 4,797 for Sc) to provide consistent support for geostatistical analysis and Mineral Resources modelling.

Five variables—copper, gold, silver, cobalt, and scandium—were estimated independently. No geological sub-domains were applied, as lithology and alteration did not demonstrate sufficiently consistent quantitative control on grade. This approach is consistent with CIM guidance for deposits where geological controls are gradational and grade continuity is best captured statistically.

Copper equivalent percent (CuEq %) values were calculated to integrate metal prices and metallurgical recoveries into a single reporting metric. The CuEq formula used in this estimate incorporates recoveries of Cu 85%, Au 89%, Co 78%, and Ag 68%, and is fully disclosed in Section 14. The formulation complies with NI 43-101 requirements for transparency and does not include scandium, which follows a separate metallurgical pathway.

Mineral Resources are reported within an optimized open-pit shell of 120 kt/d at a 0.20% CuEq cut-off grade, reflecting reasonable prospects for eventual economic extraction. Classification into Measured, Indicated, and Inferred categories follows CIM (2014) standards and is based on the volume–variance relationship, estimation variance, and the assumed production scale, providing a practical and quantitative framework for assessing the uncertainty associated with the estimates. This classification approach offers a transparent and defensible representation of estimation confidence. The resulting MRE (Table 1-1) defines:

- Measured Resources: 272 Mt at 0.44% CuEq
- Indicated Resources: 337 Mt at 0.43% CuEq
- Total Measured + Indicated: 609 Mt at 0.43% CuEq
- Inferred Resources: 503 Mt at 0.41% CuEq.

All resources are reported within the optimized pit shell and reflect the geological interpretation, drill-hole spacing, and estimation parameters available as of the effective date, February 4, 2026 (Table 1-1 and Table 1-2).

Table 1-1: Hat MRE at a 0.2% CuEq Cut-Off Effective February 4, 2026

Mineral Resource Classification	Tonnage (Mt)	Average Grade					Metal Content				
		CuEq (%)	Cu (%)	Au (g/t)	Co (g/t)	Ag (g/t)	CuEq (Blb)	Cu (Blb)	Au (Moz)	Co (Mlb)	Ag (Moz)
Measured	272	0.44	0.22	0.18	76.26	0.37	2.61	1.11	1.41	35.6	2.17
Indicated	337	0.43	0.21	0.19	76.81	0.39	3.21	1.31	1.81	44.5	2.88
Total M+I	609	0.43	0.21	0.18	76.57	0.38	5.82	2.42	3.22	80.1	5.05
Inferred	503	0.41	0.18	0.19	76.62	0.38	4.57	1.72	2.77	66.2	4.19

Table 1-2: Hat MRE at a 0.2% CuEq Cut-Off as of February 4, 2026, Scandium Oxide Resources

Mineral Resource Classification	Tonnage (Mt)	Sc Tonnage[1] (Mt)	Average Grade Sc (g/t)	Metal Content[2] Sc_2O_3 (t)
Measured	272	34	28.79	1,081
Indicated	337	42	28.76	1,334
Total M+I	609	76	28.77	2,415
Inferred	503	63	28.69	1,996

Notes:

[1] Scandium tonnages represent 12.5% of the mineralized material by category, reflecting the proportion of tailings expected to be processed through a dedicated scandium leach circuit under current metallurgical design constraints.

[2] Scandium oxide metal content has been calculated using the metallurgical recovery of 72% and conversion factor from Sc to Sc_2O_3 of 1.534.

- Mineit's Qualified Person, Tomasz Wawruch, FAusIMM, completed the MRE, and has reviewed and approved the technical disclosure related to the MRE contained in this news release. Mr. Wawruch is a senior geology and mineral resource consultant independent of Doubleview. Mr. Gilles Arseneau, PhD., P.Geo., of ARSENEAU Consulting Services Inc., provided an independent review of this MRE.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
- Inferred Mineral Resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves.
- The Mineral Resource Estimate was prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards (CIM, 2014), and CIM Best Practice Guidelines (CIM 2019).
- The effective date of the MRE is February 4, 2026.
- Metal contents have been calculated using the following metallurgical recovery factors: Cu = 85%, Au = 89%, Co = 78%, and Ag = 68%.
- Economic assumptions used include US4.80/lb Cu, US20.00/lb Co, US3,200/oz Au, US46/oz Ag, and a 2% NSR royalty.
- Mineral Resources are reported within optimized open pit constraints and 0.2% CuEq cut-off grade, based on a C7.93/t milled processing cost and C2.90/t milled general and administrative cost, with a mining cost of C3.01/t plus incremental mining cost increasing by C0.015/t for every bench below the reference level of 1,125 mRL.
- CuEq calculations do not include scandium. The formula used to calculate CuEq is:
  CuEq = [(((Ag × 46.0 × 0.68)/31.1035) + ((Au × 3200 × 0.89)/31.1035) + 0.0001 × (Co × 20.0 × 0.78 × 22.0462) + 0.0001 × (Cu × 4.8 × 22.0462 × 0.85))/(4.8 × 22.0462 × 0.85)], where all input variables are expressed in (ppm) and CuEq is expressed in percent (%).
- Rounding may result in minor variations between individual values and totals; such differences are not considered material to the MRE.
- Mineral Resource classification reflects the level of geological confidence and satisfies the uncertainty criteria appropriate for exploration and resource development. Additional drilling will be required to reduce uncertainty to the level expected for production planning.
- The MRE reflects the geological interpretation, drill-hole spacing, and estimation parameters available at the time of modelling. Any additional drilling is expected to influence the current outcome by improving confidence in the estimates and refining the geometry of the mineralized domains.
- The Mineral Resource results are presented in situ within the optimized pit. Mineralized material outside the pit has not been considered as a part of the current MRE tabulation. Calculations used metric units (metres, tonnes, g/t).
- A total of 97 diamond drill holes, comprising 49,548 m of core, were incorporated into the Mineral Resource Estimate. All drilling data used in the MRE were subject to standard QA/QC validation prior to inclusion.

1.6 Mineral Reserve Estimates

There are no current mineral reserve estimates for the Hat Project.

1.7 Mining Methods

The mine plan for the Hat Project is based on conventional open-pit truck-and-shovel mining at a nominal processing rate of 42 Mt/a over a 25-year mine life. Pit optimization and mine planning were completed using a 0.30% CuEq cut-off grade for direct mill feed, with material grading between 0.20% and 0.30% CuEq designated for stockpiling and deferred processing during the later years of operation.

Pit optimization identified an economic pit shell at a revenue factor of 0.81. The selected pit design contains approximately 1,023 Mt of total mill feed at an average grade of 0.44% CuEq, comprising both direct mill feed and deferred low-grade stockpile material, together with approximately 1,644 Mt of waste, for an overall strip ratio of 1.6:1. Of the total mill feed, approximately 242 Mt is scheduled for stockpiling and later processing. Table 1-3 summarizes the material inventory by resource classification.

Table 1-3: Material Inventory Breakdown by Resource Category

	Resources Classification	Tonnage (Mt)	Average Grade				
			CuEq (%)	Cu (%)	Au (g/t)	Co (g/t)	Ag (g/t)
ROM Ore (>0.30% CuEq)	Measured	217	0.48	0.23	0.19	78.81	0.39
	Indicated	225	0.51	0.24	0.21	80.27	0.43
	Total M+I	**443**	**0.49**	**0.24**	**0.20**	**79.56**	**0.41**
	Inferred	340	0.49	0.20	0.22	78.98	0.39
Low-Grade Stockpile 1 (0.25%–0.30% CuEq)	Measured	13	0.27	0.10	0.13	69.34	0.25
	Indicated	55	0.28	0.10	0.13	73.63	0.35
	Total M+I	**68**	**0.28**	**0.10**	**0.13**	**72.81**	**0.33**
	Inferred	61	0.28	0.11	0.12	72.86	0.38
Low-Grade Stockpile 2 (0.20%–0.25% CuEq)	Measured	14	0.23	0.07	0.11	67.04	0.20
	Indicated	27	0.23	0.08	0.11	67.29	0.29
	Total M+I	**41**	**0.23**	**0.08**	**0.11**	**67.20**	**0.26**
	Inferred	72	0.22	0.09	0.08	75.65	0.36
Total Mill Feed	Measured	244	0.45	0.22	0.18	77.66	0.37
	Indicated	308	0.45	0.20	0.19	77.93	0.40
	Total M+I	**551**	**0.45**	**0.21**	**0.18**	**77.81**	**0.39**
	Inferred	472	0.42	0.18	0.19	77.69	0.39

The production schedule was developed to prioritize higher-grade and higher-value mill feed in the early years of operation, while lower-grade material is stockpiled for treatment in the later years of the mine life. Mining is assumed to be conducted in 15 m benches, which were selected based on the scale of the

operation, equipment productivity, and the required total material movement rate. Drill-and-blast operations are required for both ore and waste.

The mining fleet is based on a conventional large-scale open-pit configuration comprising 67.6 m^3 electric rope shovels, 38 m^3 hydraulic shovels, 290-tonne haul trucks, and the associated drilling, loading, dozing, grading, and auxiliary support equipment required to achieve the planned mining rates.

The mine plan includes Measured, Indicated, and Inferred Mineral Resources. Inferred Mineral Resources 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 mine plan or economic results will be realized.

1.8 Mineral Processing and Metallurgical Testing

The PEA evaluates two processing pathways at a nominal plant throughput of 120,000 t/d (42 Mt/a). Scenarios A1 and A2 are based on a conventional copper-gold-silver flotation concentrator, while Scenario B represents an expanded development case that retains the base concentrator and adds a pyrite-cobalt flotation circuit together with a downstream hydrometallurgical circuit for recovery of cobalt, additional gold and silver, and scandium.

The base concentrator includes conventional crushing, grinding, copper flotation, concentrate handling, and tailings management facilities. Under Scenarios A1 and A2, the Project is assumed to produce a marketable copper concentrate containing payable copper, gold, and silver, with cobalt credits reported through the concentrate terms applied in the economic analysis. No pyrite-cobalt flotation circuit is included in these two cases.

Under Scenario B, the base concentrator is expanded to include a pyrite-cobalt flotation circuit. The resulting cobalt-enriched pyrite concentrate will be processed through an additional roasting and hydrometallurgical circuit to recover cobalt and additional gold and silver, while sulphuric acid generated within the process will be used in a downstream leach circuit for scandium recovery from flotation tailings. The pyrite-cobalt flotation circuit and associated hydrometallurgical and scandium recovery circuit are therefore unique to Scenario B. The scandium circuit is based on a less mature process route than the conventional flotation circuit and therefore carries higher technical and commercialization uncertainty.

The metallurgical assumptions applied in the PEA are based on testwork completed by Sepro Laboratories, Langley, British Columbia. Early-stage metallurgical testwork focused on characterization and demonstration of metal recoverability under standard laboratory conditions for support of the Mineral Resource estimate and the PEA. Flotation conditions were not fully optimized, and more advanced variability and optimization testwork remains to be completed. Accordingly, the recoveries adopted in Scenario A1 are considered conservative and reflect current testwork results, while Scenario A2 represents an expected-performance case based on anticipated improvement through additional optimization and development work.

For the economic analysis, Scenario A2 assumes improved copper and gold flotation performance relative to the current testwork, including copper recovery of 89% and gold recovery of 75%. Scenario B incorporates the additional downstream recovery of cobalt, gold, silver, and scandium associated with the hydrometallurgical circuit.

The process recoveries applied in the economic analysis are summarized in Table 1-4.

Table 1-4: Net Recovery for Each Processing Scenario

Net Recovery	Scenario A1	Scenario A2	Scenario B
Cu Recovery	80%	89%	85%
Au Recovery	66%	75%	89%
Ag Recovery	53%	53%	68%
Co Recovery	30%	30%	78%
Sc Recovery	N/A	N/A	72%

1.8.1 Process Plant

The envisaged treatment plant will be designed for a nominal throughput of 120,000 t/d and will consist of two parallel processing lines, each rated at approximately 60,000 t/d. Each line will include a primary crushing stage, followed by a grinding and classification circuit, which in turn is followed by the first flotation circuit to recover and upgrade copper to a saleable copper-gold concentrate.

Under Scenario A1 and A2, the process plant consists only of this base concentrator configuration. The resulting copper concentrate will be thickened, filtered, and stored prior to shipment to a smelter.

Under Scenario B, the base concentrator is expanded to include an additional pyrite-cobalt flotation circuit. In this case, a cobalt-enriched pyrite concentrate will also be produced, thickened, filtered and stored prior to downstream processing for recovery of cobalt, gold and silver through the hydrometallurgical circuit.

Final flotation tailings will be dewatered in tailings thickeners. Under Scenario B, a portion of the tailings stream will be diverted for scandium recovery. Under Scenario A1 and A2, no tailings are diverted for scandium recovery, and all thickened tailings will be deposited in the tailings storage facility.

Water management will include recycling of water from tailings storage facility to the process plant. Fresh make-up water will be required for gland service water, reagent preparation and the fire water circuit.

1.8.2 Hydrometallurgical Plant

The hydrometallurgical process will treat the pyrite concentrate for the recovery of copper, cobalt, gold, and silver, and the production of sulphuric acid, which will be used to extract copper and cobalt from the roaster calcine and scandium from flotation tailings. The process is in the early stages of development and is currently at a low level of maturity and is regarded as conceptual. Only the recovery of scandium has been

demonstrated in the laboratory with scandium in flotation tailings being successfully recovered to a di-scandium trioxide product. However, recovering copper, cobalt, gold, and silver from the roaster calcine will employ established methods such as SX/EW and carbon in pulp (CIP) or carbon in leach (CIL) and is not expected to present any undue technical difficulty. All stages of the proposed hydrometallurgical process will be tested during the planned metallurgical test work program. Currently, treatment of tailings to recover scandium is only limited by the acid that can be produced internally; higher levels of scandium production would require external supplies of pyrite, sulphur, or bulk acid.

The hydrometallurgical treatment facilities will include the following unit processes:

- Pyrite concentrate drying
- Pyrite roasting and sulphuric acid production
- Base metals extraction and recovery
  - Cementation, SX and IX, EW or intermediate salt production
- Gold and silver recovery
- Scandium extraction and recovery.

1.9 Project Infrastructure

Project infrastructure for the Hat Project has been conceptually defined to support a 120,000 t/d open-pit mining and processing operation. The current PEA-level design includes site access via a new all-season road from the existing regional road network, grid power connection with new transmission infrastructure, and the principal on-site facilities required for mining and processing, including the process plant, stockpiles, maintenance facilities, camp, water management infrastructure, tailings storage facility, and waste rock storage facility.

The infrastructure layout and supporting facilities are conceptual in nature and are considered appropriate for a PEA. Further work in subsequent study stages will be required to refine the access and power supply strategy, site layout, water management systems, and the design of major storage facilities, as well as to advance the geotechnical, hydrological, geochemical, environmental, and permitting studies needed to support project development.

1.9.1 Tailings Storage Facility

The TSF is located approximately 10 km north of the proposed open pit. The selected site was identified through an alternatives assessment, and the facility is designed as a perimeter dyke system with a starter dam constructed from NAG waste rock or borrow material followed by centerline raises using compacted cycloned sand sourced from the coarse underflow of tailings processed through an onsite cyclone plant. The TSF is classified at a preliminary High to Extreme consequence category, and the design basis assumes compliance with applicable CDA, BC regulatory, and Global Industry Standard on Tailings Management requirements. Key conceptual design criteria include a 20 m crest width, 5 m freeboard, minimum 200 m

beach length, and minimum downstream slopes of 4H:1V. Tailings would be delivered by pipeline, with seepage recovery and reclaim water return incorporated into the water management concept.

The TSF design is conceptual in nature and appropriate for a PEA-level study only. Further work will be required during subsequent stages to confirm foundation conditions, tailings and construction material properties, geochemical performance, hydrology and hydrogeology, seismic design basis, water management requirements, closure strategy, and overall storage configuration.

1.10 Environmental Studies, Permitting, and Social or Community Impact

Environmental review for the Hat Project is at an early, conceptual stage, consistent with the level of a PEA. Existing work has identified key environmental, permitting, and stakeholder considerations; however, substantial additional baseline studies, technical assessments, and engagement with First Nations, regulators, and other stakeholders will be required to support future project advancement.

Project development is expected to require provincial and federal permitting and environmental review, and further work in subsequent study stages will be needed to advance environmental characterization, support project design, and inform permitting and consultation activities.

1.10.1 Setting and Context

The property contains environmentally or culturally sensitive features that require appropriate management and mitigation of potential impacts as a result of the Project. Sensitive features include:

- Critical habitat for species at risk: Stone's sheep (Ovis dalli stonei, Ungulate Winter Range), Northern Caribou (*Rangifer tarandus caribou*, Level-Kawdy herd), and grizzly bear (*Ursus arctos horribilis*).
- Two classified endangered ecosystems within the claim boundary.
- Four pre-contact archaeological sites identified through Archaeological Impact Assessment (AIA), requiring avoidance and/or ongoing management.
- Three overlapping trapline tenures (TR0622T017, TR0626T001, TR0626T003).

1.10.2 Baseline Studies and Characterization

A comprehensive suite of baseline studies is planned to support regulatory submissions and environmental assessment. Baseline study designs will incorporate input from participating First Nations, with Traditional Knowledge of the area considered. Baseline studies that will inform the environmental impact assessment and permitting process, include water quality and quantity monitoring, terrestrial and aquatic ecosystem characterization, and species at risk surveys.

1.10.3 First Nations and Community Engagement

Meaningful consultation with the Tahltan Nation and Taku River Tlingit First Nation is required under both the BC *Environmental Assessment Act* and the federal *Impact Assessment Act*. The Project lies within the traditional territories of both Nations. Engagement priorities include:

- Identification of valued components and Traditional Use areas.
- Assessment of potential project impacts, positive or negative, to communities.
- Guidance regarding existing land and resource use plans that include the Project area and include the Taku River Tlingit-Arlin Taku Land Use Plan and the Tahltan-BC Phase 1 Land Use Plan (under development).
- Cultural heritage protocols and site management approaches during construction and operations.

Local community and stakeholder engagement will be conducted to comply with assessment and permitting standards. A human health and socioeconomic assessment is anticipated to forecast and address potential community effects.

1.10.4 Permitting and Approvals

Key future environmental authorizations required for project advancement include provincial and federal environmental assessment certificates, Major Mines approval, mining leases, water licenses, and various minor permits. The federal and provincial environmental assessment processes are expected to proceed either concurrently or through substitution under the Canada-BC Impact Assessment Cooperation Agreement, with issuance of both a Federal Decision Statement and Provincial Environmental Assessment Certificate. The Major Mines Office coordinated approach is anticipated to result in the *Mines Act* and *Environmental Management Act* approvals.

1.10.5 Risk and Mitigation

Based on preliminary baseline information, the following environmental risks have been identified and will be addressed through the impact assessment and permitting process:

- Species at Risk Impacts—Mitigation through habitat avoidance, timing restrictions, and offset measures should field studies confirm habitat within the potentially impacted area of the Project.
- Archaeological Site Protection—Avoidance of four identified pre-contact sites; alteration permits and monitoring as necessary.
- Water Quality and Quantity Impacts—Baseline data collection and a conceptual site model will be completed to inform water management design.
- Community and Socioeconomic Effects—Engagement and assessment will be completed to identify and mitigate potential concerns with the goal of informed consent.

Environmental commitments, mitigation measures, and monitoring programs will be formalized through permit conditions and the Project Environmental Management System. Environmental study information remains at a conceptual stage consistent with PEA-level studies, and substantial additional baseline studies and technical assessments will be required to support project advancement.

1.11 Capital and Operating Costs

1.11.1 Capital Costs

Table 1-5 summarizes the estimated capital costs (CAPEX) for the three PEA scenarios, including initial capital, sustaining capital, and closure and reclamation costs, reported in millions of Canadian dollars ($M).

Total initial capital is estimated at $3,552 million for Scenario A1, $3,601 million for Scenario A2, and $3,828 million for Scenario B. The higher initial capital in Scenario B reflects the additional processing scope and associated indirect and contingency costs related to the expanded processing configuration.

Total sustaining capital is estimated at $2,755 million for Scenarios A1 and A2, and $4,006 million for Scenario B. The increase in Scenario B is primarily attributable to the inclusion of the hydrometallurgical plant and scandium recovery circuit within sustaining capital. Closure and reclamation costs are estimated at $503 million for all scenarios.

Table 1-5: Initial Capital Cost Summary ($ millions)

Capital Cost Summary	Scenario A1	Scenario A2	Scenario B
Initial CAPEX			
Processing Plant (Excl. Hydrometallurgical Plant)	1,609	1,645	1,810
Mining CAPEX	394	394	394
Mining Pre-Stripping	97	97	97
Infrastructure (Power/Water/Roads/Camp)	326	326	326
Tailings And Water Management	157	157	157
Indirects + EPCM	258	262	278
Contingency (25%)	710	720	766
Total initial CAPEX	**3,552**	**3,601**	**3,828**
Sustaining CAPEX			
Processing Plant (Inc. Hydrometallurgical Plant)	285	285	1,194
Mining CAPEX	811	811	811
Infrastructure (Power/Water/Roads/Camp)	63	63	63
Tailings and Water Management	1,065	1,065	1,065
Indirects + EPCM	142	142	233
Contingency (25%)	390	390	640
Total Sustaining CAPEX	**2,755**	**2,755**	**4,006**
Closure and Reclamation	503	503	503

1.11.2 Operating Costs

Table 1-6 summarizes the principal operating cost assumptions and commercial terms applied in the PEA. Average site operating costs are estimated at approximately $16.2/t milled for the flotation-only cases (Scenarios A1 and A2) and $21.9/t milled for Scenario B, with the increase in Scenario B driven by the additional hydrometallurgical, acid generation, and scandium recovery circuits.

Commercial assumptions include concentrate transport and selling costs, treatment and refining charges, and payable terms for copper concentrate and, where applicable, for on-site produced metals under Scenario B.

On a payable metal basis, estimated C1 cash costs are US$1.75/lb CuEq for Scenario A1, US$1.74/lb CuEq for Scenario A2, and US$2.11/lb CuEq for Scenario B. Estimated all-in sustaining costs (AISC) are US$2.04/lb CuEq for Scenario A1, US$2.03/lb CuEq for Scenario A2, and US$2.47/lb CuEq for Scenario B

Table 1-6: Operating Cost Summary

Operating Cost Summary	Unit	Value
Average Mining Cost	$/t-moved	2.32
Processing Cost—Up to Concentrate Production (Scenario A)	$/t-milled	7.93
Hydrometallurgical + Acid Generation (Scenario B)	$/t-milled	3.08
Scandium Oxide Processing (Scenario B)	$/Kg Sc_2O_3	939.55
G&A	$/t-milled	2.56
Total Operating Cost	**$/t-milled**	**21.92**
Cu-Au Concentrate Product		
Transport and Selling	$/dmt	95.90
TC Cu-Au Concentrate	$/dmt	77.00
Refining Cost—Cu	$/lb	0.11
Refining Cost—Au	$/oz	6.85
Refining Cost—Ag	$/oz	0.48
Refining Cost—Co	$/lb	0.16
Payable—Cu	%	97
Payable—Au	%	95
Payable—Ag	%	90
Payable— Co	%	80
Metal Production On-Site (Scenario B)		
Payable—Au	%	97
Payable—Ag	%	97
Payable—Co	%	97
Payable—Sc_2O_3	%	95
C1 / Cash Cost (Scenario A1/A2/B)	US$/lb CuEq payable	1.75 / 1.74 / 2.11
AISC (Scenario A1/A2/B)	US$/lb CuEq payable	2.04 / 2.03 / 2.47

1.12 Economic Analysis

Table 1-7 summarizes the principal economic assumptions and key financial results for the three PEA scenarios evaluated for the Hat Project. Using a long-term price deck of US$4.88/lb Cu, US$3,272.60/oz Au, US$50.22/oz Ag, and US$19.57/lb Co, together with an exchange rate of 1.37 Canadian dollar (CAD):United States dollar (USD), the Project generates average annual earnings before interest, taxes, depreciation, and amortization (EBITDA) of $886 million for Scenario A1, $1,071 million for Scenario A2, and $1,284 million for Scenario B. Scenario B also includes a scandium oxide price assumption of US$1,500/kg Sc_2O_3.

Table 1-7: Financial Metrics Consensus Metal Prices

Metric	Unit	Scenario A1	Scenario A2	Scenario B
Financial Metrics				
Exchange Rate	CAD:USD	1.37		
Long Term Copper Price	US$/lb	4.88		
Long Term Gold Price	US$/oz	3,272.60		
Long Term Silver Price	US$/oz	50.22		
Long Term Cobalt Price	US$/lb	19.57		
Long Term Scandium Oxide Price	US$/kg	N/A		1,500
Average Annual EBITDA	C$M	886	1,071	1,284
Total EBITDA	C$M	22,162	26,770	32,101
Average Annual Free Cash Flow (Pre-tax)	C$M	756	940	1,104
Free Cash Flow (Pre-tax)	C$M	18,904	23,511	27,592
Total Provincial Tax (Including BC Mineral Tax)	C$M	(4,029)	(5,090)	(6,019)
Total Federal Tax	C$M	(1,274)	(1,859)	(2,308)
Total Taxes	C$M	(5,303)	(6,949)	(8,327)
Average Annual Free Cash Flow (Post-tax)	C$M	544	662	771
Free Cash Flow (Post-tax)	C$M	13,601	16,562	19,265
Total Free Cash Flow (Pre-tax)	C$M	15,352	19,910	23,764
Total Free Cash Flow (Post-tax)	C$M	10,050	12,961	15,437
NPV 5% (Pre-Tax)	C$M	7,883	10,576	11,567
NPV 5% (Pre-Tax)	US$M	5,754	7,720	8,443
IRR (Pre-Tax)	%	24	29	23
Payback (Pre-Tax)	years	Year 5	Year 4	Year 6
NPV 5% (Post-Tax)	C$M	4,963	6,727	7,274
NPV 5% (Post-Tax)	US$M	3,623	4,911	5,309
IRR (Post-Tax)	%	19	23	19
Payback (Post-Tax)	years	Year 6	Year 5	Year 7

On a post-tax basis, the Project has an estimated NPV(5%) of C$4,963 million for Scenario A1, C$6,727 million for Scenario A2, and C$7,274 million for Scenario B, with corresponding internal rate of returns (IRR) of 19%, 23%, and 19%, respectively. Post-tax payback is estimated in Year 6 for Scenario A1, Year 5 for Scenario A2, and Year 7 for Scenario B. Total post-tax free cash flow is estimated at C$10,050 million for Scenario A1, C$12,961 million for Scenario A2, and C$15,437 million for Scenario B.

The economic results indicate that the base case (Scenario A2) provides the strongest return profile among the three cases, while Scenario B delivers the highest total cash flow and NPV but with higher capital intensity and longer payback.

As this analysis is based on a PEA, it 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. There is no certainty that the economic results presented will be realized.

1.13 Interpretation and Conclusions

The Hat Project is interpreted to be a large-scale polymetallic porphyry-related system with the potential to support a long-life open-pit mining operation. The results of this PEA indicate that, under the assumptions applied herein, the Project demonstrates potential for economic development; however, the study is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the results of the PEA will be realized.

Geological interpretation and drilling completed to date support good continuity of copper, gold, silver, cobalt, and scandium mineralization within the currently defined deposit area. The updated Mineral Resource Estimate benefits from the 2024–2025 drilling program, which materially increased drilling density, expanded the mineralized envelope, and improved confidence in the resource model, including the delineation of a substantial Measured and Indicated inventory. The Project also retains upside potential through additional mineralization identified outside the current resource shell.

The mine plan indicates that conventional open-pit truck-and-shovel mining is an appropriate basis for development of the Project at the PEA level. The selected mine plan supports processing at 42 Mt/a over a 25-year mine life, with stockpiling of lower-grade material for later processing. While the mining concept is considered appropriate for this level of study, further geotechnical, hydrogeological, and mine design work will be required to refine pit slope design, mine scheduling, and waste and water management strategies.

Metallurgical testwork completed to date supports the conceptual application of a conventional flotation flowsheet for recovery of copper, gold, silver, and cobalt. Scenario A1 is based on current testwork results, Scenario A2 reflects an expected-performance case following optimization, and Scenario B incorporates an additional hydrometallurgical circuit for cobalt and scandium recovery. The metallurgical program completed to date is limited in scope and is not sufficient to support detailed process plant design. Additional work is required to better define ore variability, comminution characteristics, flotation

performance, reagent selection, concentrate quality, locked-cycle response, and the technical viability of the proposed downstream scandium recovery process.

Infrastructure, tailings, and water management concepts developed for the PEA are considered suitable for this stage of study but remain conceptual. Additional engineering and field investigations will be required to refine the infrastructure layout, tailings storage strategy, water management approach, and supporting geotechnical, hydrological, hydrogeological, geochemical, environmental, and permitting assumptions. Project advancement will also require continued engagement with Indigenous Nations, regulators, and other stakeholders.

From an economic perspective, all three scenarios evaluated in the PEA generate positive results. Scenario A2 provides the strongest overall return profile, while Scenario B provides the highest NPV and cash flow potential but also introduces greater capital intensity, process complexity, and execution risk. The principal opportunities for the Project include further resource growth, conversion of resource classification, metallurgical optimization, and potential additional value from cobalt and scandium recovery. Key uncertainties include reliance on Inferred Mineral Resources, limited metallurgical testwork, conceptual infrastructure and tailings designs, geotechnical and hydrogeological constraints, and the technical and commercial uncertainty associated with scandium recovery.

Advancement of the Hat Project will require additional drilling, expanded metallurgical testing, geotechnical and hydrogeological investigations, refinement of mine and infrastructure designs, environmental baseline studies, and further trade-off analysis to define the preferred development pathway for future study stages.

1.14 Recommendations

A phased program of technical studies and engineering work is recommended to advance the Hat Project to the next stage of evaluation. The proposed program includes additional drilling, Mineral Resource update, metallurgical and hydrometallurgical testwork, geotechnical and hydrogeological investigations, tailings and infrastructure studies, environmental baseline and assessment work, and completion of a pre-feasibility study.

The estimated cost of the recommended program is $85.02 million, including contingency, as summarized in Table 1-8.

Table 1-8: Estimated Cost for Proposed Recommendations

Description	Cost ($M)
Exploration and Resource Upgrading	
PFS Infill & Geotechnical Drilling	14.00
Regional Exploration & Target Testing	15.00
Resource Estimate Update (MRE)	0.45
Metallurgy and Process Design	
Comminution & Ore Property Testing	1.50
Metallurgical PFS (Flotation & Cobalt)	2.50
Hydrometallurgical PFS (Scandium Recovery & SX)	2.25
Engineering and Infrastructure	
Geotechnical, Hydrogeological, and Terrain studies.	7.00
Geochemical Testing & Acid-Rock Drainage/ML Modelling	1.20
Site investigations (TSF, WSF, Borrow)	4.50
Hydrological, Meteorological & Glacier	1.30
Infrastructure studies (Bridge, Road, Power)	6.50
Environmental, Social and Permitting	
Social, Archaeological & Heritage	1.80
Indigenous Participation Negotiations	1.20
Environmental baseline studies	5.00
Environmental assessment studies	2.50
Project Management and Studies	
Pre-feasibility Study Report & QPs	3.50
Product Marketing & Logistics Study	0.40
Tailings Trade-off Study (Dry-stack)	0.25
Subtotal	**70.85**
20% Contingency	**14.17**
Total	**85.02**

2 INTRODUCTION

Doubleview Gold Corp. ("Doubleview" or the "Issuer") requested that Mineit Consulting Inc. (Mineit) prepare this report, titled *National Instrument 43-101 Technical Report, Preliminary Economic Assessment and Updated Mineral Resource Estimate for the Hat Project, Northwest British Columbia, Canada* (this Technical Report or PEA), with an effective date of February 4, 2026, and dated April 10, 2026.

2.1 Issuer

Doubleview is a mineral exploration company incorporated under the laws of the Province of British Columbia, Canada, with its head office in Vancouver, British Columbia.

Doubleview is engaged in the acquisition, exploration, and development of mineral properties in Canada. Its principal asset is the Hat Project, located in northwestern British Columbia, which is the subject of this Technical Report. The Project comprises a large-scale polymetallic copper-gold-silver-cobalt porphyry deposit.

Doubleview holds a 100% interest in the Project, subject to underlying royalties and other encumbrances described elsewhere in this Technical Report.

Doubleview is a reporting issuer in Canada, and its common shares are listed on the Toronto Stock Exchange (TSX) Venture Exchange, the OTCQB Venture Market and the Frankfurt Stock Exchange. Doubleview's common shares trade on the TSX Venture Exchange under the symbol DBG, on the OTCQB under DBLVF, and on the Frankfurt Stock Exchange under 1D4.

2.2 Terms of Reference

The purpose of this Technical Report is to present the results of a PEA for the Hat Project (the Project), in British Columbia, Canada.

The PEA evaluates the potential technical and economic viability of the Project based on Mineral Resources, conceptual mine planning, process design, infrastructure concepts, capital and operating cost estimates.

This Technical Report was prepared in accordance with National Instrument 43-101 (NI 43-101), Standards of Disclosure for Mineral Projects and Form 43-101F1 and is intended to support public disclosure by Doubleview.

The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be classified as Mineral Reserves. There is no certainty that the results of the PEA will be realized.

All scientific and technical information contained herein is current as of the effective date, unless otherwise stated.

The assessment is based on assumptions, parameters, and methodologies considered reasonable at the time of preparation, but which are inherently subject to uncertainties.

2.3 Qualified Persons

This Technical Report has been prepared by, or under the supervision of, the qualified persons (QP) listed in Table 2-1, as defined in NI 43-101. Each QP is responsible for the preparation of, or has reviewed and approved, the sections of this Technical Report. Each Qualified Person has sufficient experience relevant to the style of mineralization, deposit type, and activity described in this Technical Report to qualify as a QP as defined in NI 43-101. Dr. Gilles Arseneau, P.Geo., provided peer review and scientific and technical advice on those sections prepared by Tomasz Wawruch.

Table 2-1: Qualified Persons, Affiliations, and Reporting Responsibilities

Name	Company	Responsibility
Shervin Teymouri, P.Eng.	Mineit	Sections 1, 2, 3, 4, 5, 16, 19, 21, 22, 23, 24, 25, 26, 27
Tomasz Wawruch, FAusIMM.	Mineit	Sections 6 to 12, 14, and corresponding subsections in 1, 25, 26, 27.
Marinus André (André) de Ruijter, M.Sc. (Met Eng.), P.Eng.	Mineit	Sections 13.1 to 13.2, 17.1 to 17.6, and corresponding subsections in 1,21, 25, 26, 27.
Andrew Carter, Eur.Ing. C.Eng. MIMMM QMR, MSAIMM, SME	Magister Metallurgy	Section 13.3 and 17.7, and corresponding subsection in 1, 21, 25, 26, 27.
Franky Li, P.Eng.	EMM Consulting Pty.	Section 18.2.8 to 18.3.6, 20, and corresponding subsections in 1, 21, 25, 26, 27.
Jayesh Rami, P.Eng.	Sacre-Davey	Section 18.1 to 18.2.7, and corresponding subsections in 1, 25, 26, 27

2.4 Work Program

The scope of work completed for this Technical Report included the review and integration of available geological, exploration, metallurgical, engineering, and cost data provided by Doubleview, along with publicly available information where applicable.

The work program comprised geological modelling and Mineral Resource estimation, metallurgical review and process design development, conceptual open pit mine planning, and the development of supporting infrastructure concepts. Capital and operating cost estimates were prepared to a preliminary level of accuracy consistent with a preliminary economic assessment, with capital costs developed to an Association for the Advancement of Cost Engineering (AACE) Class 5 estimate level, based on limited engineering definition and appropriate for early-stage evaluation. A discounted cash flow analysis was completed to assess the potential economic performance of the Project.

In addition, the work included a site inspection, review of environmental and permitting considerations, and identification of key risks, opportunities, and areas requiring further work to advance the Project.

The level of engineering, design, and cost estimation is consistent with that required for a preliminary economic assessment and is based on available information and reasonable assumptions at the time of preparation.

2.5 Basis of Technical Report

This Technical Report has been prepared by Mineit for Doubleview to support public disclosure of a PEA for the Hat Project. The Report is based on information, data, and technical inputs provided by Doubleview, including geological, metallurgical, engineering, and cost data, as well as publicly available information where appropriate.

The Qualified Persons have reviewed the information for reasonableness and internal consistency but have not independently verified all underlying data. The conclusions and opinions expressed herein are based on the information available at the time of preparation and on assumptions considered reasonable by the Qualified Persons.

2.6 Qualifications

Mineit is an independent engineering and consulting firm based in Vancouver, British Columbia, providing services to the mining industry, including geological modelling, mine planning, metallurgical process development, infrastructure design, and economic evaluation.

This Technical Report was prepared by, or under the supervision of, Qualified Persons (QPs) employed by Mineit and contributing independent consultants. Each QP is a member in good standing of a recognized professional association and has relevant experience in the style of mineralization, deposit type, and activities undertaken in this Technical Report, as required under National Instrument 43-101.

The QPs responsible for the preparation of this Technical Report are listed in Section 2.3, together with their areas of responsibility.

2.7 Personal Inspection

On October 2–3, 2025, Tomasz Wawruch visited the site of Doubleview's Hat Project, with its senior exploration manager, Reza Hamedi.

During the visit, there were insightful discussions with Mr. Hamedi on the Project's exploration activities, collection, validation, handling, and sampling. Tomasz Wawruch was also able to visit some of the historical, current, and planned drill-hole locations, as well as the core logging and storage facilities, where they discussed the Hat Project's geological logging procedure. The team was also given an aerial view of the Project site, including the proposed processing plant, and tailings and waste storage facilities locations.

Overall, Mineit's team observed that the exploration activities, site geological data collection and integrity, sampling, and site quality assurance and quality control protocols (QA/QC) were performed in a

systematic manner, and that the work performed at the Hat Project site was carried out according to industry standards.

2.8 Sources of Information and Data Reliance

In preparing this Technical Report, the QPs have relied upon information, data, and interpretations provided by Doubleview, including geological databases, drilling and assay data, metallurgical testwork, cost information, and project documentation, as well as publicly available information, including a technical report prepared by Micon International Limited (Micon, 2024).

The QPs have reviewed the Doubleview information for reasonableness and internal consistency, but have not independently verified all underlying data. The accuracy of the results and conclusions presented herein are dependent on the completeness and accuracy of the information Doubleview has supplied. Any Mineral Resource estimates disclosed in reports prepared prior to the effective date of this Technical Report are considered historical in nature unless otherwise stated and should not be relied upon.

Further details regarding reliance on other experts are provided in Section 3.

2.9 Declaration

The Qualified Persons responsible for this Technical Report have reviewed and approved the scientific and technical information contained herein. Each Qualified Person has sufficient experience relevant to the style of mineralization, deposit type, and activities undertaken to qualify as a Qualified Person as defined by National Instrument 43-101.

This Technical Report has been prepared in accordance with National Instrument 43-101—Standards of Disclosure for Mineral Projects and Form 43-101F1. The Qualified Persons are independent of Doubleview, except where otherwise disclosed, and have exercised reasonable professional judgment in the preparation of this Technical Report.

The Qualified Persons have read National Instrument 43-101 and Form 43-101F1, and this Technical Report has been prepared in compliance therewith.

2.10 Units, Currency, and Geographic Coordinate System

Unless otherwise stated, all units of measurement in this Technical Report are expressed using the Système international d'unités (International System of Units—the metric system). All monetary values are reported in Canadian dollars ($) unless otherwise noted. Precious metal weights are given in troy ounces (oz). Geographic locations are reported in the Universal Transverse Mercator (UTM) coordinate system using the NAD 83 datum.

3 RELIANCE ON OTHER EXPERTS

In preparing this Technical Report, the QPs have relied upon information, data, reports, interpretations, and representations provided by Doubleview and its representatives, as well as by other professional advisors engaged by the Company, as outlined below. Such reliance is consistent with industry practice and is permitted under NI 43-101—Standards of Disclosure for Mineral Projects.

3.1 Legal Title, Agreements, and Royalties

The QPs have not conducted an independent investigation of the legal title to the Project, including mineral tenure, surface rights, royalties, back-in rights, option agreements, encumbrances, or other legal interests. Doubleview has provided information regarding property ownership, mineral claims, underlying agreements, royalties, and related legal matters, and, where applicable, summarized from legal documents prepared by Doubleview's legal counsel.

The QPs have relied upon this information as being accurate and complete, and express no opinion as to the validity, enforceability, or priority of any legal agreements, claims, or interests.

3.2 Environmental, Permitting, and Social Matters

Doubleview or its external environmental and legal consultants have provided information relating to environmental baseline studies, permitting status, regulatory requirements, community engagement, Indigenous consultation, and social considerations. The QPs have relied on such information without independent verification, and assume that all required permits and approvals can be obtained within the timelines and conditions contemplated. QPs assume no responsibility for any matters related to environmental liabilities, permitting risks, or social agreements beyond reliance on the information provided.

3.3 Reliance on Company-Provided Technical Information

The QPs have relied on geological, exploration, drilling, assay, metallurgical, geotechnical, hydrogeological, infrastructure, cost, and operating data Doubleview and its consultants have supplied. While the QPs have reviewed such information for reasonableness and internal consistency within the context of a PEA-level study, they have not independently verified all underlying data.

The accuracy of interpretations, analyses, conclusions, and economic results presented in this Technical Report is materially dependent on the completeness and accuracy of the information Doubleview provided.

3.4 Limitations of Reliance

This PEA is conceptual in nature and includes Inferred Mineral Resources, which are considered too speculative geologically to have economic considerations applied to them that would enable them to be classified as Mineral Reserves.

The QPs expressly disclaim responsibility for any errors or omissions resulting from reliance on information Doubleview or third-party experts have supplied, provided such reliance was reasonable and consistent with NI 43-101.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Hat Project is in northwestern British Columbia, within the Atlin Mining Division. The property lies north of Hatchau Lake, approximately 95 km southwest of Dease Lake and 40 km north of Telegraph Creek (Figure 4-1).

The southwest part of the mineral tenures is positioned at approximately UTM Zone 9, 345,211 and 6,453,290 (latitude 58°12’ N, longitude 131°35’ W), corresponding to National Topographic Service map sheets 104J/12, 104J/13, 104J/22, and 104J/23.



Figure 4-1: Hat Project Location

4.2 Ownership and Property

The Hat Project comprises 16 mineral tenures totalling 13,823.09 ha. Of these, 13 tenures are recorded to Doubleview, and 3 tenures are recorded to Farshad Shirvani (Table 4-1). Figure 4-2 shows the mineral tenure locations.

Table 4-1: Summary of Mineral Tenures Comprising the Hat Project

Mineral Tenure	Mineral Claim Name	Owner	Mineral Claim Issue Date	Mineral Claim Expiry Date	Area (ha)	Annual Work Requirement ($)
501290	Hat	Doubleview	2005-01-12	2035-11-29	204.53	4,090.60
507814			2005-02-24	2035-11-29	255.74	5,114.80
511709			2005-04-26	2035-11-29	324.01	6,480.20
515549			2005-06-29	2035-11-29	187.59	3,751.80
515550			2005-06-29	2035-11-29	715.87	14,317.40
896518	Top Hat		2011-09-11	2035-11-29	204.51	4,090.20
1019552	Hat North 1		2013-05-16	2035-11-29	459.99	9,199.80
1019554	Hat North 21		2013-05-16	2035-11-29	664.19	13,283.80
1019557	Hat North 3		2013-05-16	2035-11-29	545.00	10,900.00
1057970	The Ostensoe		2018-01-26	2035-11-29	1,652.12	24,781.80
1118136	PEA 1		2024-12-16	2035-11-29	1,088.80	5,444.00
1118137	PEA 2		2024-12-16	2035-11-29	1,700.72	8,503.60
1118138	PEA 3		2024-12-16	2035-11-29	1,700.13	8,500.65
1115584	Hat TW1	Shirvani, Farshad	2024-09-04	2035-11-29	1,260.39	6,301.95
1115585	Hat NW 2	Shirvani, Farshad	2024-09-04	2035-11-29	1,327.63	6,638.15
1115586	Hat NE2	Shirvani, Farshad	2024-09-04	2035-11-29	1,531.87	7,659.35
Total					**13,823.09**	**139,058.10**

Source: Government of British Columbia (*Mineral Titles Online*) (2025, Nov 28).

4.3 Mineral Tenures and Encumbrances

The original Hat mineral claims (501290, 507814, 511709, 515549, and 515550) were initially staked by Erik Ostensoe and Thomas E. Lisle in 1993 and 1994 as Four Post claims (Figure 4-2). With the implementation of Mineral Titles Online (MTO) in British Columbia, these claims were converted to the online registration system in 2005. In 2011, Doubleview acquired the original Hat claims under an option agreement. Following the agreement's execution, Doubleview staked additional contiguous online claims to expand the property. In 2017, the option agreement was fully exercised, and the claims were transferred to Doubleview.

The Hat Property is subject to a 2% net smelter return (NSR) royalty, effective upon commencement of commercial production. The option that allows Doubleview to purchase 1% of the NSR for $1,500,000 on or before the sixth anniversary of the agreement has expired.

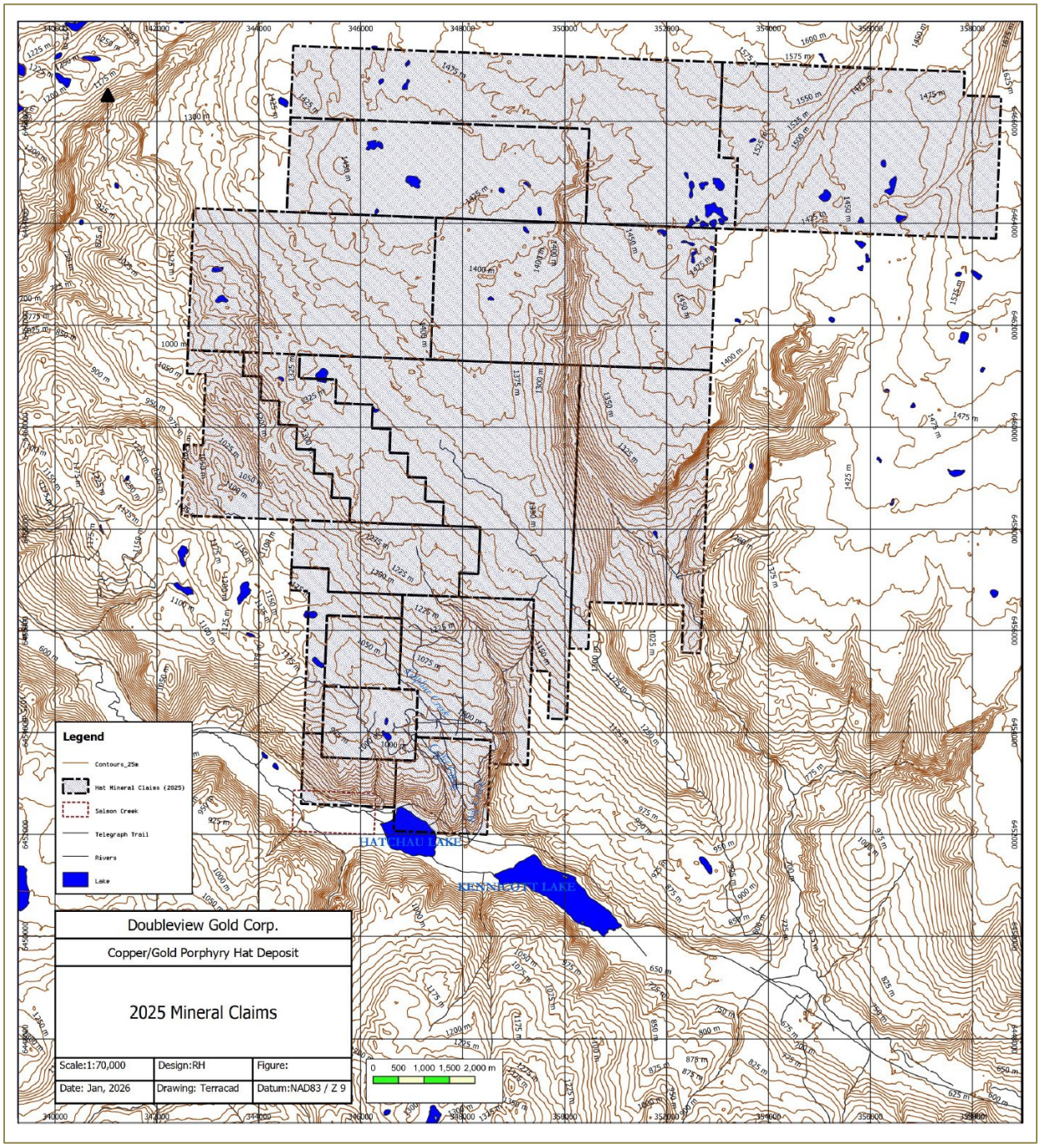


Figure 4-2: Mineral Tenures Comprising the Hat Project

The Hat Project mineral claims are governed by the *Mineral Tenure Act* and the *Mines Act* of British Columbia and are administered by the Mineral Titles Branch. Historically these functions were carried out under the Ministry of Energy, Mines and Low Carbon Innovation; following provincial reorganization in November 2024, responsibility for mining and mineral regulation is located with the Ministry of Mining

and Critical Minerals (MCM). Exploration and development activities remain subject to statutory permitting requirements: most mechanized or ground-disturbing work requires a *Mines Act* Notice of Work and related permits, and assessment work must be registered via Mineral Titles Online in accordance with current guidance. The provincial Health, Safety and Reclamation Code revisions and the introduction of MineSpace for permit administration must also be considered when planning field programs.

To maintain mineral claims in good standing, the recorded holder must complete qualifying exploration and development work or make a cash-in-lieu payment through MTO, in accordance with the *Mineral Tenure Act* and *Mineral Tenure Act* regulations. All eligible work must be submitted through the MTO system within one year of completion. Once accepted, the recorded value of the work is converted into "time," which extends the expiry date of the relevant mineral claims for up to a maximum of ten years.

Work may be credited to the claim on which it was performed, to adjoining claims, or to a contiguous group of adjoining claims. A Statement of Work must be filed before the claim's anniversary date, describing the nature, timing, and value of the work completed. The minimum exploration or development work value required to maintain a mineral claim for one year is prescribed in the Mineral Tenure Act Regulation as follows:

- $5/ha for the first and second anniversary years
- $10/ha for the third and fourth anniversary years
- $15/ha for the fifth and sixth anniversary years
- $20/ha for each subsequent year.

In lieu of recording exploration or development work, a claim holder may elect to make a cash-in-lieu payment through the MTO system. The payment amount is equivalent to twice the prescribed value of work required to maintain the claim for a period of six to twelve months.

4.3.1 Verification Comment

The project claims and associated permit status are maintained in the Province of British Columbia's digital platform, MineSpace, which supports submissions, permit tracking, and inspection history. On November 5, 2025, the Qualified Person verified the claim status via the Mineral Titles module of MineSpace and confirmed all claims are in good standing (Table 4-1). Use of MineSpace ensures streamlined compliance with annual work submission requirements and assists in monitoring tenure status, amendments, and renewals.

4.4 Permitting and Environmental Liabilities

4.4.1 Work Permits

Mineral exploration activities at the Hat Project are authorized under *Mines Act* Permit No. MX-1-872 (the Permit), a multi-year area-based exploration permit issued by the MCM. The Permit was originally issued

in 2012 and has been amended and extended multiple times to accommodate successive exploration programs and is valid until March 31, 2037.

The current Permit area encompasses mineral tenures 501290, 507814, and substantial portions of 1019552, 896518, 515550, 515549, and 511709 (Figure 4-2). As of the most recent amendment, the Permit authorizes the following exploration activities: up to 40 line-km of ground geophysical surveys; 45 helicopter-supported drill sites; construction of approximately 2.63 km of new exploration trails; and continued use of the existing temporary exploration camp.

The Permit includes detailed conditions for environmental protection and reclamation of disturbed areas, as well as specific operational and safety provisions: a reclamation and surface protection plan addressing soil, vegetation, and watercourse impacts; maintenance of a Level 3 Occupational First Aid attendant (with appropriate equipment) during all field operations; implementation of a site-specific emergency response plan; and adherence to an Archaeological Chance Find Procedure.

To satisfy special permit conditions, Doubleview commissioned multiple Archaeological Impact Assessments (AIA) under the supervision of a professional archaeologist. Reports were filed with the B.C. Archaeology Branch of the Ministry of Forests (September 2024) for all proposed ground-disturbance sites. Consequently, all drill sites within the Hat Project area have been assessed or inspected by a professional archaeologist.

A Wildlife Management Plan is in place to regulate seasonal activities. Field operations are restricted to the period after October 31 and before July 16 each year to mitigate wildlife disturbance. The southern portion of the permitted area requires preparation of a salmon ecosystem management plan, to be produced by a qualified professional prior to Doubleview conducting operations in that sector.

A small segment at the southern edge of claim 515549 overlaps the Salmon Creek 3 Indian Reserve, and no work is allowed on the Reserve. Any proposed modifications to the work program, trail alignments, camp facilities, or other surface disturbances beyond those authorized require written approval from an MCM Mines Inspector under the *Mines Act*.

The permit is secured by a reclamation bond of $167,050, held by the Receiver General for British Columbia. This bond will be returned to the permit holder upon successful reclamation and inspection of all work areas, or otherwise used by the Province to complete reclamation if required.

4.4.2 First Nations

Doubleview has undertaken due diligence to confirm that title to the mineral tenures comprising the Hat Property has been properly obtained and registered with the appropriate public offices. Nevertheless, mineral tenure in British Columbia may be subject to unregistered agreements, transfers, or Aboriginal rights and title claims. Title to these mineral claims could be affected by such interests or by undetected defects.

The Hat property is within the Traditional Territories of the Tahltan Nation and the Taku River Tlingit First Nation. The closest communities are Dease Lake 95 km northeast and Telegraph Creek 40 km south of the Project.

To improve communication, Doubleview and the Tahltan Central Government (TCG) executed a Communications and Engagement Agreement in July 2021, intended to establish a formal process for dialogue.

The Hat Property also lies within the Atlin Taku Land Use Plan (*Wóoshtin wudidaa*) area jointly managed by the Taku River Tlingit First Nation and the Province of British Columbia. The Project area falls within the Hackett Camp Island Area-Specific Resource Management Zone, which "allows industrial development but contain resource management direction to maintain the specific values for which the zone was identified." The southernmost portion of the property crosses a salmon ecosystem management area. The Hackett River is recognized as high-value spring and fall grizzly bear habitat.

The Tahltan are also co-developing a land use plan with the Province of British Columbia (Tahltan Central Government, 2025), which may be enacted into law and is anticipated to provide further guidance on conservation and development objectives relevant to Project development.

Doubleview reports that its exploration programs have included Indigenous employment and contracting opportunities, and the Company intends to strengthen its Indigenous engagement strategy in accordance with the principles of the Declaration on the Rights of Indigenous Peoples Act (British Columbia, 2019) and the ongoing Mines Act permitting modernization under MineSpace.

4.4.3 Archaeological Impact Assessment

On September 5, 2024, the results of the most recent AIAs and associated field inspections completed for the Hat Project were accepted by the Archaeology Branch.

The AIA programs were carried out by Taiga Heritage Consulting Ltd. of Grande Prairie, Alberta, under Section 12 Permits issued by the Archaeology Branch in accordance with the *Heritage Conservation Act (1996).* Fieldwork was led by experienced, permitted archaeologists with extensive experience in boreal forest regions of western Canada.

The objectives of the AIA were to:

- Identify any archaeological or heritage resources within the project area
- Assess potential impacts from the proposed exploration activities
- Provide management with recommendations based on the results of the permitted fieldwork
- Recommend viable alternatives for managing any unavoidable adverse impacts.

Over the past decade, Doubleview has been committed to conducting responsible work practices which include archaeological surveys and subsurface testing. As such, the following programs have taken place:

- 2014: reconnaissance and in-field survey of the Lisle Zone, resulting in an Archaeological Overview Assessment report
- 2018 and 2020: in-field survey and shovel testing in the Lisle Zone and additional drill target areas, producing interim AIA reports for proposed drilling, as well as an AIA report

- 2022: completion of a full Archaeological Overview Assessment and AIA under *Heritage Conservation Act* Permit 2022-0470, including shovel testing and field inspection.
- 2023: mitigation excavation of site IbTv-04 under the *Heritage Conservation Act* under Permit 2022-0470 and additional in-field survey and shovel testing of the Hat claims for planned drill sites.
- 2024: issuance of formal management direction by the Archaeology Branch requiring avoidance or site alteration permitting for sites IbTv-04 through IbTv-07 and further assessment of feature SE013 if avoidance was not feasible
- 2025: application for and receipt of Site Alteration Permit 2025-0139 to cap site IbTv-04, followed by capping fieldwork completed in July 2025 and physical perimeter markers were installed at the boundary of the capped zone. Submission of the final report in January 2026 (Taiga Heritage Consulting Ltd., 2026).

The 2022 and 2023 programs (Permit 2022-0470) included more than 990 shovel tests across approximately 180 areas of archaeological potential, as well as extensive traversing and helicopter reconnaissance over the project area. In total, four archaeological sites were identified in the southern portion of the Hat Property: three represented by small lithic scatters (each ~10 m in diameter) and one by an isolated lithic find. These sites are now protected under the *Heritage Conservation Act* (HCA) by a 50 m buffer zone. The archaeological record suggests these may represent historic hunting activity or travel corridors along upland drainages.

4.4.4 Environmental Liabilities

To the best of the QP's knowledge, there are no known environmental liabilities associated with the Hat Property. Exploration activities have been conducted under *Mines Act* Permit MX-1-872, which includes an approved reclamation plan and a posted reclamation security of C$167,050 held by the Province. No legacy workings, historical waste, or contaminated sites are known within the property boundaries. All current environmental obligations are limited to the permitted exploration disturbances and are fully bonded under provincial requirements.

4.5 Royalties, Back-In-Rights, and Encumbrances

The Hat Project is subject to a 2% NSR royalty, payable upon the commencement of commercial production. Other than this royalty, the QPs are not aware of any additional royalties, back-in rights, or encumbrances affecting the Hat Property.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCE, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

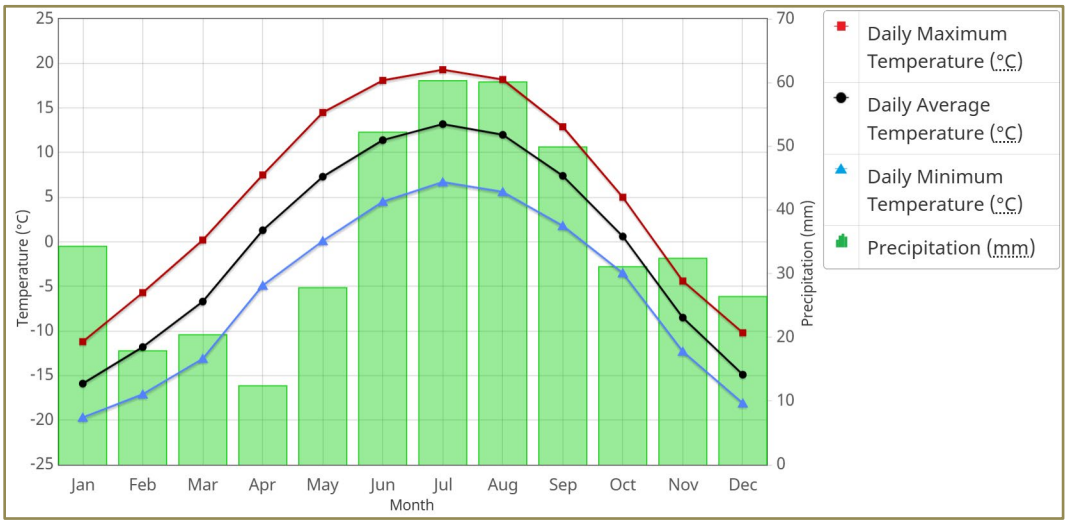
The Hat Project is in a remote region of northwestern British Columbia (Figure 4-1). The nearest serviced community is Dease Lake, approximately 95 km to the northeast, while Telegraph Creek, with a smaller population, lies roughly 40 km to the southeast.

Personnel and freight typically travel through Whitehorse, in the Yukon Territories, Smithers/Terrace, British Columbia, transiting via Dease Lake, before reaching the Golden Bear Mine Road (Muddy Lake Road) at KM 6, where a helicopter staging point is located. Helicopter transport is presently required for moving personnel, supplies, and drill equipment between the staging area and the Hat camp.

The Technical Report notes the presence of ATV trails connecting the camp with drill sites, though these pathways are only suitable for light equipment and seasonal use.

5.2 Climate

The Hat property is situated in northwestern British Columbia, characterized by a northern montane/subarctic climate, with long, cold winters and relatively short summers with temperature varying between -20°C and 20°C (Figure 5-1). Snowfall is significant (one to one-half metres), and weather conditions influence the logistics of exploration and potential mine development. Operational windows for surface work are seasonally restricted, and year-round mining operations would require appropriate infrastructure and winterization.



Source: Environment and Climate Change Canada, n.d. (1991–2020).

Figure 5-1: Temperature and Precipitation Graph for 1991 to 2020 Canadian Climate Normals, Dease Lake

5.3 Physiography

The Hat Property is located on the west side of the Interior Plateau physiographic province of the Canadian Cordillera, near the transition to the Coast Ranges. Elevations range from approximately 675 masl in the valley floor to about 1,500 masl in the northern portion of the mineral tenures. The southern area exhibits a steep rise from the valley to roughly 900 masl, above which the terrain becomes gently inclined toward the north. This gently sloping terrain comprises most of the Project area and has been the focus of recent exploration.

The Property lies north of Hatchau Lake and the Hackett River. Multiple small streams cross the tenures and drain into these waterbodies. A few small ponds are also present.

The area is forested, though with limited merchantable timber. Vegetation in the valley includes cottonwood, alder, birch, and swamp spruce. South-facing slopes support poplar and spruce, while upper slopes contain discontinuous stands of spruce and hemlock. Pine and fir occur locally. Poorly drained areas, some underlain by permafrost, contain dense willow growth.

Wildlife documented in the area includes moose, black bear, and several fur-bearing species such as beaver, otter, and marten.

5.4 Infrastructure and Local Resources

Northwestern British Columbia has limited regional infrastructure, reflecting the small and widely distributed local population. Dease Lake, located along Highway 37 approximately 95 km northeast of the Hat Property, provides essential services including accommodations, a general store, postal services, and a small hospital. Telegraph Creek, with a population of roughly 100, lies about 40 km to the southeast.

Personnel and freight typically travel through Whitehorse, Smithers or Terrace and then through Dease Lake before mobilizing to the Project area. Access to the site is currently by helicopter from a staging point at KM 6 on the Golden Bear Mine Road (Muddy Lake Road), which branches from the Telegraph Creek Road approximately 81 km from Dease Lake. The Golden Bear Mine Road leads to a former mine site located about 40 km northwest of the Property. Helicopter transport is used to move staff, supplies, and drilling equipment between the staging point and the on-site camp.

The Property contains a temporary exploration camp capable of supporting a drill crew. Facilities include a cookhouse, washhouse, bunkhouses, office, first-aid facility, and a nearby core shack. Power is supplied by generators. Core processing infrastructure—such as logging tables, core photography stations, and saws—is located adjacent to the camp. ATV paths connect the camp with drill sites, and helicopter support is required for drill moves.

In recent years, the Golden Triangle region has seen substantial mining and infrastructure development, including expansions of road access, construction of new mine sites, and commissioning of processing and logistics facilities. A key development has been the establishment and subsequent upgrades of the Northwest Transmission Line, which has enabled large-scale operations such as Red Chris, Brucejack, and others to connect to the provincial power grid. Beyond the existing regional transmission network, BC

Hydro has identified the proposed North Coast Transmission Line as a potential future enhancement to power infrastructure in northwest British Columbia. While this development may improve regional access to electrical power over the longer term, the Project's assumption of grid power supply remains preliminary and would require capital investment in transmission and site connection infrastructure. Accordingly, power supply is considered a material project risk that will require further evaluation in subsequent study phases and is discussed in greater detail in later sections of this Technical Report.

The available area within the mineral tenures appears adequate to accommodate future mine infrastructure; however, detailed assessments of land suitability have not yet been completed. Water sources are present within the Project area, but further baseline studies are required to confirm long-term availability and to evaluate potential constraints related to salmon-bearing streams.

6 HISTORY

6.1 Exploration History of the Hat Project

The exploration history summarized in this section is derived from multiple publicly available sources, including British Columbia Assessment Reports, historical claim records, and Doubleview Gold Corp.'s technical disclosures. Early reconnaissance work completed between 1969 and 1971 by Colorado Corporation and Skyline Explorations Ltd. is documented in the Skyline Project geochemical report (P. H. Sevensma Consultants Ltd., 1970). The 1985 restaking of the Moon 1–4 claims and associated multi-element soil and rock sampling programs are drawn from Lisle (1985). Lisle and Ostensoe (2008) undertook systematic prospecting, mapping, and soil geochemical surveys between 1994 and 2011 supported by the Hat claims assessment report. Exploration activities completed after Doubleview optioned the property in 2011—including drilling, geophysical surveys, relogging programs, and geological reinterpretation—are based on Doubleview's NI 43-101 technical reports, corporate news releases, and internal project records.

6.1.1 Early Prospecting (1930s–1960s)

Although informal prospecting reportedly occurred in the 1930s, the earliest documented mineral exploration in the Hat Project area began in 1963, when independent prospector John Hoey conducted work in the Hatchau Lake valley. Hoey reported strong gold values from grab samples collected near Big Creek, including assays approaching one ounce per ton gold. His work was supported by a consortium of Eastern Canadian mining companies, with Dr. Robert Seraphim acting as their technical liaison.

In 1964, Newmont Mining Corp. identified a large, high-intensity magnetic anomaly in the Hat area through airborne geophysical surveys, marking the first major geoscientific dataset over the property.

6.1.2 Systematic Exploration (1969–1980)

Between 1969 and 1970, Colorado Corporation conducted the first comprehensive exploration program, collecting 432 soil samples north of Hatchau Lake (Sevensma, 1970). Results showed background copper values near 100 ppm and anomalous values up to 3,000 ppm Cu, with localized high-grade mineralization (up to 4.7% Cu and 0.3 oz/t Au) along Big Creek.

In 1971, Skyline Explorations Ltd. re-staked the area as the "OH" claims and collected 293 additional soil samples (Sevensma, 1970). Chalcopyrite mineralization was noted near the diorite–volcanic contact in the southwest portion of the modern Hat concession.

In 1976, United Cambridge Mines Ltd. undertook prospecting, mapping, and soil sampling across the Hackett River valley, revisiting earlier showings such as Copper Creek and Polar Creek. Hand trenching and chip sampling were completed at the Hoey discovery area.

From 1978 to 1980, Utah Mines Ltd. conducted the most extensive early exploration program, including:

- 1,300 soil samples
- 122 line-km of induced polarization and ground magnetics
- Detailed geological mapping
- ~200 m of bulldozer trenching.

Copper anomalies ranged from 300 to 3,000 ppm, with localized values up to 0.5% Cu. Utah Mines later dropped the claims due to depressed metal prices (Utah Mines Ltd., 1979).

6.1.3 Re-staking and Renewed Exploration (1980–2011)

The claims were re-staked in 1985 as the Moon claims for United Cambridge. Soil and rock sampling programs expanded the geochemical dataset to include gold, molybdenum, lead, zinc, silver, cobalt, arsenic, and antimony (Lisle, 1985). Notable results included:

- 2% Cu and 6,600 ppb Au in soil
- 1.96 ppm Au over 0.5 m in rock chip samples
- 4.8% Cu in rock samples.

In 1987, airborne magnetic and very low-frequency (VLF)-electromagnetic (EM) surveys (202.1 km) were flown, identifying strong magnetic responses across the northern half of the property. Additional geochemical surveys were completed in 1988, although proposed trenching near Hatchau Lake was denied pending a Heritage Resource Impact Assessment. The assessment identified areas of high archaeological potential along the Hackett River corridor but recorded no new archaeological sites.

In 1990, the Moon claims were again re-staked, and limited rock and soil sampling returned values up to 0.087 oz/t Au and 0.8% Cu from float samples.

In 1994, geologists Erik Ostensoe and Thomas E. Lisle staked the Hat and Bob claims, forming the foundation of the modern Hat Project. Over the next 17 years, they conducted systematic prospecting, mapping, and sampling, gradually developing the geological model that recognized the potential for a significant porphyry copper-gold system (Lisle & Ostensoe, 2008).

In 2011, Doubleview Capital Corp. (now Doubleview Gold Corp.) optioned the Hat claims from Ostensoe and Lisle, marking the transition to modern exploration.

6.2 Modern Exploration and Discovery (2011–Present)

Doubleview initiated greenfield drilling in 2013–2014, leading to the discovery of the Hat alkalic porphyry system. Subsequent drilling expanded the mineralized footprint to approximately 2.5 × 2.5 km, with mineralization remaining open in all directions and at depth.

The Hat deposit is now recognized as a polymetallic system containing copper, gold, cobalt, silver, and scandium. Metallurgical testwork completed for the Hat Project confirms that scandium extraction is technically feasible (see Section 13).

As of 2025:

- Drilling continued to expand the Mineral Resource envelope.
- An updated Mineral Resource estimate was completed.
- Metallurgical testwork was conducted by multiple accredited laboratories.
- The project advanced toward completion of a PEA (this Technical Report).

6.3 Historical Mineral Resource Estimates

The most recently filed Mineral Resource estimate for the Hat Property is Micon International Limited's Mineral Resource estimate, disclosed in *NI 43-101 Technical Report for the 2024 Mineral Resource Estimate on the Hat Project, Atlin Mining Division, British Columbia, Canada* (Micon, 2024), with an effective date of July 17, 2024. This estimate established the initial Indicated and Inferred Mineral Resources for the Project based on drilling, geological interpretation, density measurements, and QA/QC procedures available at that time.

Since the completion of the 2024 Mineral Resource estimate, Doubleview has undertaken additional drilling, updated geological modelling, and refinement of the structural and alteration framework. These new data have been incorporated into a revised Mineral Resource model that supports the mine planning and economic analyses presented in this PEA. The updated model reflects:

- Inclusion of drill holes completed after the effective date of the July 2024 Mineral Resource estimate
- Revised lithological, structural, and mineral domain interpretations
- Updated density assignments and geostatistical parameters
- Re-estimation of grades within the updated mineralized domains.

The updated Mineral Resource model used in this PEA supersedes the block model used in the 2024 Micon Mineral Resource estimate for the purpose of mine design and economic evaluation.

See Section 14 for a summary of differences between the 2024 Micon Mineral Resource estimate and the updated 2026 PEA Mineral Resource model, including differences in tonnage, grade distribution, domain geometry, and classification.

6.4 Production History

No historical mining or production has occurred on the Hat Project.

6.5 Hat Project Exploration Timeline

1930s

Informal prospecting reported in the Hatchau Lake and Big Creek valleys.

↓

1963

John Hoey conducts first recorded exploration; high-grade Au samples reported near Big Creek.

↓

1964

Newmont Mining Corp. identifies large high-intensity magnetic anomaly via airborne geophysical survey.

↓

1969–1970

Colorado Corporation completes first systematic soil geochemistry (432 samples; Cu anomalies up to 3,000 ppm). Sevensma (1970)

↓

1971

Skyline Explorations Ltd. collects 293 soil samples; identifies Cu–Mo anomalies and chalcopyrite near diorite–volcanic contact. Sevensma (1970)

↓

1976

United Cambridge Mines Ltd. conducts prospecting, mapping, trenching, and revisits early showings (Copper Creek, Polar Creek).

↓

1978–1980

Utah Mines Ltd. executes major program:

- >1,300 soil samples
- 122 line-km IP and ground magnetics
- Detailed mapping
- Bulldozer trenching (~200 m)

↓

1985
Moon claims re-staked; multi-element soil and rock sampling identifies high Cu and Au values at Hoey showing Lisle (1985).
↓
1987
Airborne magnetic and VLF-EM surveys flown (202.1 km).
↓
1988
Additional geochemical surveys; proposed trenching deferred pending Heritage Resource Impact Assessment.
↓
1990
Rock and soil sampling returns up to 0.087 oz/t Au and 0.8% Cu.
↓
1994
Thomas E. Lisle and Erik Ostensoe stake the Hat and Bob claims, establishing the modern Hat Project.
↓
1994–2011
Systematic prospecting, mapping, sampling, and geological interpretation; recognition of alkalic porphyry potential (Lisle & Ostensoe, 2008).
↓
2011-Present
Doubleview Capital Corp. options the Hat claims, initiating modern exploration and project development. Systematic diamond drilling commenced in 2013 and has continued to the present day.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

Regionally, the Stuhini Group forms the basement to the overlying Hazelton Group, marking a transition from Late Triassic arc volcanism to Early Jurassic arc-related volcanic and sedimentary environments. The Stuhini Group is also closely associated with several important mineral deposit types in the region, including porphyry copper–gold systems, skarns, and volcanic massive sulphide (VMS)-style mineralization, reflecting its role as both a host and a structural-stratigraphic control on mineralizing events.

In northwestern B.C., the Stuhini and overlying Hazelton Groups form the principal host rocks of the Golden Triangle, a prolific district containing major copper–gold porphyry, epithermal, and VMS deposits. Notable examples include the Red Chris and KSM copper–gold porphyry systems, the Brucejack and Scottie epithermal gold deposits, and the Eskay Creek and Granduc VMS deposits, all of which are spatially associated with Stuhini–Hazelton stratigraphy. These deposits illustrate the metallogenic fertility of the Stuhini arc and its capacity to host large-scale mineralizing systems. These deposits occur within the same Stikine Terrane volcanic arc, the same tectonic setting as the Hat Project area.

The Stuhini Group represents a thick, regionally extensive Late Triassic volcanic and sedimentary arc succession that forms a major component of the Stikine terrane in northwestern British Columbia. In the Iskut River area, Anderson (1994) describes the Stuhini Group as comprising predominantly volcanic and volcaniclastic rocks, interlayered with lesser sedimentary units, deposited in an island-arc to arc-proximal marine basin.

Volcanic rocks are dominated by andesitic to basaltic flows, breccias, and tuffs, reflecting a calc-alkaline arc magmatic affinity. These units commonly display auto brecciation, flow banding, and variable degrees of hydrothermal alteration. Volcaniclastic strata include lapilli tuffs, crystal tuffs, and heterolytic volcanic breccias, indicating active explosive volcanism and synvolcanic mass-flow processes.

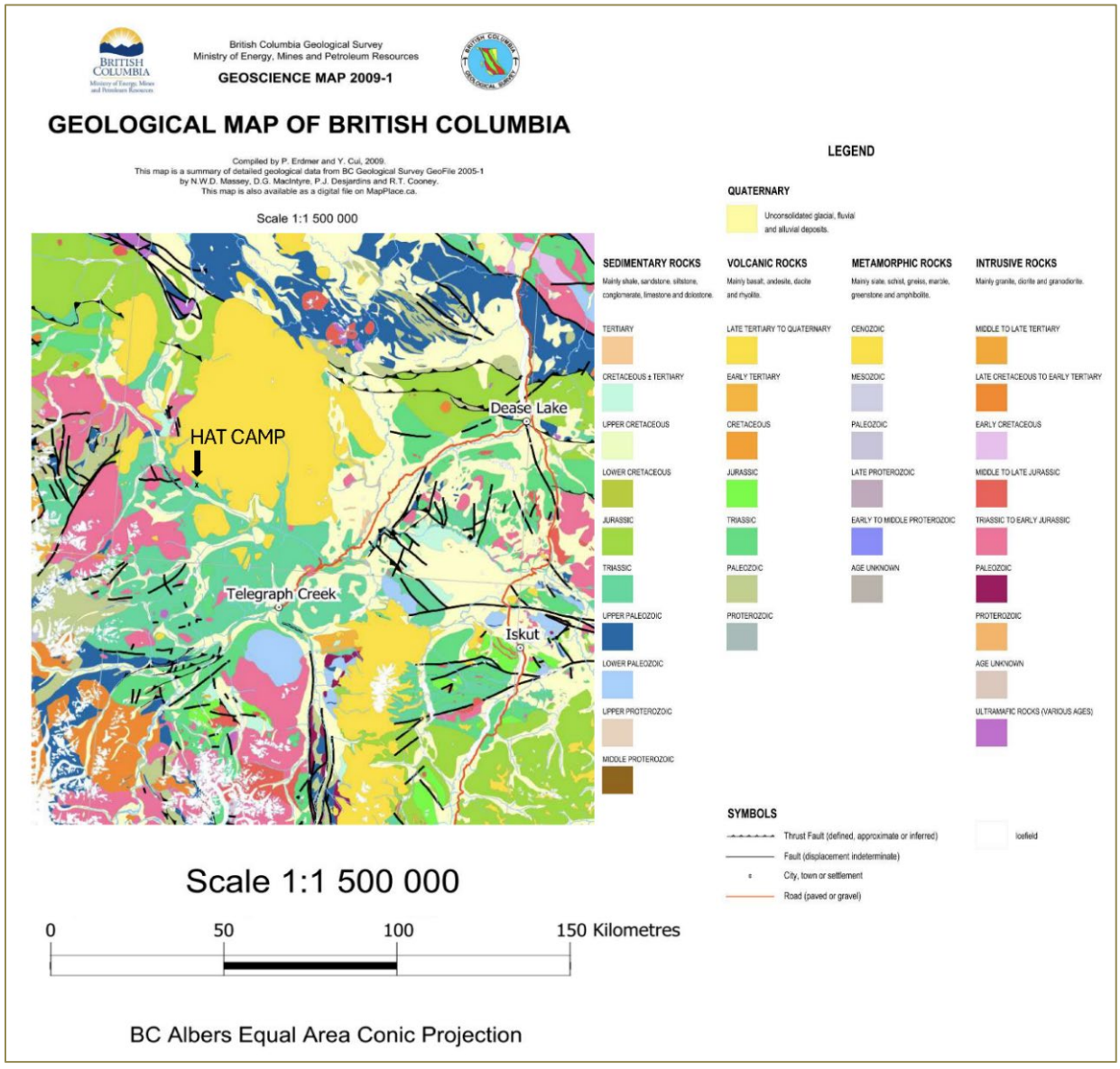
Sedimentary components of the Stuhini Group include argillite, siltstone, greywacke, and minor limestone, representing deposition in quieter marine settings peripheral to the main volcanic centers. Local conglomerates and debris-flow deposits record periods of tectonic instability and uplift within the arc.

Anderson (1994) emphasizes that the Stuhini Group in the Iskut River area is structurally complex, with stratigraphy disrupted by multiple phases of deformation, faulting, and intrusive activity. The group is intruded by Late Triassic to Early Jurassic plutons of the Stikine and Hazelton intrusive suites, which locally overprint primary textures and complicate stratigraphic relationships.

7.2 Property Geology

The Hat Project is situated on the northern flank of the Stikine Arch, within the Intermontane Physiographic Province of the Canadian Cordillera. It lies near the eastern margin of the Coast Crystalline

intrusions, part of the Insular and Coastal Physiographic Sub Province (Figure 7-1). Telegraph Creek sits on the western side of the Stikine Plateau, east of the Coast Belt. Dease Lake lies further northeast but still within the Stikine Terrane and Intermontane Belt. The "Stikine Arch" is a prominent structural feature superimposed on the regional geological domain known as Stikinia, which comprises Triassic to Jurassic volcanic, sedimentary, and volcaniclastic formations. Granitic, mafic, and ultramafic plutons are widespread throughout Stikinia, along with Tertiary-age volcanic edifices.



Source: Erdmer and Cui (2009).

Figure 7-1: Regional Geology—Telegraph Creek and Dease Lake Area

The Hat property has been systematically prospected and mapped. Outcrop exposure is limited, with the best exposure occurring along stream-channel margins. Much of the property is covered by swamps and dense willow thickets, where bedrock is not exposed.

The principal rock units are assigned to the Upper Triassic Stuhini Group, comprising a highly variable assemblage of tuffaceous and volcaniclastic rocks. The Stuhini Group is broadly analogous to the Nicola and Takla Groups—formations of similar age and composition that host significant copper and other mineral deposits elsewhere in British Columbia. This topic is discussed in Section 7.1.

Multiple facies of the Stuhini Group are distinguished in the Iskut River area by age, abundance, and composition of the volcanic rocks (e.g., bimodal vs. mafic and intermediate composition), and nature of the intercalated marine strata (e.g., limestone vs. medium- to fine-grained siliciclastic rocks). Coarse-grained clinopyroxene (salite)-phyric flows and tuff make up the mafic volcanic rocks; sparsely feldspar-phyric dacitic crystal-lithic tuff is typical of the felsic volcanic rocks.

The southeastern portion of the Hat property is underlain by brown to black, sandy-textured, thin- to coarse-bedded argillite and wacke, while the remainder consists predominantly of tuffaceous to massive andesite.

The processed magnetic data discussed in Section 9.2.3 delineate the intrusive architecture and associated demagnetized zones, emphasizing the Lisle Zone as the principal exploration target, where chalcopyrite and pyrite mineralization occurs with magnetite in chloritized andesite, coarse- to medium-grained gabbro, diorite, monzonite, and transitional zones between these lithologies. Fracturing is pervasive and appears to be controlled by one or more prominent northwest-trending fault structures.

7.3 Mineralization

Mineralization at the Hat deposit is dominated by chalcopyrite, pyrite, and magnetite, which occur within both the tuffaceous meta-andesite formations and the gabbroic intrusive units. These minerals define the principal sulphide and oxide assemblages associated with copper and gold enrichment.

Copper and gold are the primary economic metals, and analytical data show a consistent and strong association between them. This relationship indicates that gold is closely linked to the sulphide system, most likely incorporated within or immediately adjacent to chalcopyrite and pyrite, including along microfractures, vein selvages, and fine-grained sulphide aggregates.

Chalcopyrite is the dominant copper mineral, ranging from very fine disseminations to medium-coarse aggregates. It may occur intergrown with pyrite or as centimetre-scale bands with minimal associated sulphide. In some sheared, dark-andesite intervals, copper values are present even where sulphides are visually obscure, suggesting that part of the copper inventory is hosted in very fine or altered mineral phases.

Pyrite is abundant but irregularly distributed, forming both massive bands up to 40 cm thick and intimate intergrowths with chalcopyrite. Its variability reflects both primary mineralization and later structural modification.

Magnetite is widespread and varies from fine disseminations in weakly metamorphosed andesite to semi-massive sections tens of centimetres thick in chloritized or epidotized andesite. It is also a substantial component of the gabbro. The distribution of magnetite supports the interpretation of a strong iron-rich magmatic or hydrothermal contribution to the mineralizing system.

Although copper and gold are the principal metals of interest, silver and cobalt are consistently detected in assays and show a tendency to increase outward from the central copper–gold zones. While no diagnostic silver- or cobalt-bearing minerals have been identified visually, their geochemical patterns suggest that these elements are more prominent in distal or structurally influenced domains, consistent with broader metal zonation trends observed in similar intrusive-related systems.

Scandium is also present within the deposit. Scandium distribution at the Hat Project is primarily controlled by lithology and alteration, with the highest concentrations occurring in mafic to intermediate intrusive rocks. These units contain abundant iron–magnesium silicate minerals—notably amphibole, pyroxene, and chlorite—into which scandium substitutes during crystallization.

The most favorable scandium zones correspond to areas affected by propylitic alteration, characterized by chlorite–epidote–carbonate ± amphibole assemblages. This alteration style preserves the original silicate hosts and maintains the mineralogical framework in which scandium resides.

Lower scandium concentrations are typically associated with felsic intrusive phases, zircon- or titanite-rich domains, and strongly clay-altered or carbonate-rich zones, where scandium is either diluted by silica-rich mineralogy or incorporated into refractory accessory minerals. These lithologies also tend to show higher levels of elements such as titanium, zirconium, aluminum, and calcium, which correlate negatively with scandium abundance.

Overall, scandium is geologically linked to mafic and intermediate intrusive rocks and their propylitic halos (Kotze et al., 2022; Natural History Museum, n.d.), where iron–magnesium silicates dominate and accessory minerals containing titanium or zirconium are less abundant. These geological domains consistently show the strongest correlation with elevated scandium grades across the Hat drill-hole dataset.

The mineralization is characterized by variable but widespread development of sulphides and magnetite within both volcanic and intrusive host rocks. A well-defined copper–gold core is surrounded by a broader silver–cobalt halo, the latter distinguished primarily through geochemical patterns rather than discrete mineralogical phases. Scandium occurs at background levels within the Hat system, reflecting its natural association with mafic to intermediate igneous rocks and their iron–magnesium silicate mineral assemblages, rather than representing a distinct porphyry-style commodity.

7.3.1 Geometry and Extent of Mineralized Domains

Copper Domain

The copper domain forms a large, steeply dipping mineralized volume that extends for approximately 1.6 km along strike within the central structural–magmatic corridor of the Hat system. Cross-sectional

thickness vectors define a continuous corridor between roughly 347600 E and 348400 E, with local variations in width and geometry but a consistently traceable footprint across the full strike length.

Copper mineralization is best developed within the central portion of the system, between approximately 347600 E and 348400 E, where overlapping mineralized panels combine into a broad, coherent body shown in Figure 7-2 and Figure 7-3. In this central corridor, true thicknesses commonly range from 150 m to 250 m and locally exceed these values where multiple subparallel panels merge down-dip. From 348100 E eastward, the geometry is characterized by strong vertical persistence, with steep to subvertical continuity maintained over 300 m to 600 m of elevation.

Toward both the western and eastern flanks, the copper domain becomes narrower and more segmented. Western sections show reduced thicknesses, typically between 80 m and 250 m, with mineralization developed as a series of narrower, steeply dipping panels that taper laterally and vertically. Eastern sections exhibit a similar narrowing trend, with thickness decreasing progressively beyond approximately 348300 E and the domain ultimately closing at depths of 100 m to 150 m. These margins define the recognized geometric limits of the copper system.

Overall, the copper domain is best described as a structurally focused mineralized corridor with a thick, vertically persistent central core and progressively thinning margins toward both ends of strike. The geometry is internally variable but remains laterally continuous over approximately 1.6 km, reflecting a robust and well-defined mineralized volume consistent with the broader structural and magmatic architecture of the Hat system.

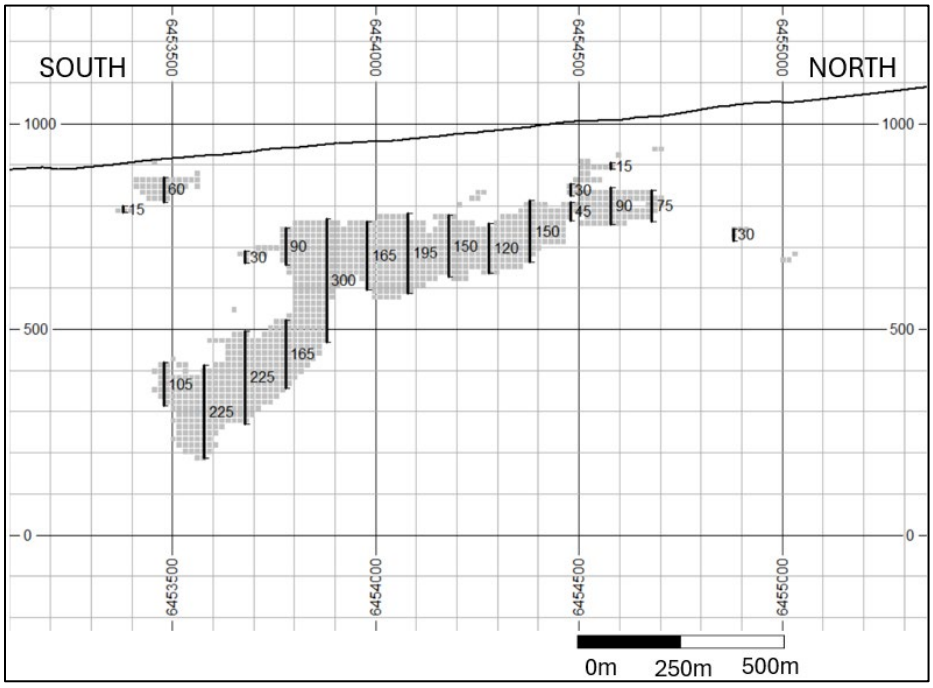


Figure 7-2: East 347700 Looking West, Cross-Section Illustrating the Geometry, Thickness Variability, and Lateral Continuity of the Copper (Cu) Domain Along the Central Structural–Magmatic Corridor

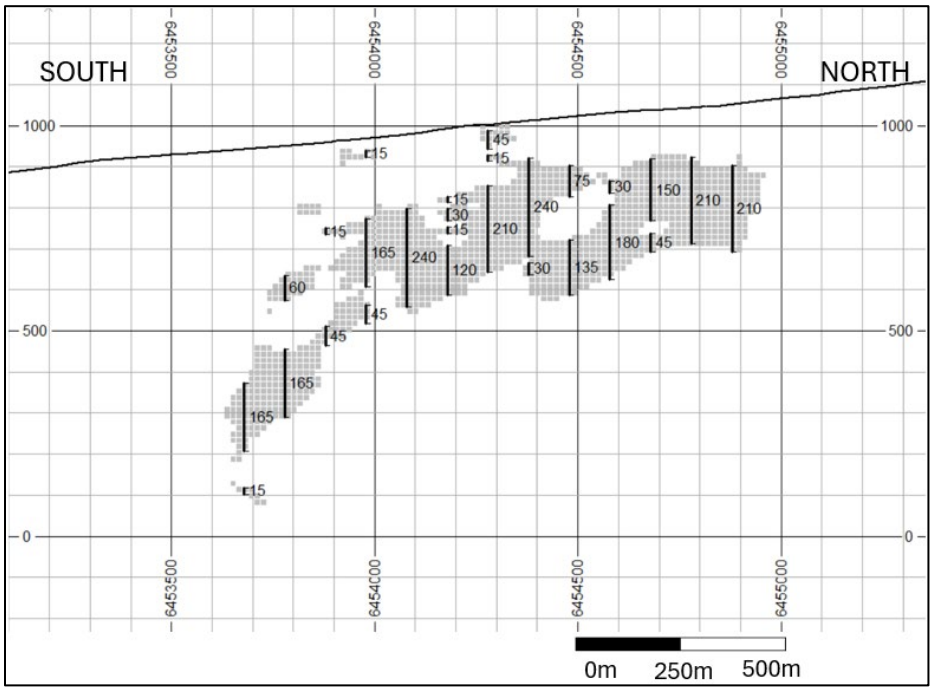


Figure 7-3: East 348000 Looking West, Cross-Section Illustrating the Geometry, Thickness Variability, and Lateral Continuity of the Copper (Cu) Domain Along the Central Structural–Magmatic Corridor

Gold Domain

The gold domain forms a continuous but internally variable mineralized corridor extending for approximately 1.4 km between 347100 E and 348500 E. Mineralization follows the northwest–southeast structural development (azimuth 150–160°) and is traceable over a strike length of roughly 1.5 km. Gold is narrow along the western margin, where thicknesses typically range from 90 m to 135 m, but it broadens rapidly eastward in Figure 7-4 and Figure 7-5 and reaches its maximum development in the central corridor between 347600 E and 348000 E. Within this central zone, composite thicknesses commonly range from 150 m to 330 m and locally exceed 450 m, reflecting several overlapping, steeply dipping panels that provide strong vertical and lateral continuity.

East of 348000 E, the domain remains well developed but becomes more segmented, with individual panels thickening and thinning along strike. From approximately 348100 E eastward, the geometry shifts in character: vertical development becomes increasingly dominant, and gold occurs as steep, vertically extensive lenses with down-dip thicknesses commonly >300 m and local maxima of 450 m to 650 m. This eastern segment represents a deeper-rooted structural zone where gold mineralization persists across large elevation ranges and maintains continuity through multiple sections.

Toward 348500 E, the domain tapers into thinner, more discontinuous pods, marking the practical eastern limit of continuous gold mineralization.

Overall, the gold domain is best described as a moderately continuous, centrally thickened corridor composed of several stacked mineralized panels that swell and pinch along strike, with a notable transition toward strong vertical continuity and increased thickness in the eastern part of the system.

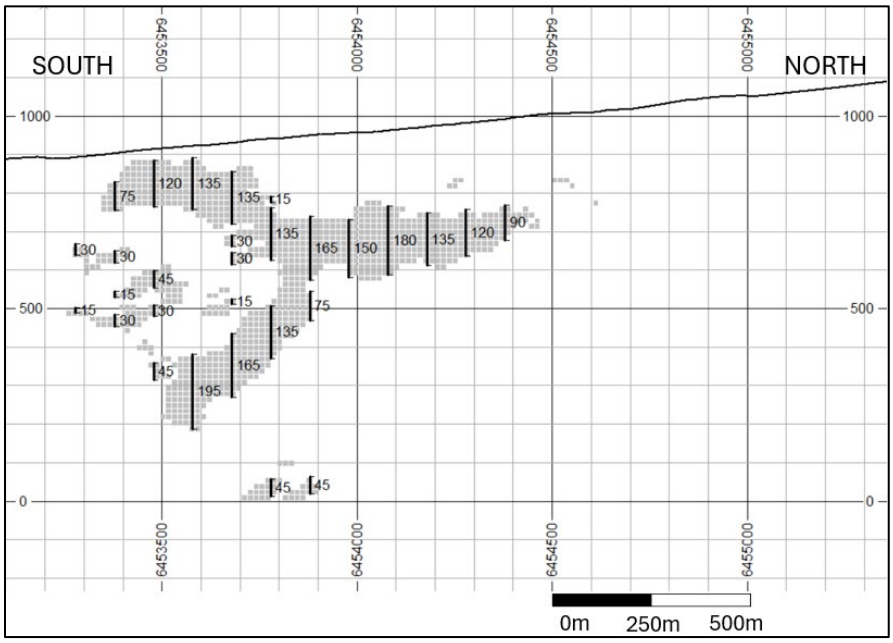


Figure 7-4: East 347700 Looking West, Cross-Section Illustrating the Geometry, Thickness Variability, and Lateral Continuity of the Gold (Au) Domain Along the Central Structural–Magmatic Corridor

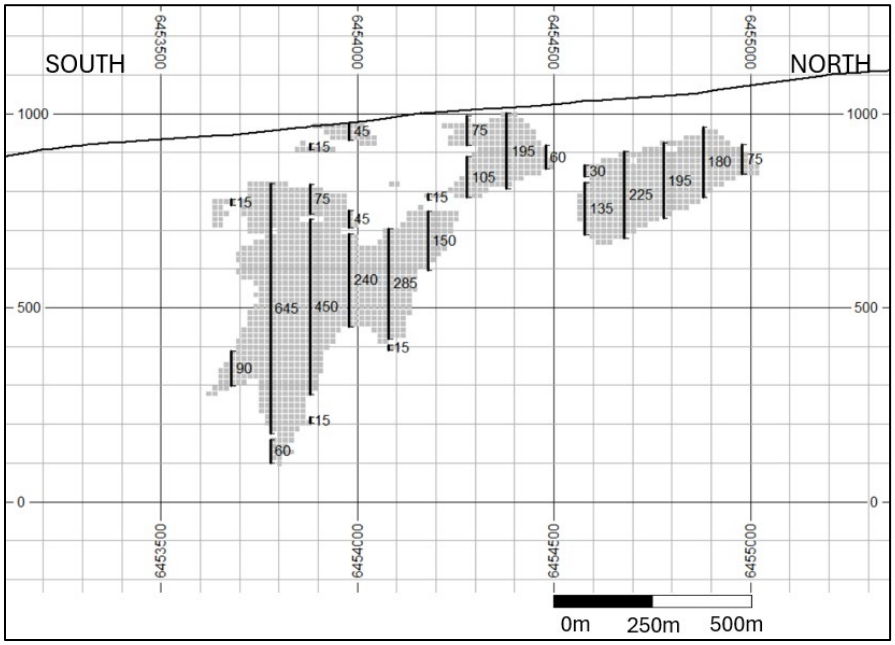


Figure 7-5: East 348200 Looking West, Cross-Section Illustrating the Geometry, Thickness Variability, and Lateral Continuity of the Gold (Au) Domain Along the Central Structural–Magmatic Corridor

Silver Domain

The silver domain forms an approximately 1.7 km long northwest–southeast–striking mineralized corridor (azimuth 150°–160°) with a steep dip and a well-developed central zone flanked by progressively narrowing margins. Mineralization is thin and discontinuous at the western end but strengthens rapidly eastward, developing into a coherent central corridor observed in Figure 7-6 and Figure 7-7 between approximately 347300 E and 347800 E. Within this interval, down-dip thicknesses commonly reach 300 m to 400 m and locally exceed 700 m. The central zone is characterized by overlapping mineralized sections contained within a shared structural envelope, producing strong vertical continuity across several hundred metres of elevation. Internal voids are present locally, introducing spatial discontinuities.

East of 347800 E, the silver domain becomes more variable, with a mixture of moderate-thickness intervals and thinner, discontinuous sections. Near 348200 E, a very irregular, subvertical structural "chimney" links the deepest recognized portion of the deposit to shallower subsurface positions through a series of discontinuous pods, ultimately tapering out toward approximately 348500 E.

Overall, the silver body is best described as a steep, structurally focused mineralized corridor with a thick, vertically persistent central core and margins that thin progressively toward both ends of strike.

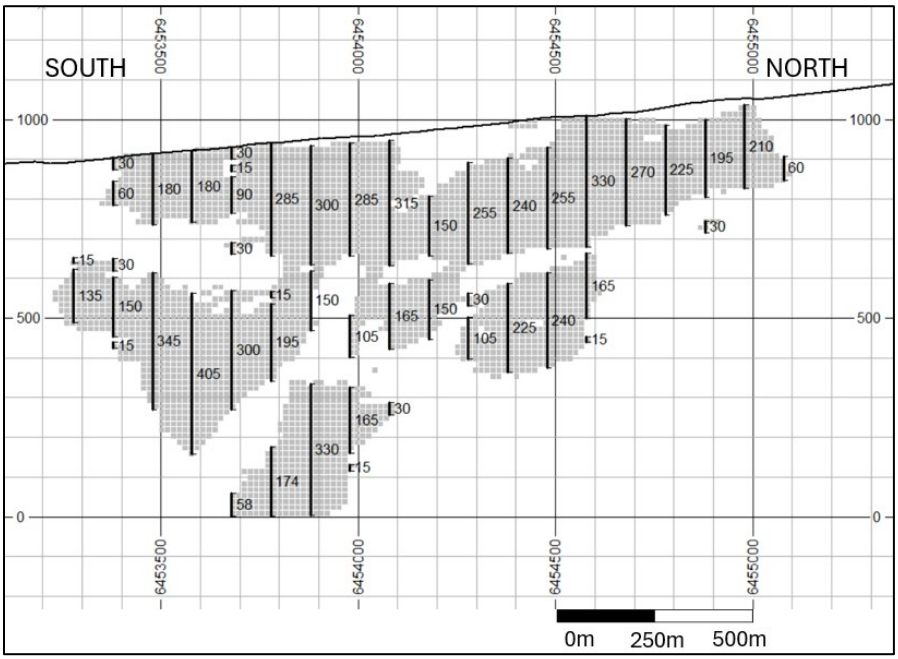


Figure 7-6: East 347700 Looking West, Cross-Section Illustrating the Geometry, Thickness Variability, and Lateral Continuity of the Silver (Ag) Domain Along the Central Structural–Magmatic Corridor

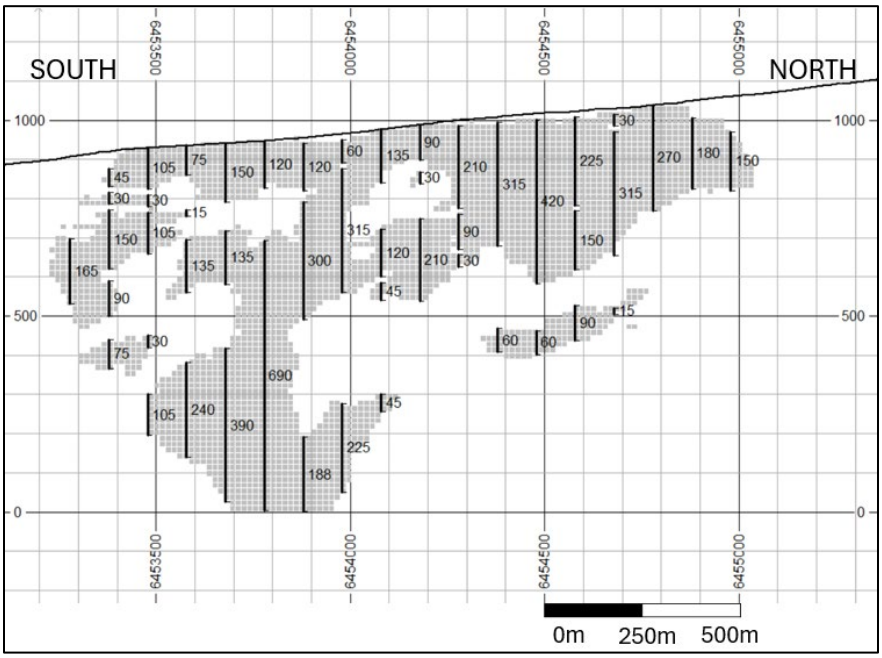


Figure 7-7: East 347900 Looking West, Cross-Section Illustrating the Geometry, Thickness Variability, and Lateral Continuity of the Silver (Ag) Domain Along The Central Structural–Magmatic Corridor

Cobalt Domain

The cobalt domain represents the most laterally extensive and vertically persistent mineralized volume within the Hat system. Thickness vectors constructed across 19 cross-sections define a continuous, steeply dipping corridor extending for approximately 1.7 km (azimuth 150–160°) between 346900 E and 348600 E. Although internal geometry varies locally, the domain maintains a strong and coherent footprint along its entire strike length.

True thickness values range from <60 m to >800 m, with average section thicknesses typically between 175 m and 500 m. The domain narrows toward both flanks, where average thicknesses are on the order of 50 m to 330 m and thickens markedly toward the central structural–magmatic corridor, where average values commonly exceed 300 m and locally surpass 600 m in Figure 7-8 and Figure 7-9 (e.g., sections 347300 E–348100 E and 348200 E–348400 E, the latter with more limited lateral development). This geometry defines a broad, lens-like cobalt-bearing volume that swells through the center of the system and tapers toward both the western lateral zone near 347100 E, where mineralization is developed along roughly 200 m to 300 m depth—and the eastern margin near 348500 E, where the domain narrows and closes at vertical section extensions greater than ~600 m.

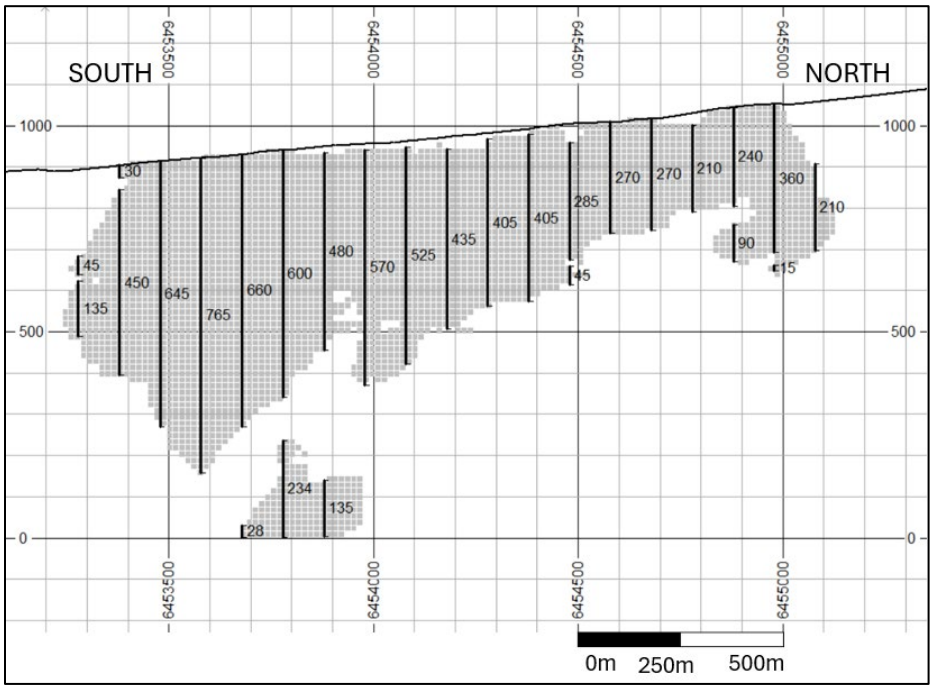


Figure 7-8: East 347700 Looking West, Cross-Section Illustrating the Geometry, Thickness Variability, and Lateral Continuity of the Cobalt (Co) Domain Along the Central Structural–Magmatic Corridor

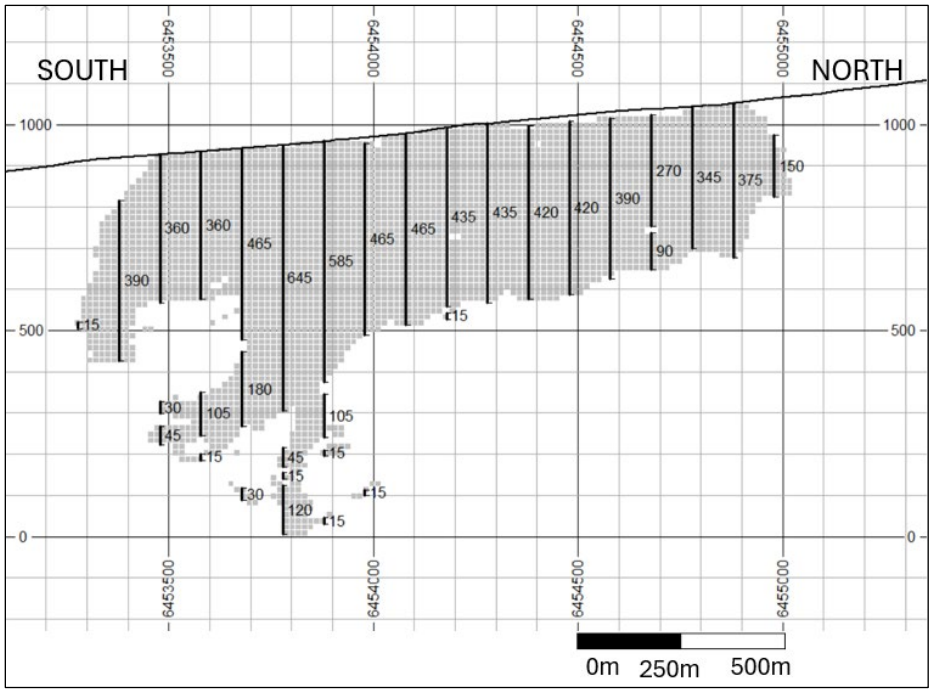


Figure 7-9: East 348000 Looking West, Cross-Section Illustrating the Geometry, Thickness Variability, and Lateral Continuity of the Cobalt (Co) Domain Along the Central Structural–Magmatic Corridor

7.4 Lithology and Alteration

The lithological and alteration interpretations presented in this section build on a standardized geological framework established during a relogging program Doubleview commissioned and CSA Global completed in 2018 (CSA Global, 2018a, 2018b). That work provided consistent lithological definitions, clarified several legacy misclassifications, and formalized the alteration terminology used across the Hat Property. The relogging confirmed that intrusive phases of mafic to intermediate composition dominate the Lisle Zone and that copper mineralization is preferentially associated with potassic and inner propylitic alteration assemblages, commonly overprinting earlier sodic alteration.

The present descriptions adopt this refined framework and integrate additional observations from subsequent drilling, petrography, and core photography. To illustrate the characteristic lithologies and alteration styles discussed in this section, drill hole H023 is used as a reference example. H023 was included in the CSA Global relogging program and provides a continuous, well-documented sequence that captures the principal intrusive units and alteration overprints of the Lisle Zone. Photographs from H023 at multiple depths are incorporated to demonstrate the variability of lithology, alteration intensity, and mineralization style within a single drill hole.

The Hat property is characterized by a multiphase dioritic intrusive complex emplaced into Stuhini Group mafic volcanics. The intrusive is heterogeneous, with both pyroxene-phyric diorite and equigranular diorite phases observed in drill core. The pyroxene-phyric diorite shown in Figure 7-10 is medium to coarse grained, showing gradual variations from leucocratic to melanocratic depending on pyroxene and hornblende content. This phase dominates the upper portions of the drill hole and is locally mineralized with disseminated and vein-fill copper sulphides. Strong, pervasive K-feldspar alteration is also present in intervals of this lithology.

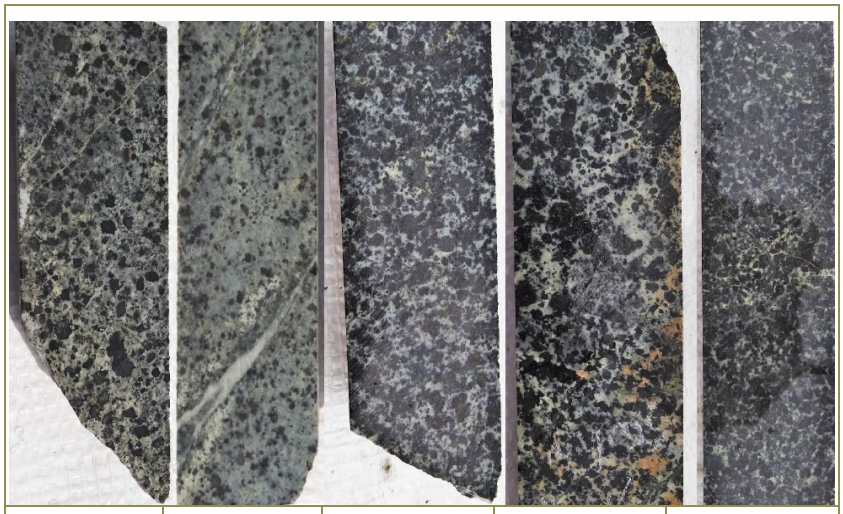
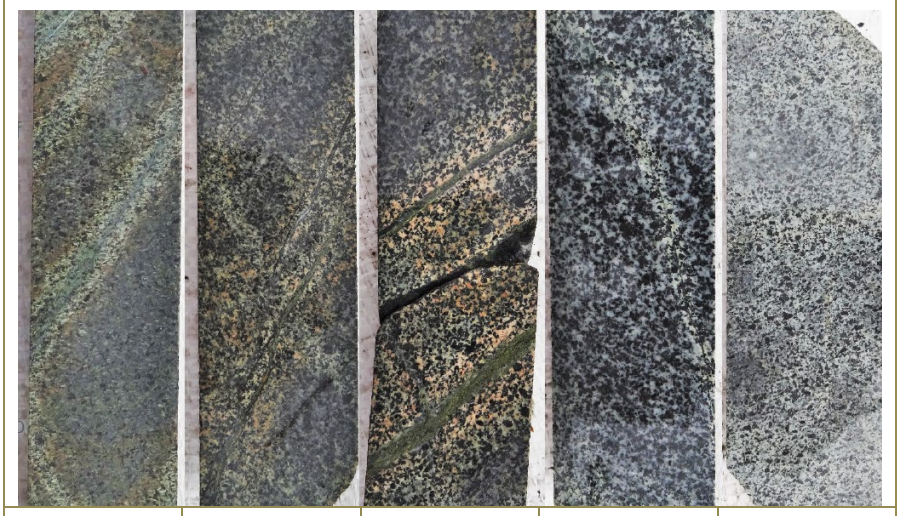


Figure 7-10: Pyroxene-Phyric Diorite

In contrast, the equigranular diorite in Figure 7-11 is fine to medium grained and dominates the lower half of H023. It is largely unmineralized, with only trace chalcopyrite disseminations. Alteration in this phase is expressed as moderate to strong sodic–calcic halos, including epidote–chlorite veining with albite envelopes, accompanied by significant calcite veining and vuggy infill.



H023-270.0 m	H023-335.0 m	H023-346.0 m	H023-419.5 m	H023-491.5 m

Figure 7-11: Fine- to Medium-Grained Equigranular Diorite

Intervals of pyroxenite and melanocratic diorite and gabbro in Figure 7-12 are also present, though rarely preserved in core due to extensive shearing and alteration. These units are typically transformed into chlorite–actinolite rubble. Importantly, they host pods of massive magnetite–chlorite, which represent localized enrichment zones.



H023-274.5 m

Figure 7-12: Pyroxenite

Alteration styles are diverse but consistently linked to mineralization. Potassic alteration in Figure 7-13 (K-feldspar ± biotite ± magnetite) is common, particularly below 180 m in H023, and is often associated with quartz-filled shear zones. Within these potassic domains, chalcopyrite and pyrite mineralization is most abundant. Intervals of chlorite–epidote–magnetite alteration coincide with elevated sulphide content, interpretation, and increasing confidence in the resulting geological and resource models.

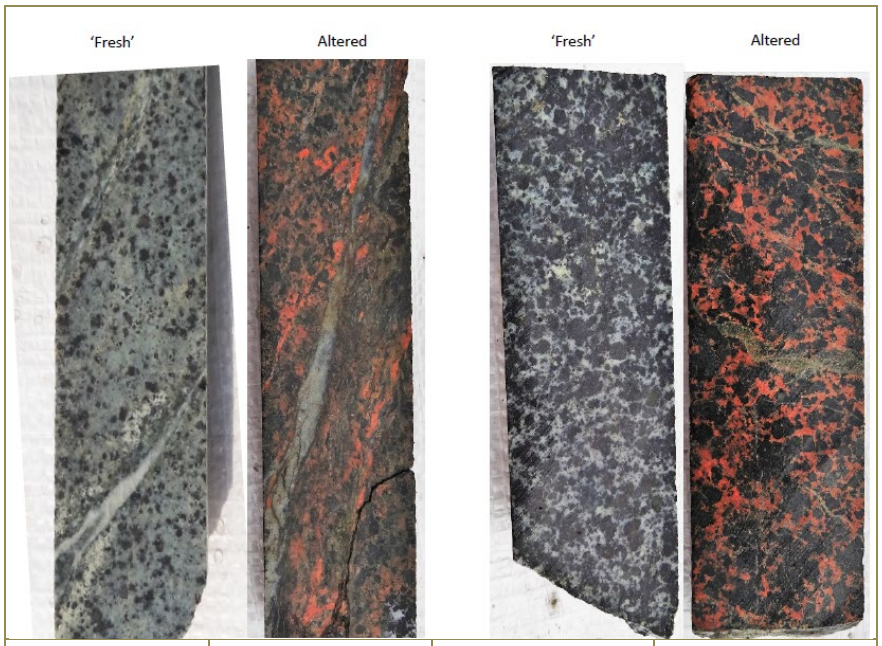


Figure 7-13: Potassic Alteration

Other alteration features presented in Figure 7-14, include sodic halos around epidote veins in both pyroxene-phyric and equigranular diorite. In the former, coarse chalcopyrite grains fill axial zones of veins, while in the latter, chalcopyrite is absent. Extensive sodic alteration fracture halos, calcite veins, and calcite vug infill are also documented.

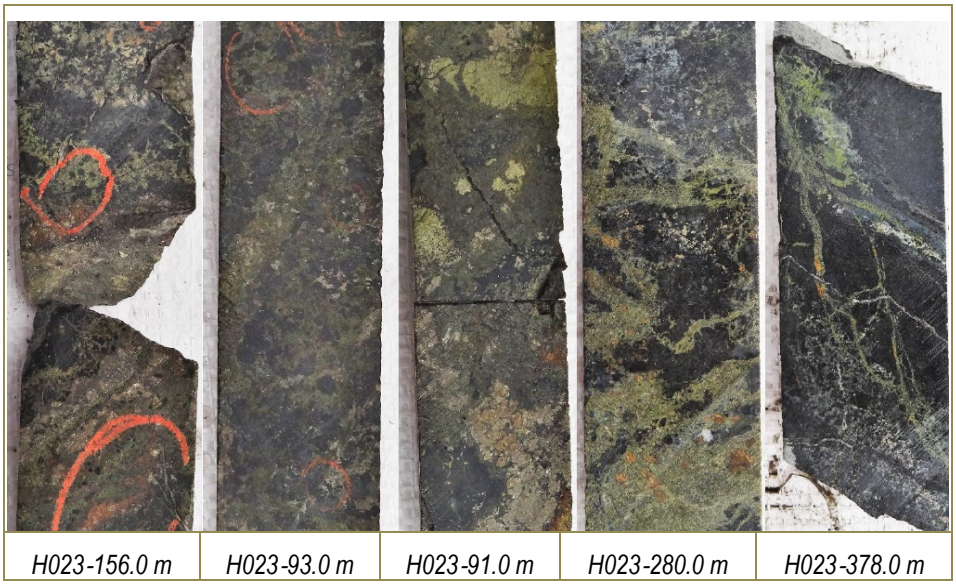


Figure 7-14: Chlorite/Epidote/Magnetite

7.4.1 Synthesis of Lithology and Alteration Controls on Mineralization at the Hat Deposit

The reported results (CSA Global, 2018a, 2018b) allow to formalize following conclusions: The geological framework of the Hat deposit is dominated by a multiphase diorite intrusive complex, comprising both pyroxene-phyric and equigranular phases with subordinate pyroxenite and gabbro. These intrusions are emplaced into Stuhini Group volcanic rocks and collectively define the principal lithological architecture that governs the distribution of copper mineralization. Among these units, the Hat Diorite consistently emerges as the primary host, providing both the physical and chemical environment most conducive to mineralizing fluid flow and sulphide deposition.

Alteration patterns across the system reveal a clear temporal and spatial evolution. Early sodic–calcic alteration is pervasive, particularly within the equigranular diorite, forming broad halos that predate mineralization. This early event establishes the initial permeability architecture but does not itself introduce significant copper. Mineralization is instead closely linked to subsequent potassic and inner propylitic alteration. Potassic zones, characterized by K-feldspar with or without magnetite, are spatially associated with quartz veining and localized shear zones, and they consistently correlate with the highest copper grades. Magnetite is ubiquitous throughout the system and locally forms massive pods intimately intergrown with chlorite, reinforcing the strong thermal and hydrothermal overprint associated with mineralizing pulses.

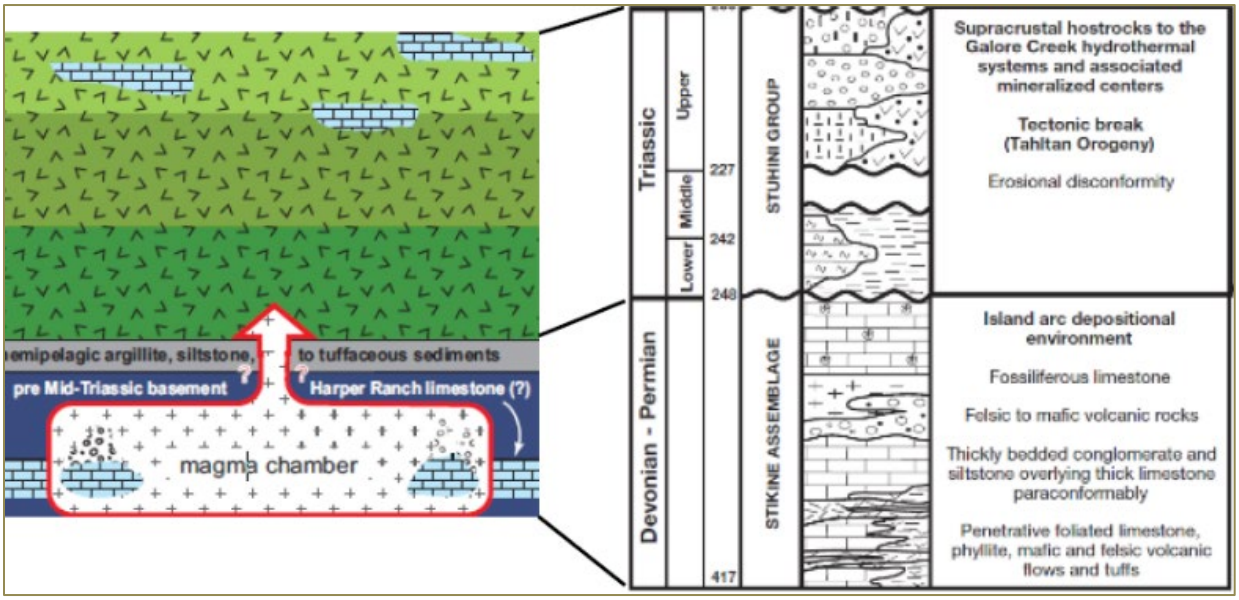
Inner propylitic alteration, comprising epidote, pyrite, and calcite, also plays a meaningful role in hosting chalcopyrite, particularly within epidote-rich vein networks. The distribution of chalcopyrite and pyrite—occurring as disseminations, fracture fillings, and epidote-vein infill—demonstrates that mineralization was introduced during these later alteration stages rather than during the early sodic event. Structural features, including breccias and shear zones, further enhance mineralization by focusing hydrothermal fluids and creating localized traps where copper grades are elevated.

Taken together, the lithological and alteration characteristics of the Hat system define a mineralizing environment that departs in several respects from classic porphyry models as described by Ayuso et al. (2010), particularly in its alteration zonation and the prominence of chlorite–magnetite assemblages. When compared with Andean porphyry systems documented by Piquer et al. (2021), the Hat deposit shows a distinct structural and hydrothermal evolution that does not conform to typical potassic-centered porphyry architecture.

The atypical combination of pervasive chlorite, strong magnetite development, and the interplay between sodic, potassic, and propylitic assemblages requires an integrated exploration approach. Geophysical and geochemical datasets are essential for delineating mineralized domains, as alteration zonation alone does not follow conventional porphyry patterns. Nonetheless, the consistent association of high-grade copper with potassic alteration and structurally focused fluid pathways provides a robust predictive framework for ongoing exploration and resource definition.

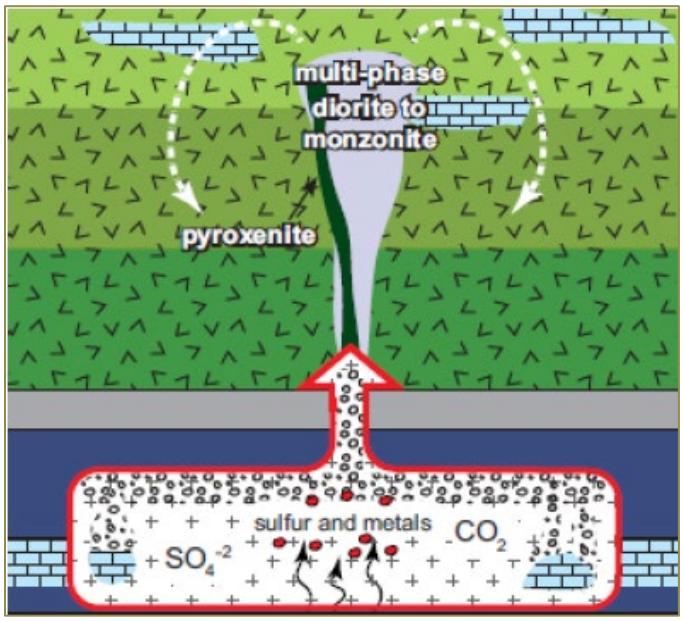
7.4.2 Proposed Genetic Model for Hat Intrusion–Centered Mineralization

Figure 7-15 to Figure 7-18 illustrates the intrusion-centered mineralizing system. They present a stepwise depiction of the system's evolution, beginning with the pre-existing Stuhini volcanic and sedimentary basement and progressing through emplacement and differentiation of the Hat intrusive complex, the brecciation and hydrothermal events responsible for copper–gold mineralization, and finally the late subvolcanic and structural overprints. This phased framework provides essential context for interpreting the lithological, alteration, and mineralization relationships described in the preceding sections.



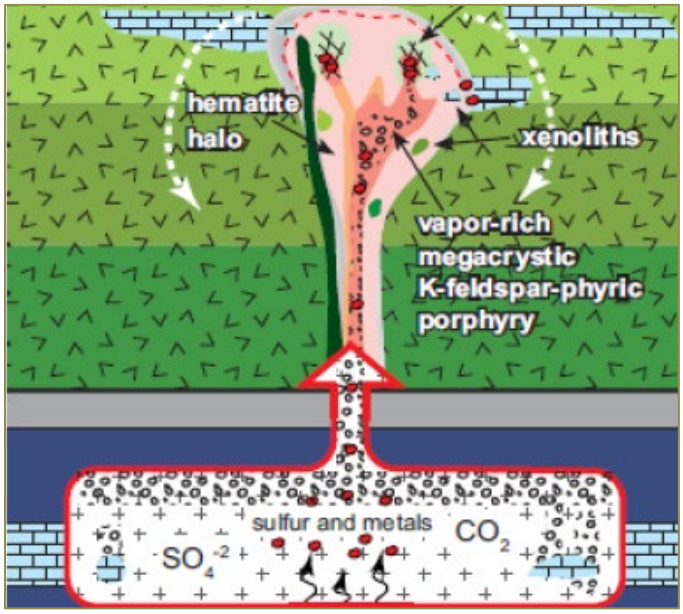
Source: Adapted from Pass et al. (2014).

Figure 7-15: Phase 1—Pre-Hat Intrusion: Triassic Stuhini Volcanics and Pre-Triassic Carbonate-Sedimentary Basement (Stikinia Arc Terrain)



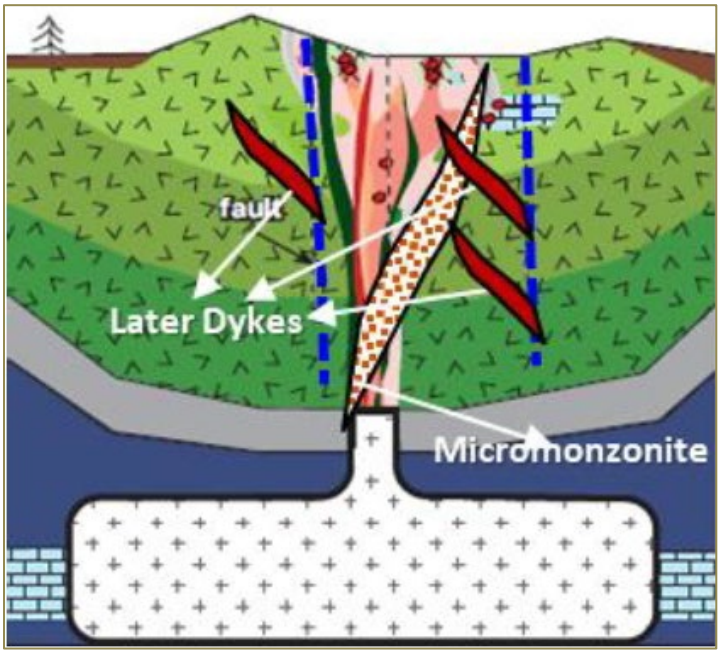
Source: Adapted from Pass et al. (2014).

Figure 7-16: Phase 2—Main Diorite and Pyroxenite Intrusions (Hat Intrusive Complex). As Magmas Ascended Through the Sedimentary Substrate, they Assimilated Carbonate From Wall-Rocks, Which Caused CO_2 Saturation and Drove Magma to Silica-Undersaturated Compositions that Facilitated Pyroxene Crystallization and Crystal Fractionation (Coeval Diorite Phase)



Source: Adapted from Pass et al. (2014).

Figure 7-17: Phase 3—Brecciation and Mineralization Events. The CO_2 Exsolution Drove a Magma-Hydrothermal Brecciation Event, Where Oxidized Magmatic-Hydrothermal Fluids Precipitated Producing High-Temperature (>400°C) Breccia Cement and Pervasive Potassic (K-Feldspar-Magnetite) Alteration Along with Cu-Fe Sulphides Precipitation



Source: Adapted from Pass et al. (2014).

Figure 7-18: Phase 4—Post-Mineralization, Late Subvolcanic Phases and Faulting. Post-Mineralization Micro-Monzonite Unit Formation and Later Andesitic to Basaltic Dykes

7.5 Local Lithology Features based on Lithological Modelling

Detailed analysis of geological logging indicates that 21 distinct rock names were initially recorded in the database. Several of these were symbolic or represented very low-frequency occurrences and were therefore consolidated into major lithologies to ensure consistency. After this refinement, the final lithological set comprises four intrusive rocks (diorite, gabbro, monzonite, pyroxenite) and three volcanic rocks (andesite, trachyte, tuff).

Knowledge of local lithological distribution derives from geological modelling undertaken for geotechnical purposes and to establish the relationship between intrusive units and mineralization. Intrusive rocks are recognized as the principal hosts of the mineralizing system. The deepest drill holes (H036, 972 m; H101, 1,015 m) appear to intersect the roots of the intrusion. At 1,000 m deep, diorite and monzonite dominate, intruding through a massive trachyte unit, while the broader system is hosted by andesite with local occurrences of tuff.

Laterally, lithological distribution varies across the property (Figure 7-19 to Figure 7-22):

- Eastern side—monzonite predominates, with occasional diorite
- Western side—diorite becomes the dominant lithology
- Southern side—characterized by an intercalated relationship between monzonite and diorite

- Northern side—monzonite dominates in the east, diorite in the west, with diorite becoming increasingly prevalent.

In addition to these lateral trends, two lithologies occur in irregular, non-directional patches:

- Pyroxenite—occurs in small to moderate patches, generally associated with diorite rather than monzonite
- Gabbro—forms minor patches within diorite but is generally absent in monzonite.

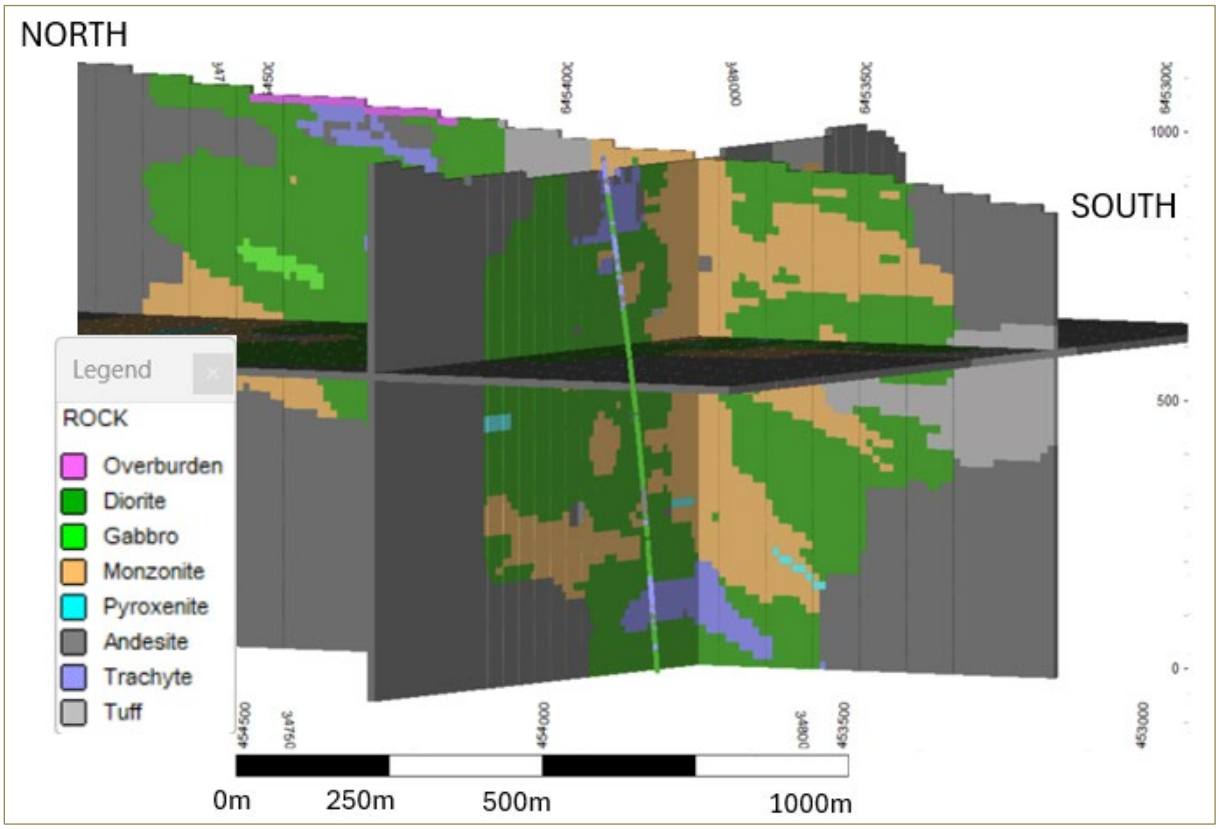


Figure 7-19: 3-D Perspective View of Drill Hole H036 within the Intrusion-Centered Hydrothermal System

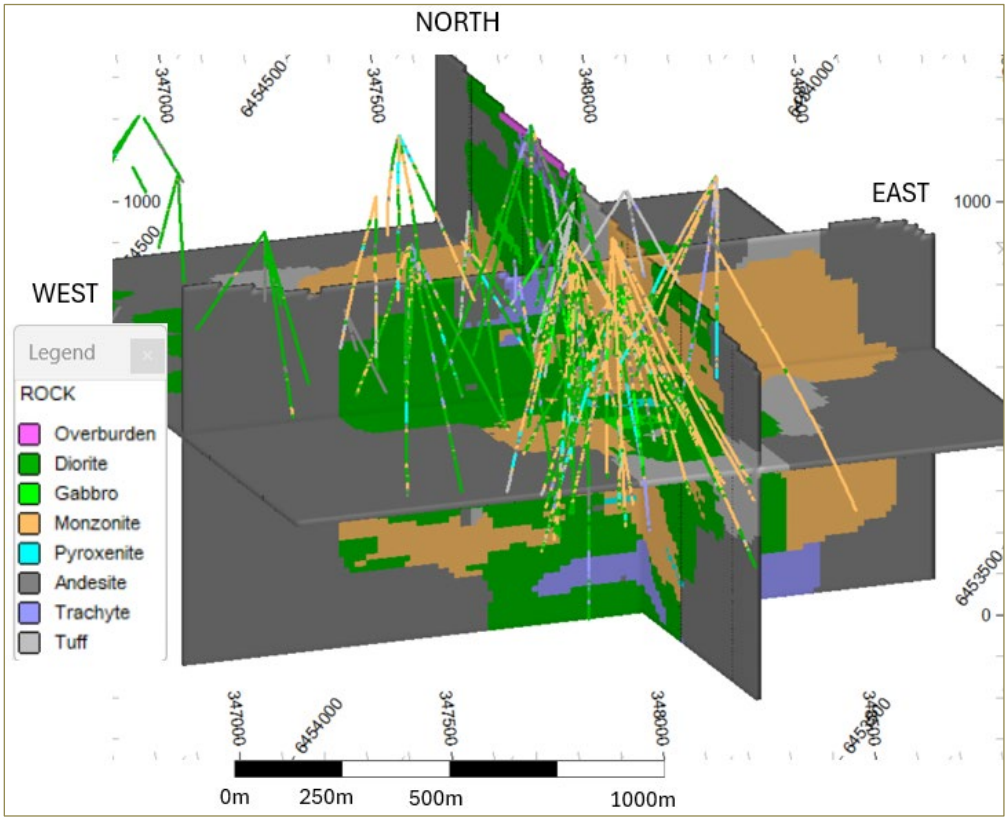


Figure 7-20: 3-D Spatial Projection of Drill Holes within the Intrusion-Centered Hydrothermal System

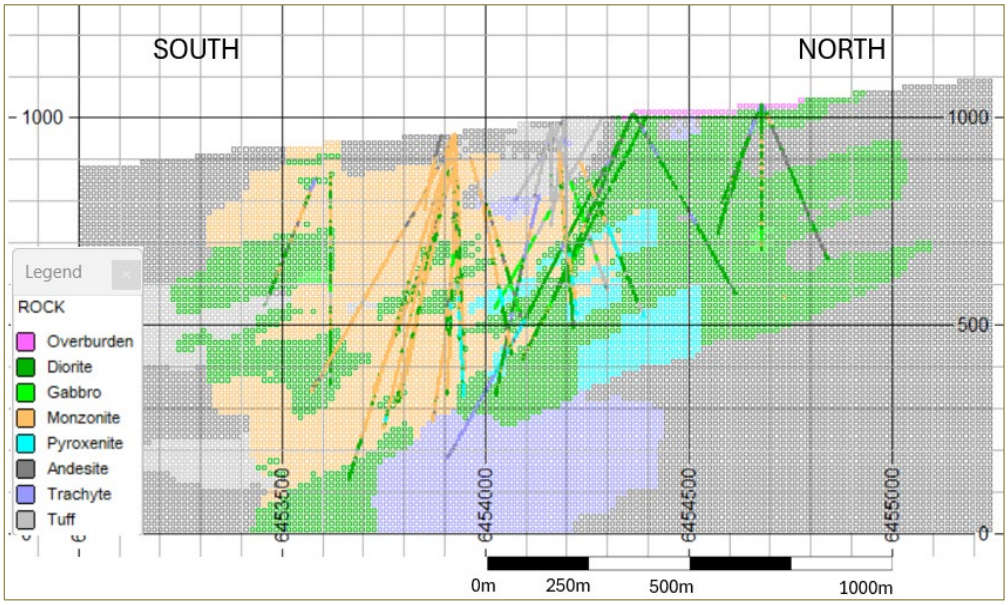


Figure 7-21: East 348100 Looking West, Geological Cross-Section through the Intrusive Complex Block Model and Drill Hole Projection

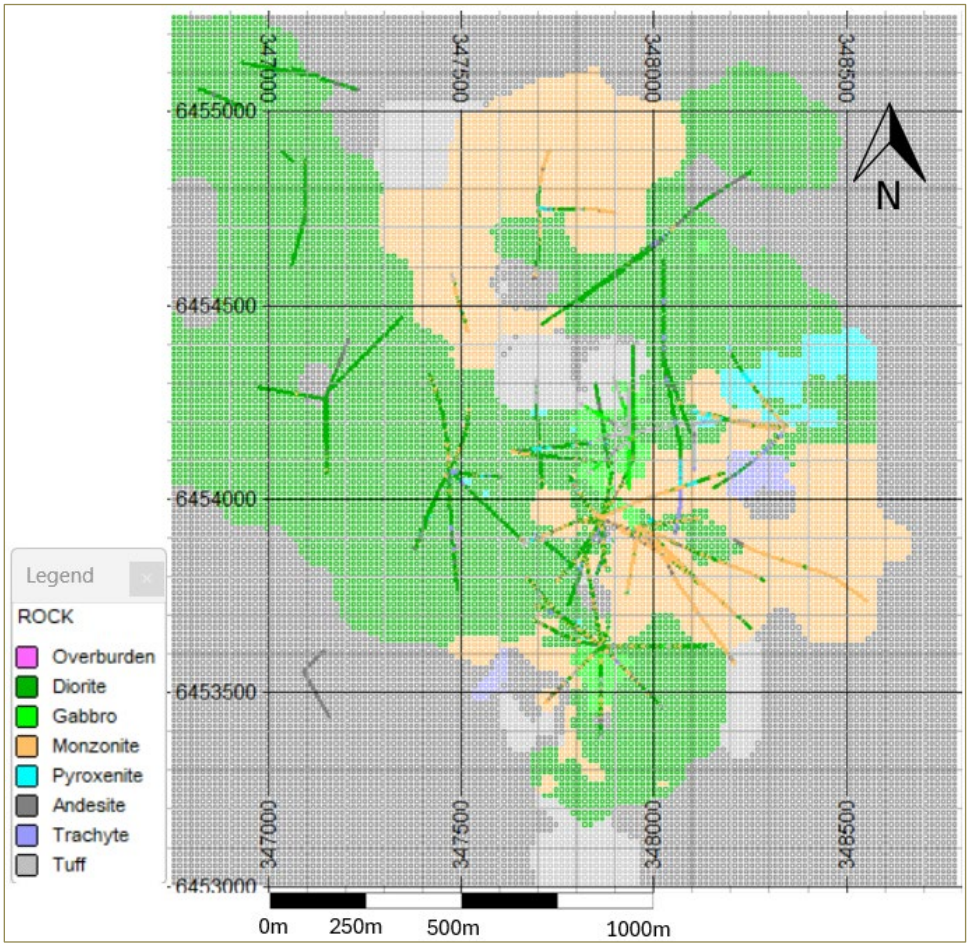


Figure 7-22: Horizontal Section through the Lithology Block Model at 660 masl

Figure 7-21 to Figure 7-22 illustrate the lithological distribution within a single vertical and horizontal slice, respectively, and should not be interpreted in isolation as a complete representation of the property-scale patterns described above.

Volcanic rocks play a secondary but important role. Andesite and trachyte, with occasional tuff, host the intrusive system. Volcanic rocks also occur as patches within intrusive units, with andesite and tuff frequently present. Trachyte patches are observed at the top of the mineralized system, showing a distribution similar to that of andesite and tuff.

Mineralization is present throughout both intrusive and volcanic rocks, as illustrated in Figure 7-23 to Figure 7-27. Although intrusive units tend to display slightly higher overall grades than the volcanic intervals enclosed within the intrusive complex, spatial analysis does not reveal any major or systematic lithological control on the distribution of mineralization. Figure 7-23 to Figure 7-27 present the spatial conditional grade distributions for gold, silver, copper, cobalt, and scandium, each evaluated with respect to lithology type.

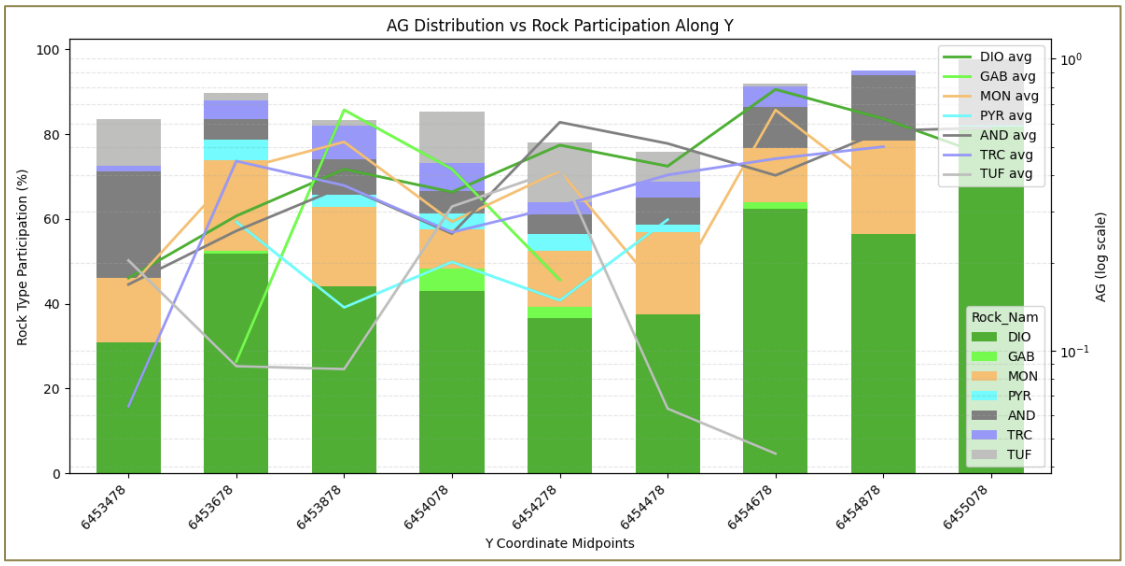


Figure 7-23: Grade Distribution Across Volcanic and Intrusive Lithologies: Silver (Ag)

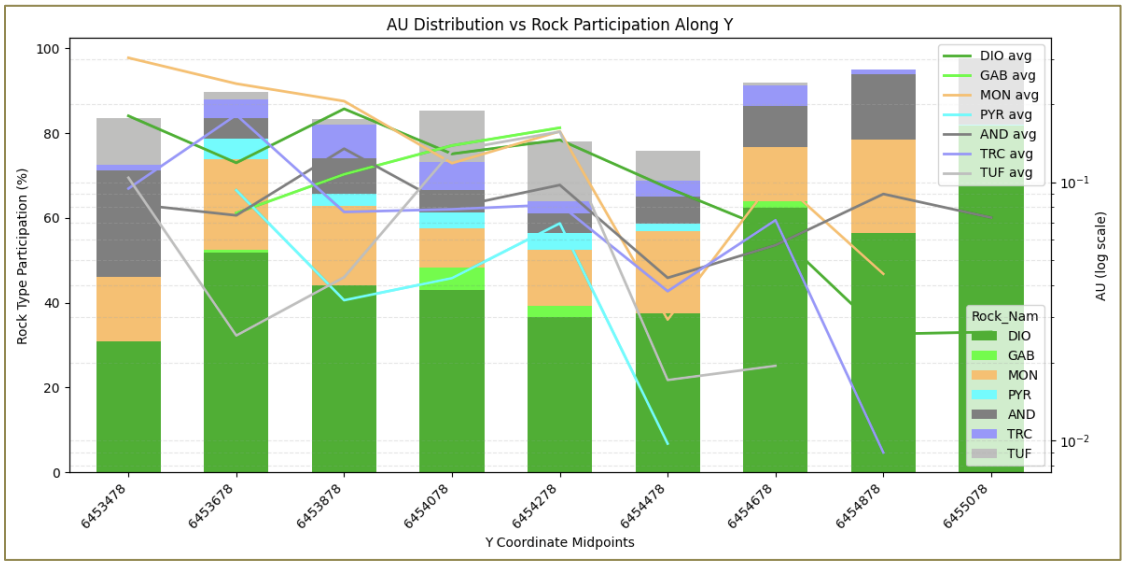


Figure 7-24: Grade Distribution Across Volcanic and Intrusive Lithologies: Gold (Au)

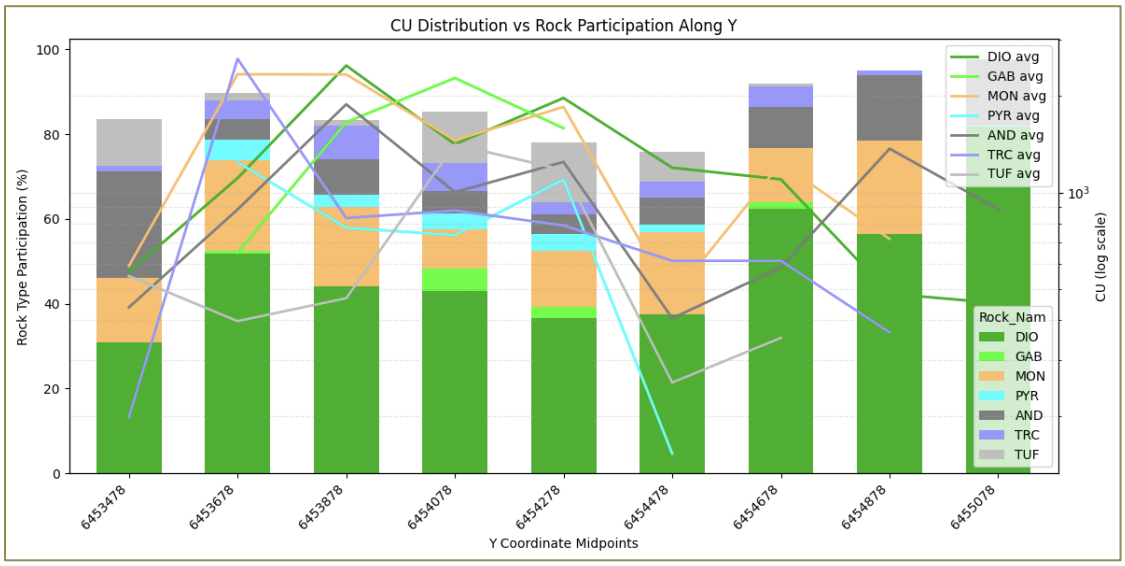


Figure 7-25: Grade Distribution Across Volcanic and Intrusive Lithologies: Copper (Cu)

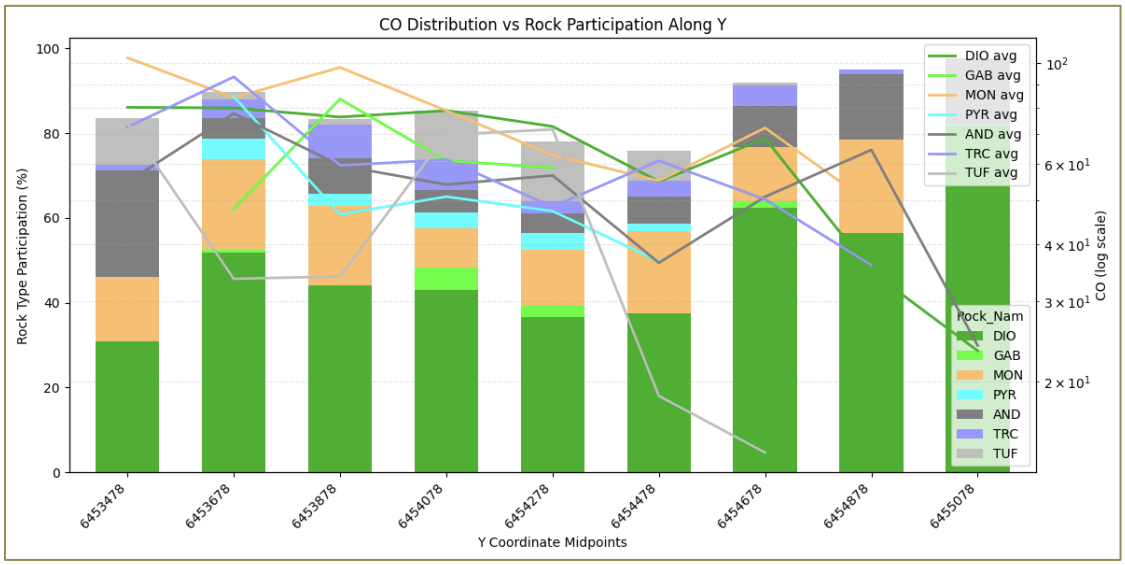


Figure 7-26: Grade Distribution Across Volcanic and Intrusive Lithologies: Cobalt (Co)

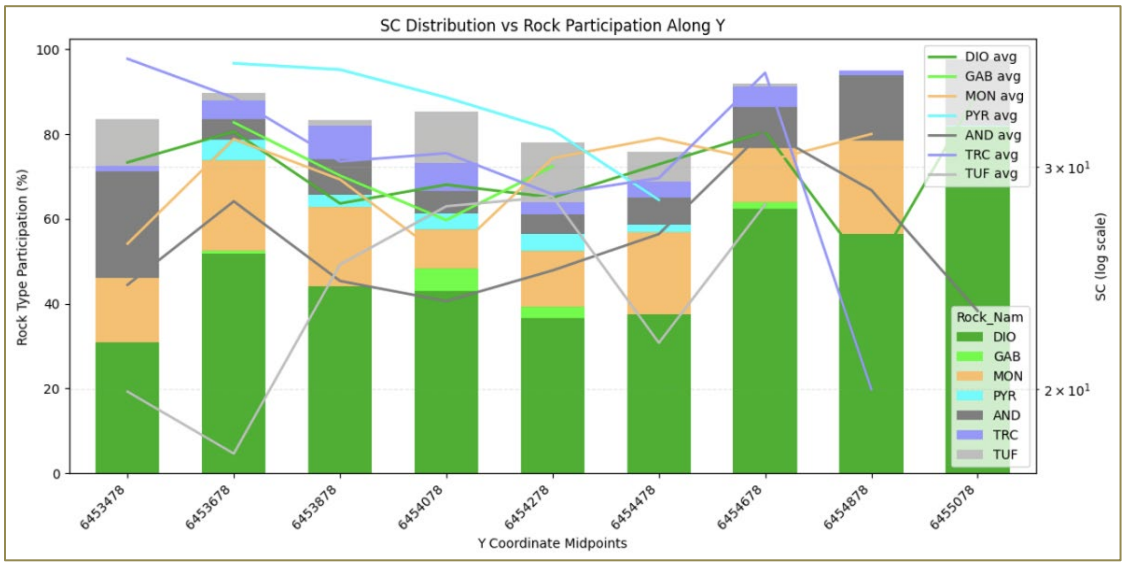


Figure 7-27: Grade Distribution Across Volcanic and Intrusive Lithologies: Scandium (Sc)

8 DEPOSIT TYPES

The Hat deposit is classified as an alkalic porphyry copper–gold system, formed through the interaction of multi-pulse dioritic intrusions, and a long-lived magmatic–hydrothermal system (Ayuso et al., 2010). Although the deposit displays many hallmarks of porphyry copper–gold mineralization, several geological features distinguish it from classical calc-alkalic models, including its dioritic intrusive affinity, magnetite-rich alteration assemblages, and the spatial association of copper mineralization with potassic and propylitic overprints (Doubleview, 2014; McLeod Geological, 2014; Payne, 2021). Together, these characteristics reflect the unique magmatic, structural, and hydrothermal evolution of the Hat system and support its interpretation as an alkalic porphyry copper–gold deposit (Anderson, 1994; Nelson & Kyba, 2014).

8.1 Intrusive Framework

Mineralization is centered on a dioritic intrusive complex emplaced in multiple pulses (Payne, 2021; Read, 1979). Field mapping, drill core relogging, and petrographic observations indicate at least two distinct generations of dioritic intrusions, each contributing to the thermal and hydrothermal evolution of the system (CSA Global, 2018; Doubleview, 2014).

The geometry of the intrusive bodies, together with associated breccia zones and fracture corridors, plays a central role in focusing mineralization (Piquer et al., 2021). Higher-grade domains commonly occur where intrusive contacts create permeability contrasts; hydrothermal breccias act as fluid-focusing conduits; and fracture networks provide sustained pathways for mineralizing fluids (United States Geologic Survey [USGS], 2010). This spatial relationship between intrusive architecture and mineralization is one of the defining features of the Hat system.

8.2 Alteration Characteristics

Hydrothermal alteration is well developed across the Hat system but displays several atypical features relative to classical porphyry copper–gold deposits (USGS, 2010). Alteration styles are spatially and temporally linked to the multiphase intrusive architecture and the structural pathways that focused fluid flow (Piquer et al., 2021). The principal alteration assemblages include potassic, sodic, and chlorite–epidote (propylitic) zones, each reflecting distinct stages of the magmatic–hydrothermal evolution (Anderson, 1994).

8.2.1 Potassic Alteration

Potassic alteration represents one of the earliest and highest-temperature hydrothermal events (USGS, 2010) and forms the core of the mineralizing system. In the Hat deposit, potassic alteration is not uniformly pervasive; instead, it occurs in patchy, structurally focused zones, commonly developed along

intrusive contacts, breccia bodies, and fracture corridors where permeability was highest (Doubleview, 2014; Payne, 2021).

Potassic alteration is characterized by:

- Secondary biotite replacing primary mafic minerals
- K-feldspar replacement and localized flooding
- Magnetite ± chalcopyrite assemblages (McLeod Geological, 2014).

The distribution of potassic alteration reflects the irregular geometry of the multi-pulse dioritic intrusions and the episodic nature of early fluid flow. In several areas, potassic assemblages are partially overprinted by later chlorite–epidote alteration, but the underlying mineralization signature is commonly preserved. Where potassic alteration is best developed, copper and gold grades tend to be elevated, confirming its role in early metal introduction (USGS, 2010).

8.2.2 Sodic Alteration

Sodic alteration is also a significant component of the hydrothermal system, particularly within early intrusive phases and deeper structural levels (Anderson, 1994). Like potassic alteration, sodic alteration in the Hat system is spatially irregular, forming discontinuous zones controlled by early fracture networks and high-temperature fluid overprints (Payne, 2021).

Sodic alteration is expressed by:

- Pervasive to patchy albite replacement of plagioclase
- Actinolite–magnetite assemblages
- Early high-temperature fluid overprints.

Sodic alteration may represent an early, high-temperature fluid pulse that prepared the rock mass for subsequent mineralizing events (USGS, 2010). Although not uniformly developed, sodic assemblages are locally associated with moderate-to-high copper and gold grades, particularly where albite and actinolite occur together with magnetite and chalcopyrite (Doubleview, 2014). This suggests that sodic alteration contributed to early metal introduction, with later potassic alteration upgrading or redistributing mineralization.

Where sodic assemblages are overprinted by chlorite–epidote alteration, grades may decrease; however, in many cases the early high-temperature mineralization signature remains intact.

8.2.3 Chlorite–Epidote (Propylitic) Alteration

Chlorite–epidote alteration is present along the outer margins of the hydrothermal system and represents the propylitic halo of the porphyry environment (USGS, 2010). This alteration is characterized by chlorite replacement of mafic minerals, patchy epidote development, epidote–chlorite veinlets, and local calcite ± pyrite. In the Hat system, chlorite–epidote assemblages occur primarily in peripheral intrusive phases

and along late fracture networks and commonly overprint earlier potassic and sodic assemblages (Payne, 2021).

Mineralization within this alteration type is generally weak; however, where chlorite–epidote overprints potassic or sodic cores, moderate copper and gold grades may be preserved. The presence of chlorite–epidote alteration provides an important indicator of the thermal gradient and fluid evolution of the system and helps define the transition from the mineralized core into cooler, more distal hydrothermal regimes.

8.2.4 Atypical Alteration Features

Several features distinguish the Hat system from more classical porphyry copper–gold deposits:

- Weak or poorly developed phyllic (sericitic) alteration
- Limited quartz veining, often visible only in thin section
- Absence of well-defined alteration halos at the deposit scale.

These characteristics suggest a system dominated by high-temperature, low-sulfidation fluids, with limited development of the cooler, more acidic fluid regimes typical of quartz–sericite alteration (Creixell et al., 2019). The subdued phyllic overprint and scarcity of quartz veining are consistent with a magmatic hydrothermal system where early high-temperature alteration and mineralization were dominant, and late-stage acidic fluids were either weakly developed or poorly preserved.

8.3 Mineralization Style

Copper and gold mineralization occurs in several structural and textural settings, including:

- Disseminations within altered intrusive rocks
- Fine veinlets and stockworks
- Fracture-controlled networks
- Hydrothermal breccias.

The principal ore minerals include chalcopyrite, bornite, magnetite, and pyrite (Doubleview, 2014; Payne, 2021). Gold is typically associated with chalcopyrite and magnetite, consistent with copper–gold porphyry systems. Mineralization intensity increases where intrusive contacts, breccia bodies, and fracture corridors intersect (Piquer et al., 2021), highlighting the importance of structural preparation and permeability contrasts.

8.4 Deposit Interpretation

The geological characteristics of the Hat deposit define a large-volume porphyry copper–gold system, with:

- Multi-pulse dioritic intrusions
- Structurally focused hydrothermal pathways

- Patchy but significant potassic and sodic alteration linked to higher grades
- Weak phyllic alteration and limited quartz veining
- Mineralization concentrated in breccia and fracture networks.

The deposit is best described as a porphyry copper–gold system with atypical alteration development, shaped by local magmatic, structural, and lithological controls.

9 EXPLORATION

Exploration activities at the Hat Project extended beyond traditional drilling and incorporated a suite of complementary, non-invasive methods designed to characterize geochemical, lithological, and alteration patterns at surface and at depth. Soil geochemical sampling and ground geophysical surveys, particularly induced polarization (IP), provided critical datasets that enhanced the understanding of mineralizing processes and supported the development of a coherent exploration model.

Systematic soil sampling delivered spatially continuous geochemical information, enabling the identification of metal anomalies, alteration signatures, and lithological contrasts that are not readily resolved through drilling alone. These data contributed to mapping the distribution of key pathfinder elements and alteration assemblages, offering valuable vectors toward mineralization.

IP geophysics, through chargeability and resistivity measurements, supplied three-dimensional constraints on subsurface physical properties. These results helped delineate potential sulphide-bearing zones, structural controls, and lithological boundaries, and allowed correlation with surface geochemical patterns. The integration of geochemical and geophysical datasets revealed meaningful spatial relationships that strengthened the geological interpretation and guided subsequent targeting.

Together, these alternative exploration methods provided a strategic framework for Mineral Resource development at the Hat Project, demonstrating their importance in refining targets, reducing uncertainty, and supporting efficient decision-making prior to drilling or during the drilling campaign.

9.1 Soil Sampling Program

A soil-geochemistry program was completed between July 21 and August 1, 2018, as part of the broader 2018 exploration campaign. A five-person crew from Pika Exploration carried out the work, collecting a 433 B-horizon[1] soil samples in the vicinity of the Hat Camp and the Lisle Prospect. Sampling focused on delineating geochemical responses associated with intrusive and volcanic lithologies and identifying potential extensions of the Lisle mineralized system.

Samples were collected using shovels and hand augers, targeting the B-horizon to ensure consistent material and minimize surficial contamination. Each sample was georeferenced using handheld GPS units, and field notes recorded soil characteristics, depth, and local terrain conditions. Samples were dried, sieved, and prepared for shipment following standard industry protocols.

[1] The B horizon is a subsurface mineral soil layer that occurs below an O, A, or E horizon and above the C horizon. It develops through pedogenic processes, including the downward movement and accumulation (illuviation) of materials from overlying horizons. The B horizon is typically characterized by the accumulation of silicate clay, iron, aluminum, or other leached constituents, and commonly displays stronger structure and greater density than the A horizon, reflecting active soil-forming processes (USDA Soil Survey Manual, 2017)

9.1.1 Sample Preparation and Analytical Methods

All soil samples were submitted to Bureau Veritas Laboratories (Bureau Veritas), an ISO-certified facility. The ISO certification is described in Section 11.6. Sample preparation included drying and sieving to 230 mesh, a step specifically intended to enhance the signal-to-noise ratio for low-level gold and trace-element analyses.

Analytical work consisted of:

- AQ251 + LF200 multi-element package (aqua regia digestion with ICP-MS finish)
- Additional analyses triggered by threshold values:
  - MA200 for base metals
  - FA130 for gold
  - MA370_Cu for Cu >8,000 ppm
  - MA370_Mo for Mo >3,200 ppm
  - FA430 for Au >1ppm.

This workflow ensured accurate quantification across a wide range of concentrations and provided high-resolution multi-element data suitable for geochemical interpretation.

9.1.2 Quality Assurance and Quality Control

The soil-sampling program incorporated standard QA/QC procedures, including inserting field duplicates, blanks, and certified reference materials (CRM). In addition, Bureau Veritas applied its internal QA/QC protocols, including method blanks, pulp duplicates, and certified standards. No material issues were reported, and the dataset is considered reliable for exploration purposes. QA/QC results for the soil sampling program were not available at the time of reporting and could not be included as part of Section 11.

9.1.3 Geochemical Results and Interpretation

The soil geochemistry survey successfully delineated several significant geochemical patterns relevant to understanding the Lisle Zone mineralization system.

Copper and Gold Distribution

Copper anomalies (>250 ppm) occur at Lisle Zone drilling sites (exploration campaigns from 2013 to 2016 with 10,727m drilled, Section 10.4) and extend into previously unrecognized areas north and south of the Lisle Zone in Figure 9-1.

Elevated gold values are spatially associated with less-differentiated intrusive phases and equivalent volcanic rocks. (There are no geochemical maps of the area.)

High copper and gold concentrations correlate with least-altered lithotypes, suggesting mineralization is closely linked to primary magmatic processes rather than late-stage alteration.

Molybdenum and Silver Distribution

Molybdenum and silver display the opposite behaviour to copper and gold, showing enrichment in more evolved intrusive phases (Figure 9-1). (Geochemical map for silver is not available.)

Molybdenum and copper overlap in some areas, but do not show a strong relationship with alteration type.

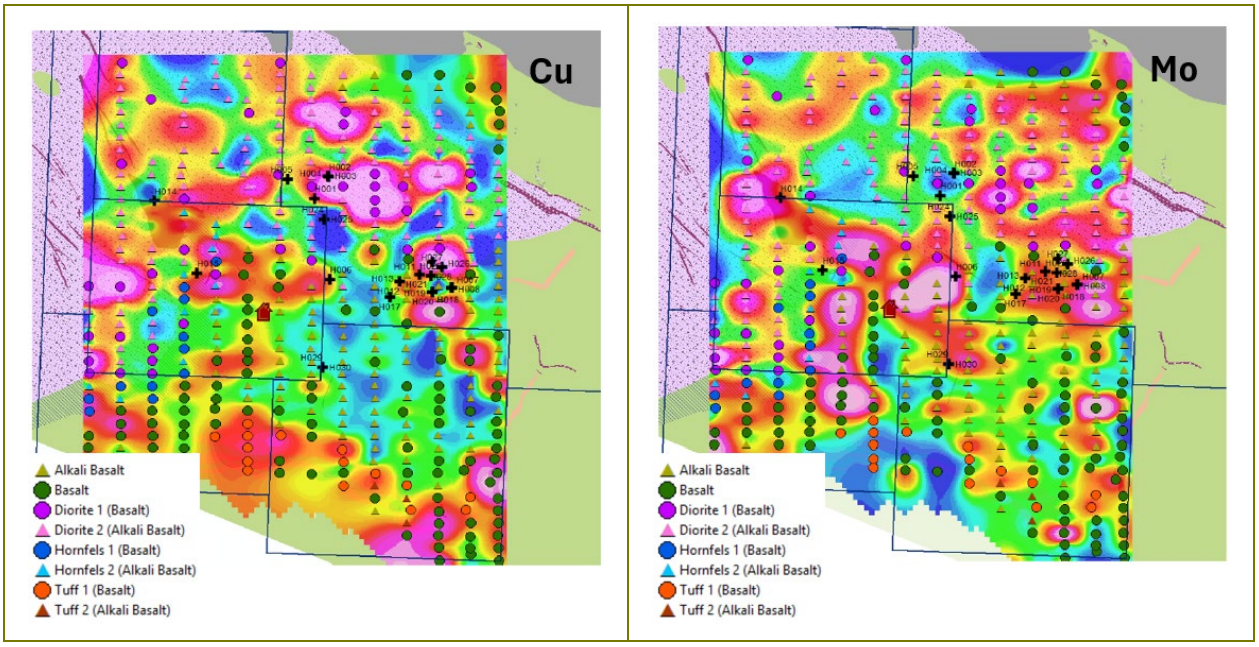


Figure 9-1: Copper (Cu) and Molybdenum (Mo) Distributions in B-Horizon Soils

Pathfinder Elements

Tin and arsenic exhibit broad areal distributions and provide important vectors (Figure 9-2):

- Tin is a proximal indicator of porphyry-related mineralization associated with dioritic intrusives, particularly the more-evolved alkaline basaltic phases, and is spatially coincident with the Lisle Zone.
- Arsenic (and tin, not shown) display enrichment patterns characteristic of a distal epithermal environment, with elevated concentrations in volcanic rocks positioned away from the main copper mineralization. This spatial distribution in Figure 9-2 aligns with geochemical signatures in Figure 9-3 typical of distal epithermal systems.

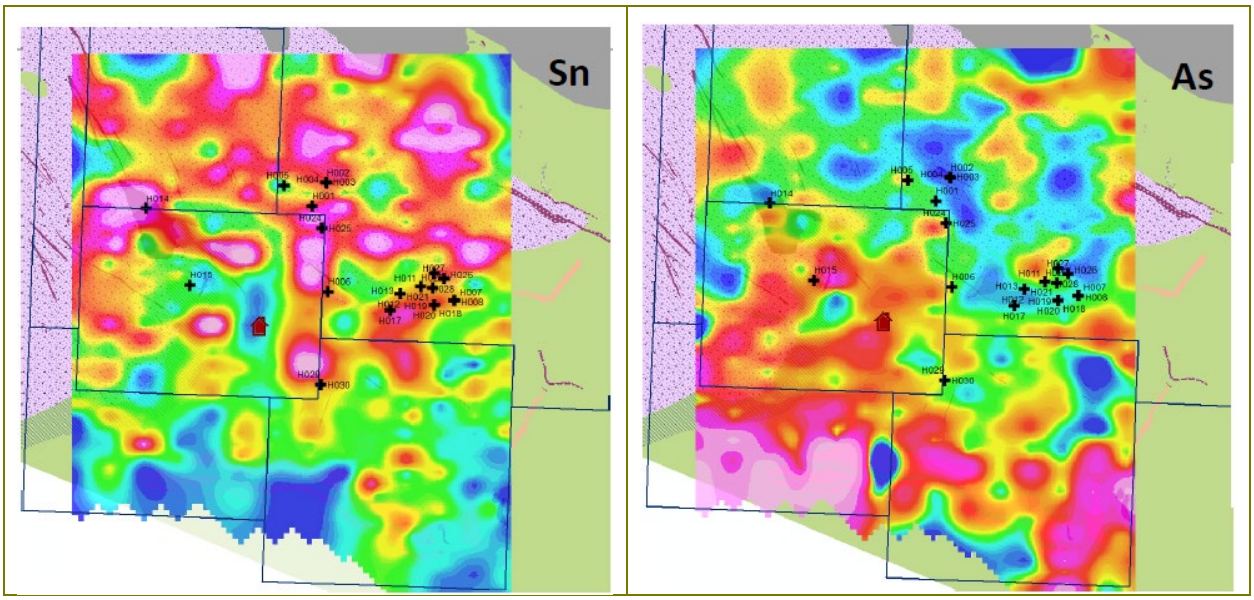


Figure 9-2: Broad Areal Distributions of Sn and As, Offering Geochemical Vectors that Reflect Magmatic Differentiation Processes

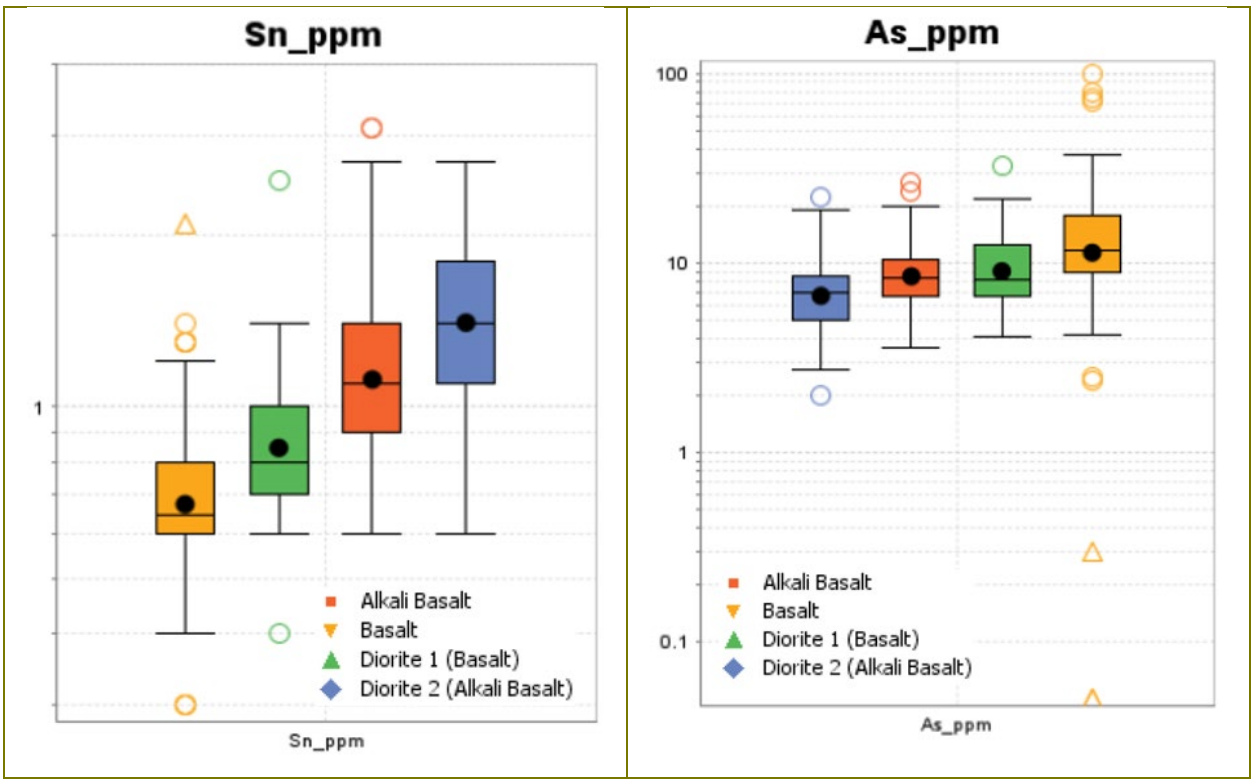


Figure 9-3: Contrasting Sn and As Geochemical Distributions, Each Inversely Correlated with Dioritic Intrusions and Basaltic Volcanic Lithologies

Lithological Controls

- B-horizon soils reflect the underlying intrusive and volcanic lithologies, which range from basaltic to alkaline basaltic compositions.
- Two magmatic phases are distinguishable based on trace-element ratios (high field strength elements [HFSE], Nb/Y) see in Geochemical Framework sub-section, and these phases show distinct metal associations.
- Major-element oxide distributions indicate limited calcic and phyllic alteration within the volcanic package.

Geochemical Framework

High field strength elements (HFSE)—such as Zr (zirconium), Nb (Niobium), Ti (Titanium), Y (Yttrium), and Hf (Hafnium) are highly immobile during hydrothermal alteration and weathering (Gill, 2010; Rollinson, 1993), making them reliable tracers of original magmatic composition. Their ratios record key magmatic processes, including degrees of differentiation, source characteristics, and mineral fractionation, providing a stable geochemical framework even in altered rocks.

HFSE ratios were used to distinguish magmatic phases underlying the soil-sampling area and to interpret associated metal distributions.

The intrusive–volcanic rocks at Hat define two related magmatic phases that share homogeneous zirconium/titanium dioxide and niobium/zirconium ratios indicating derivation from a common parental magma (Figure 9-4). However, the phases are differentiated by significantly distinct niobium/yttrium ratios, reflecting varying degrees of magmatic evolution (Figure 9-5). The use of zirconium/titanium dioxide and niobium/zirconium ratios to distinguish magmatic series follows Winchester and Floyd's (1977) classification scheme.

This geochemical distinction explains the contrasting metal associations observed in soils: copper and gold anomalies are preferentially developed above the less-differentiated intrusive and volcanic units, whereas molybdenum and silver are enriched above the more-evolved magmatic phase. These relationships demonstrate a clear magmatic control on metal zonation, and provide a robust framework for interpreting the thematic maps and vectoring toward mineralized centres within the Lisle Zone system.

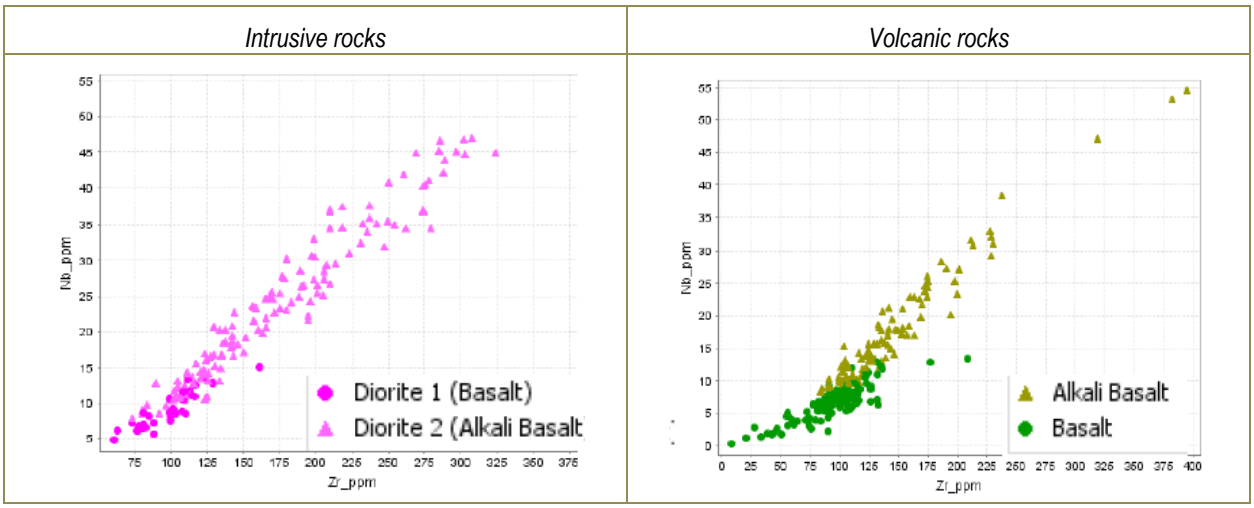


Figure 9-4: Zr/TiO_2 and Nb/Zr Ratio Showing a Primary Magma Origin of the Intrusive and Volcanic Rocks

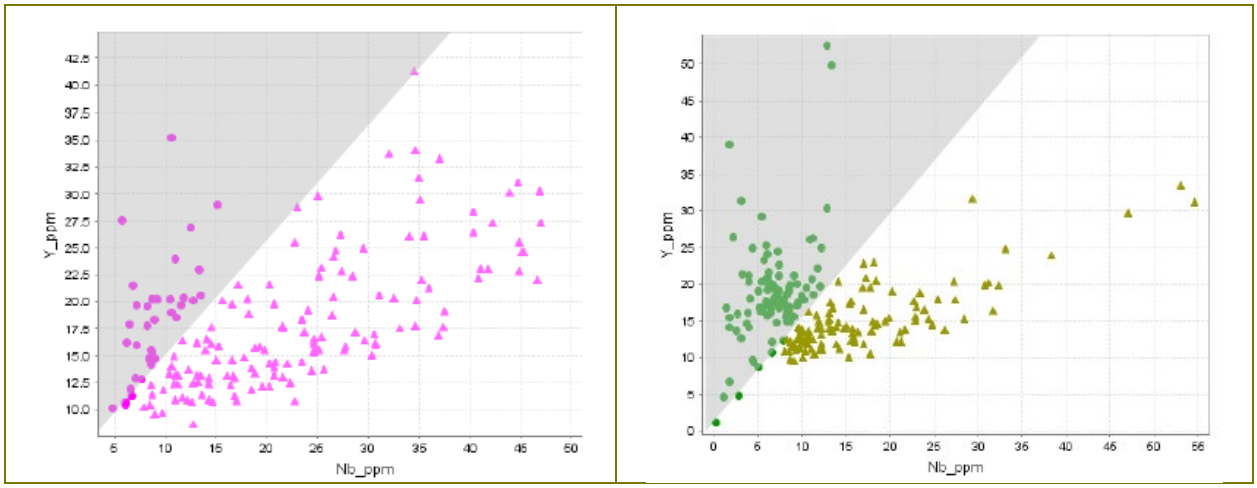


Figure 9-5: Significantly Distinct Nb/Y Ratios, Reflecting Varying Degrees of Magmatic Evolution

Alteration Type Interpretation

Figure 9-6 illustrates two alteration types—calcic alteration (calcite–epidote [Cb–Ep], dolomite–ankerite [Dol–Ank]) and phyllic alteration (muscovite–Illite [Ms–Ill])—each associated, to variable degrees, with copper, gold, silver, and molybdenum mineralization.

1. Cb–Ep
    - Alteration type—calcic (propylitic) alteration
    - Typical minerals—epidote, chlorite, calcite, albite ± actinolite
    - Setting—common in the outer or lower-temperature zones of porphyry systems, especially in mafic or volcanic host rocks

2. Dol–Ank
   - Alteration type—carbonate alteration (carbonate–sericite or carbonate–propylitic)
   - Interpretation:
     - Can represent late carbonate flooding or carbonate-rich propylitic alteration
     - Often associated with fluid–rock interaction in volcanic or sedimentary units.
3. Ms–Ill
   - Alteration type—phyllic alteration.
   - Typical minerals—muscovite/sericite, illite, quartz, pyrite.
   - Setting—intermediate-temperature, acidic fluids; often structurally focused.

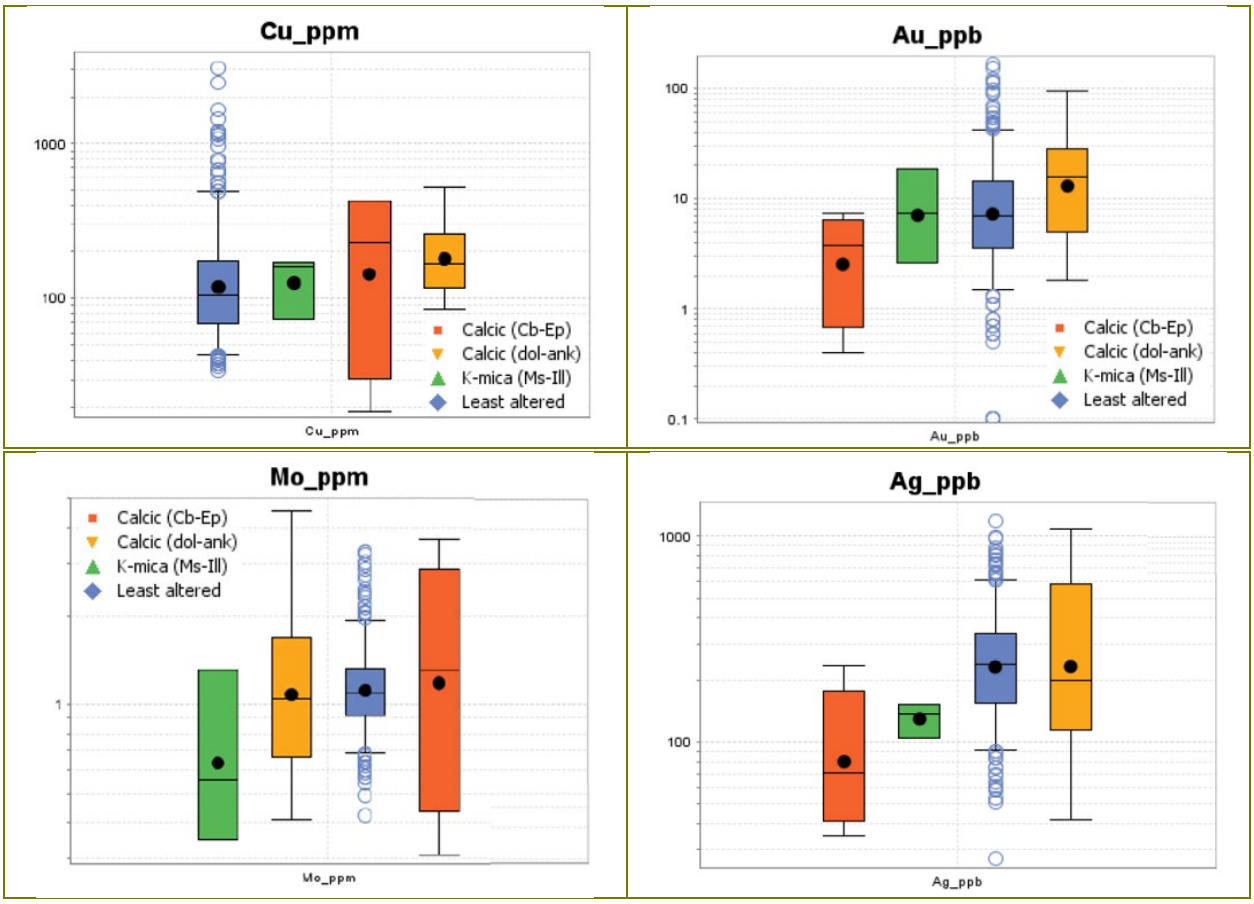


Figure 9-6: Geochemical Distributions (Cu–Mo–Au–Ag) and their Associations with Calcic and Phyllic Alteration

The distribution of major-element oxides indicates only limited development of calcic alteration (Cb–Ep and Dol–Ank assemblages), with phyllic alteration (Ms–Ill) present to a lesser extent, both spatially associated with volcanic lithologies. Copper and molybdenum show broadly overlapping distributions and do not exhibit a strong relationship to specific alteration types, although molybdenum displays a weak association with K-mica. Elevated gold and silver values are preferentially associated with calcic (Dol–Ank)

alteration. In all cases, anomalous base- and precious-metal concentrations occur within the least-altered lithological units.

9.1.4 Soil Exploration Significance

The soil-geochemistry program has:

- Identified new copper-gold anomalies beyond the historically drilled Lisle Zone
- Demonstrated strong lithogeochemical control on metal distribution
- Highlighted tin and arsenic as effective pathfinders for porphyry-style and epithermal-style mineralization, respectively
- Provided geochemical vectors that align with and reinforce the 3-D direct current-induced polarization (DCIP) geophysical anomalies, supporting integrated targeting for future drilling.

Overall, the soil-geochemistry results represent a robust dataset that enhances the geological understanding of the Hat property and provides clear guidance for follow-up exploration.

The Integrated thematic map in Figure 9-1 illustrates the contrasting metal distributions:

- The Lithogeochemistry map is showing copper enrichment associated with the less-differentiated intrusive phases and equivalent volcanic rocks. Anomalous copper values coincide with known mineralization at Lisle Zone and extend into previously untested areas. Gold anomalies broadly overlap the copper-enriched zones and are similarly concentrated within less-evolved intrusive and volcanic units, supporting a magmatic-related metal association.
- Molybdenum exhibits an opposite spatial trend to that of copper and gold, with enrichment preferentially occurring in the more-evolved intrusive phases, indicating contrasting metal behavior during magmatic differentiation. Silver follows a pattern comparable to molybdenum, with elevated values in more-evolved intrusive domains, reinforcing the geochemical divergence between copper–gold and molybdenum–silver associations.

These relationships support the interpretation of magmatic controls on metal zonation within the Lisle Zone system.

9.2 Geophysics

9.2.1 Geological Context

The Lisle Zone represents the principal mineralized horizon at the Hat property. It is hosted within a dioritic intrusive complex emplaced into basaltic to andesitic volcaniclastics of the Stuhini Group. The intrusive is compositionally heterogeneous, with phases ranging from main diorite to micro-monzonite and pyroxenite, reflecting magmatic differentiation. Late dykes crosscut both intrusive and volcanic units but remain geochemically similar to the main diorite.

Hydrothermal alteration is pervasive across both intrusive and volcanic rocks and is dominantly chloritic. Petrographic and geochemical evidence suggest that this alteration largely occurred through direct replacement of alkali feldspars, particularly albite. Drilled core observations from a variety of drill holes described in Section 7.4 validate this interpretation, showing widespread chlorite alteration accompanied by ubiquitous magnetite within the intrusive phases. In several intervals, magnetite occurs in localized, patchy enrichments that are spatially associated with chlorite and sulphide mineralization, including pyrite and chalcopyrite.

Despite these features, the alteration assemblage lacks many of the typical hallmarks of porphyry systems. Sericite, sericite-chlorite, potassic, and propylitic assemblages are largely absent, with the exception of localized halos of K-feldspar alteration that occur adjacent to quartz veins within the dioritic intrusive.

Given the heterogeneous intrusive architecture, the pervasive chlorite–magnetite overprint, and the limited expression of diagnostic porphyry alteration assemblages, additional subsurface constraints are required to refine drill targeting beyond the areas tested to date. Non-invasive geophysical methods, particularly DCIP, provide a practical means of mapping chargeability and resistivity contrasts associated with magnetite, sulphide mineralization, and lithological boundaries across the broader survey area. Integrating DCIP responses with the 2016 drilling results and soil-geochemical anomalies allows high-potential zones to be delineated more confidently prior to committing to further drilling, while minimizing disturbance and optimizing exploration efficiency.

Three-Dimensional Direct Current Induced Polarization

- 3-D DCIP is a geophysical method that measures how the subsurface responds to an applied electrical current. The technique records two key physical properties:
  - Resistivity, which reflects how strongly materials oppose electrical current.
  - Chargeability, which indicates the ability of rocks to temporarily store electrical charge; it is often associated with disseminated sulphides.
- By inverting these measurements in three dimensions, 3-D DCIP produces volumetric models that help identify lithological boundaries, alteration zones, and potential sulphide-bearing targets at depth.

9.2.2 Geophysical Campaigns—Hat Property

Three geophysical campaigns were carried out on the Hat property, providing data for structural interpretation, mineralization placement, alteration mapping, and supporting resource evaluation.

Early Surveys (Historical Context)

- Utah Mining (1978) and Doubleview (2013) conducted initial DCIP surveys.
- These were relatively shallow (<150 m penetration) and limited in coverage, providing only partial anomaly definition.

Hudbay Minerals (2018)

- Commissioned a modern DCIP survey across Hat property (Figure 9-7).
- Approximately 41 line-km of test lines were completed.
- Improved depth penetration (>300 m) compared to earlier surveys.
- Results highlighted chargeability anomalies (>10 mV/V) across the Lisle Zone (Hudbay Minerals Inc., 2018).

DIAS Geophysical (2019)

- Carried out a 3-D DCIP survey using distributed arrays.
- Approximately 37 line-km of test lines were completed.
- Delivered systematic, deep coverage with inversion models for chargeability and resistivity.

Key Outcome

The geophysical campaigns progressively advanced from shallow, incomplete surveys to modern 3-D DCIP coverage. Together, they provided nearly 80 km of test lines, defining chargeability and resistivity anomalies that directly guided drilling in the Lisle Zone, and confirmed its role as the principal mineralized horizon. The interpretation of these results is presented in Section 9.2.3 (DIAS Geophysical Ltd., 2019).

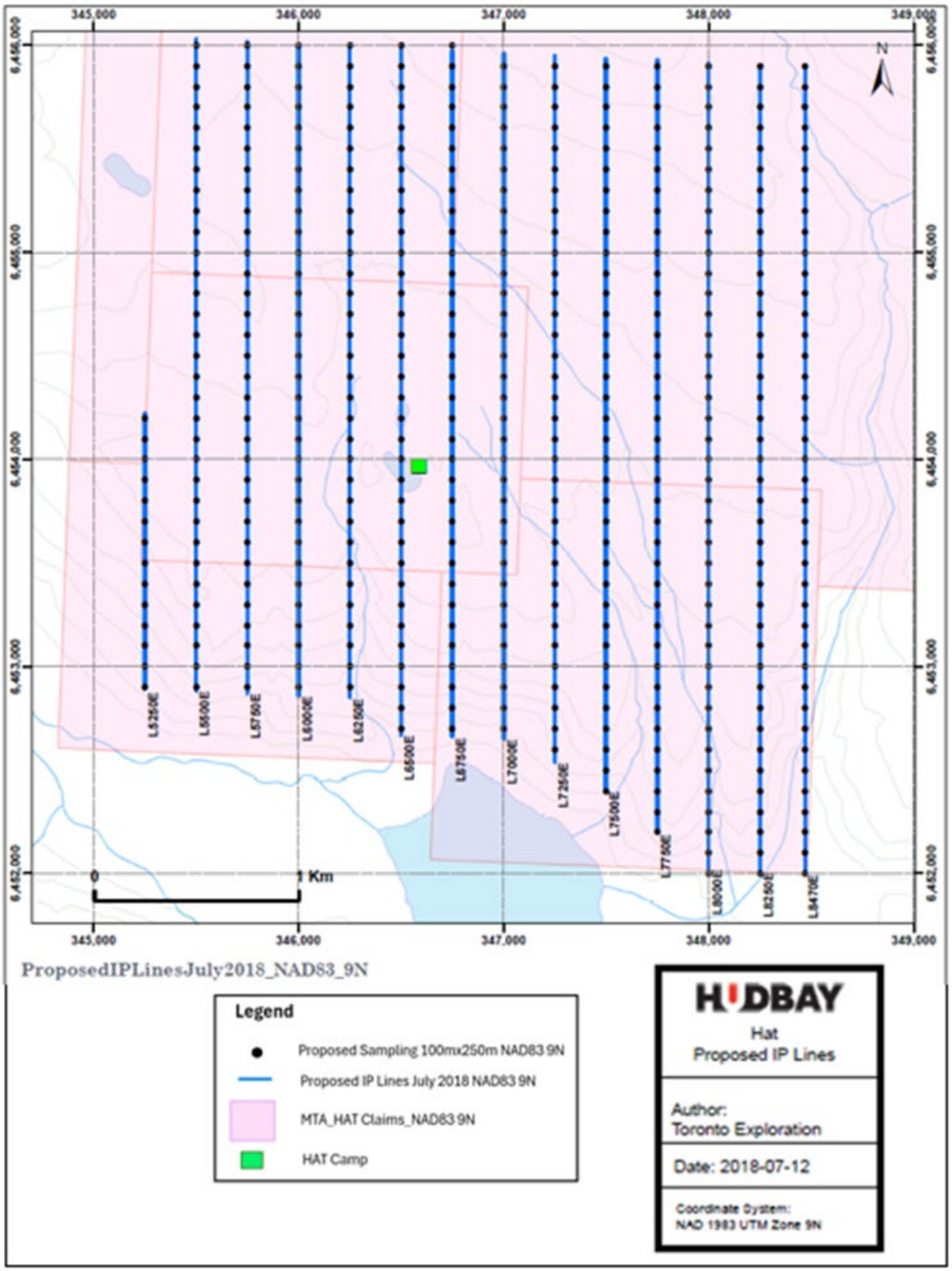


Figure 9-7: Hudbay (2018) DCIP Survey Coverage, Comprising Approximately 41 km of Dipole–Dipole Lines Across the Lisle Zone and Surrounding Targets

9.2.3 Geophysical Signature in Hat Project

Chargeability

As described in Section 9.2.1, DCIP surveys measure chargeability and resistivity responses that help distinguish lithological and alteration domains.

The chargeability results from the 3-D DCIP inversion (Figure 9-8) highlight bulk anomalies exceeding 9 mV/V, which define the mineralized core of the Lisle Zone. These anomalies show a strong correlation with copper and gold assays as well as soil geochemistry, reinforcing their geological significance. Furthermore, hand samples collected in the field (Section9.1), confirmed localized responses that exceed the modelled values, providing additional validation of the inversion results.

Red zones in Figure 9-8 indicate bulk chargeability of approximately >9 mV/V. The low-chargeability blue area delineates a sharp zonal footwall where mineralized sulphides encounter an impermeable boundary between intrusive and volcanic rocks. A group of drill holes confirms the existence of a deeper extension of the sulphide zone trending in a sub-vertical direction. The position of detected sulphide mineralization evolves from horizontal to sub-vertical along the south–north axis.

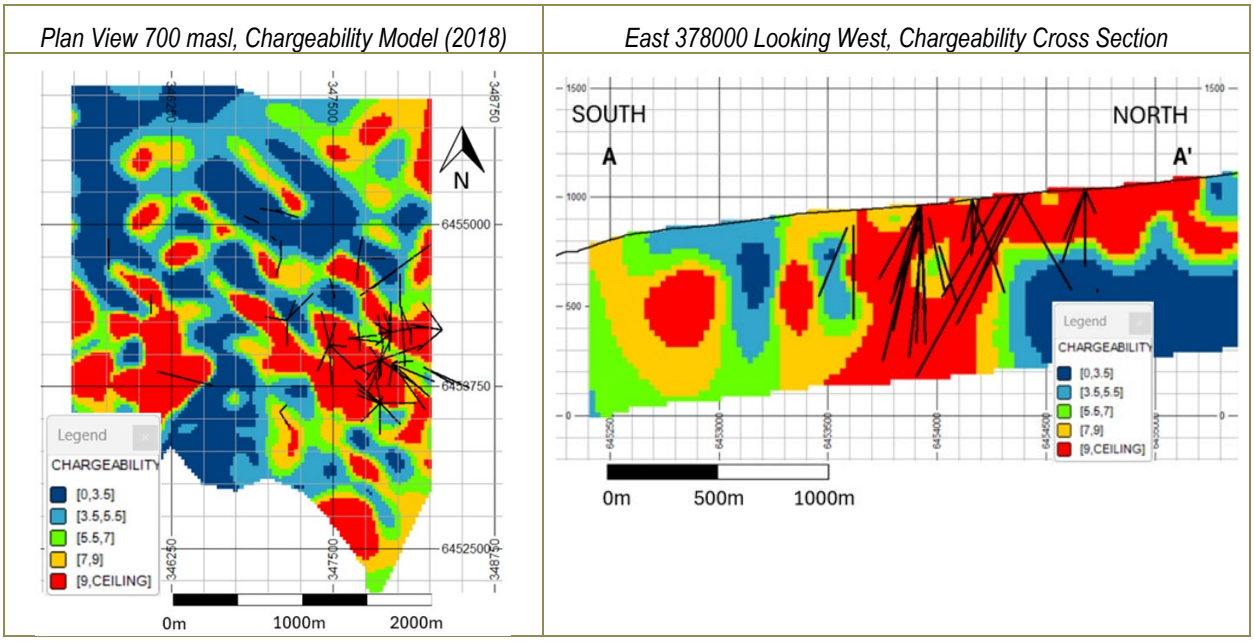


Figure 9-8: Integrated Induced-Polarization and Drilling Data in Plan View and Cross-Section Showing High-Chargeability Domains (>9 mV/V).

Resistivity

The resistivity results in Figure 9-9 indicate that low values (<140 Ohm·m) coincide with zones of potassic alteration and sulphide mineralization. In addition, conductive domains delineate alteration halos and feeder zones, providing a clear geophysical signature of mineralized structures.

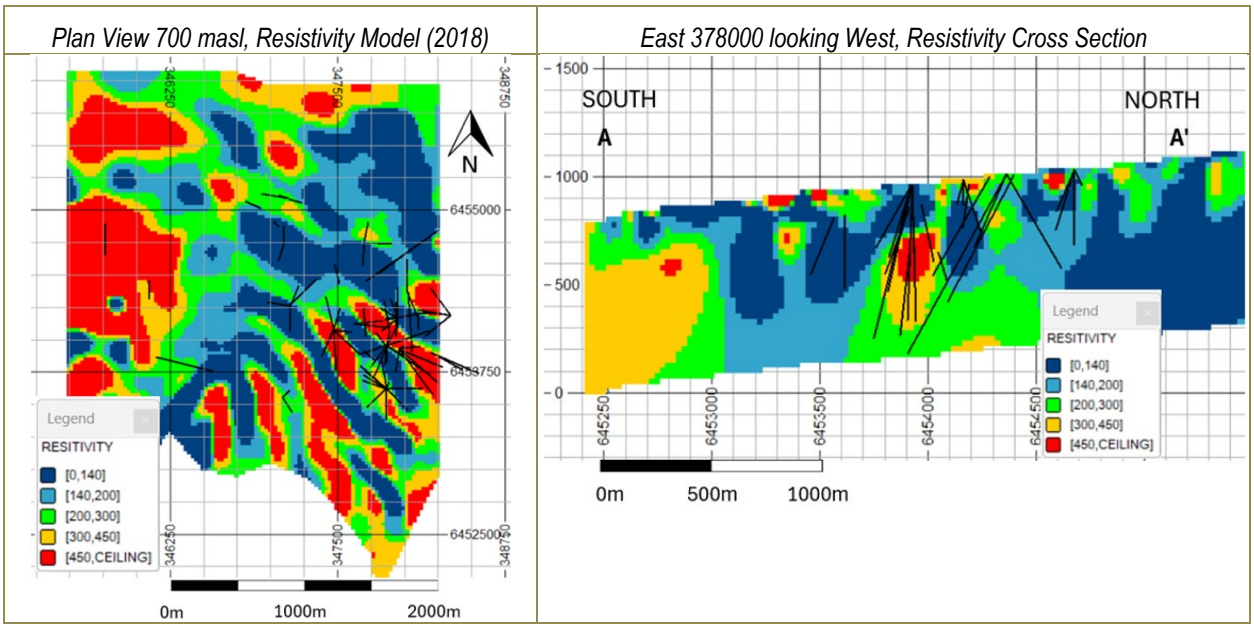


Figure 9-9: Resistivity Inversion Interpretation in Plan View and Cross-Section Showing Low-Resistivity Domains (<140 Ohm·m)

Magnetics

The airborne magnetic survey shows a muted magnetic response over the Lisle Zone compared to the northern diorite (Figure 9-10). Within this framework, magnetic lows correspond to mineralized diorite, whereas magnetic highs mark unmineralized intrusive phases, highlighting the contrast between mineralized and barren domains.

Geochemical Validation

Geochemical indicators reinforce geophysical interpretation. Copper/zinc ratios and the balance between high-temperature and low-temperature metals confirm Lisle Zone as the core mineralized system. Chlorite enrichment is widespread across the property, while potassic alteration is spatially associated with mineralization at depth, strengthening the link between alteration patterns and grade distribution.

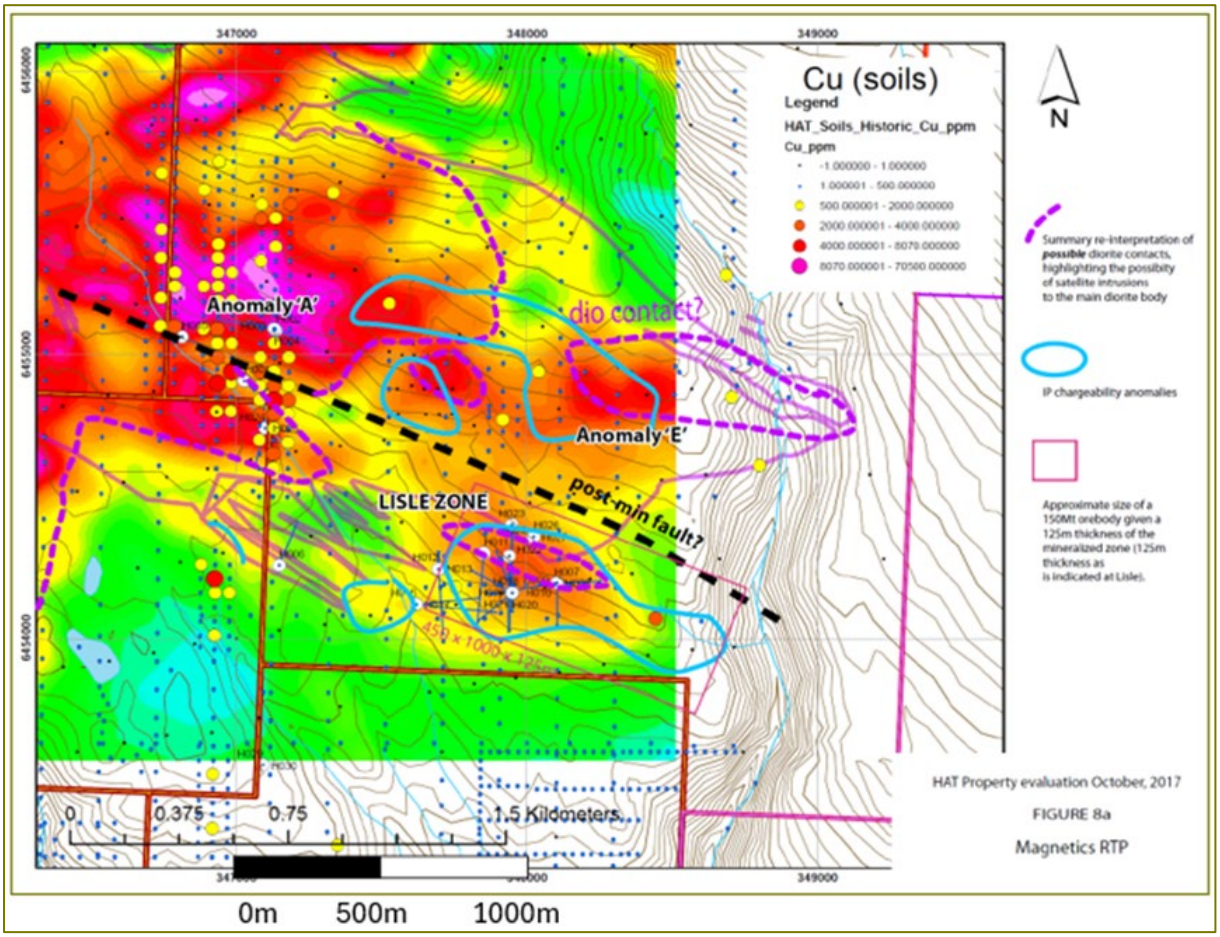


Figure 9-10: RTP Magnetic Map of the Hat Project (Lisle Zone) with Adjacent Northern Prospects

9.2.4 Lisle Zone—Geophysics and Geology Summary

The summary table (Table 9-1) is important because it demonstrates how independent geophysical datasets correlate directly with geology, alteration, and known mineralization in the Lisle Zone. This integration is essential for effective exploration targeting. Each method highlights a different physical property—chargeability, resistivity, and magnetics—and when these responses align with mapped intrusive phases and alteration patterns, they collectively strengthen the geological model.

The correlations presented in Table 9-1 indicate that:

- Chargeability highs (>9 mV/V) coincide with the mineralized dioritic core, confirming the spatial extent of copper–gold mineralization and defining the primary drill-targeting domain.
- Resistivity lows (<140 Ohm·m) map conductive, sulphide-bearing, and altered potassic zones, outlining potential feeder structures consistent with a porphyry system architecture.
- Magnetic lows correspond to altered, mineralized diorite, contrasting with magnetic highs in unmineralized intrusive phases, thereby helping distinguish productive from barren intrusions.

Together, these relationships show that geophysics is not operating in isolation; it is diagnostic of geology and alteration, and therefore directly relevant to exploration decision-making. Table 9-1 captures these cross-disciplinary links in a concise format, demonstrating how integrated geophysical–geological interpretation improves confidence in drill targeting and supports the broader mineral resource development strategy at the Hat Project.

Table 9-1: Integration of Geophysical Responses with Geological Units and Alteration Patterns Relevant to Exploration Targeting

Figure	Dataset/Method	Anomaly Type	Geological Link	Alteration Association	Exploration Implication
Figure 9-11	DCIP 3-D inversion (chargeability)	>9 mV/V highs	Dioritic intrusive core within Stuhini volcanics	Chlorite dominant; localized potassic enrichment in footwall	Defines mineralized core; validates Cu-Au assays; primary drill targeting domain
Figure 9-12	DCIP 3-D inversion (resistivity)	<140 Ohm-m lows	Conductive domains in mineralized diorite	Potassic alteration + sulphide mineralization	Outlines feeder zones; supports porphyry system architecture
Figure 9-13	Airborne magnetics	Magnetic lows over Lisle	Mineralized diorite vs. magnetic highs in unmineralized northern diorite	Alteration reduces magnetization	Refines drill targeting; distinguishes mineralized vs barren intrusive phases

Chargeability anomalies >9 [mV/V] defining the Lisle Zone mineralized core, coincident with copper-gold assays and soil geochemistry (Figure 9-11). Chlorite alteration dominates, with localized potassic enrichment in the footwall.

Resistivity lows (<140 Ohm-m) outlining conductive domains associated with sulphide mineralization and potassic alteration at Lisle Zone (Figure 9-12) delineate feeder zones within dioritic intrusive rocks.

Unconstrained magnetic inversion in Figure 9-13 reveals the geometry and distribution of magnetically susceptible bodies (e.g., magnetite or pyrrhotite). Magnetic prospection clipped > 0.01 [SI] highlights magnetically significant sources. In the Hat Project, magnetic lows over the Lisle Zone correspond to mineralized diorite, in contrast to magnetic highs observed in the unmineralized northern diorite. Integration of the magnetic response with chargeability anomalies provides improved definition of priority drill targets.

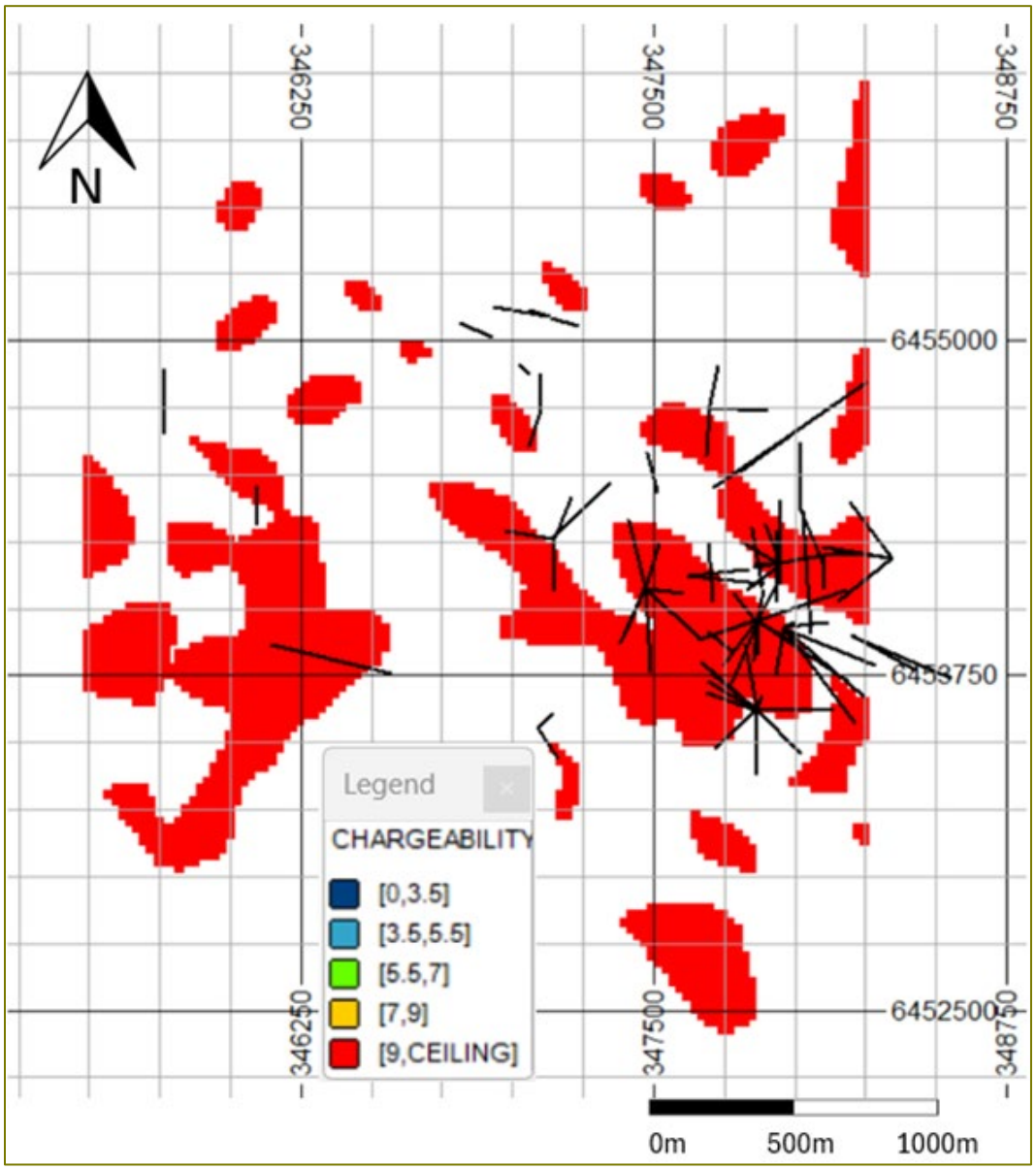


Figure 9-11: Elevation 700 masl; Chargeability Anomalies (>9 mV/V) Defining the Lisle Zone Mineralized Core

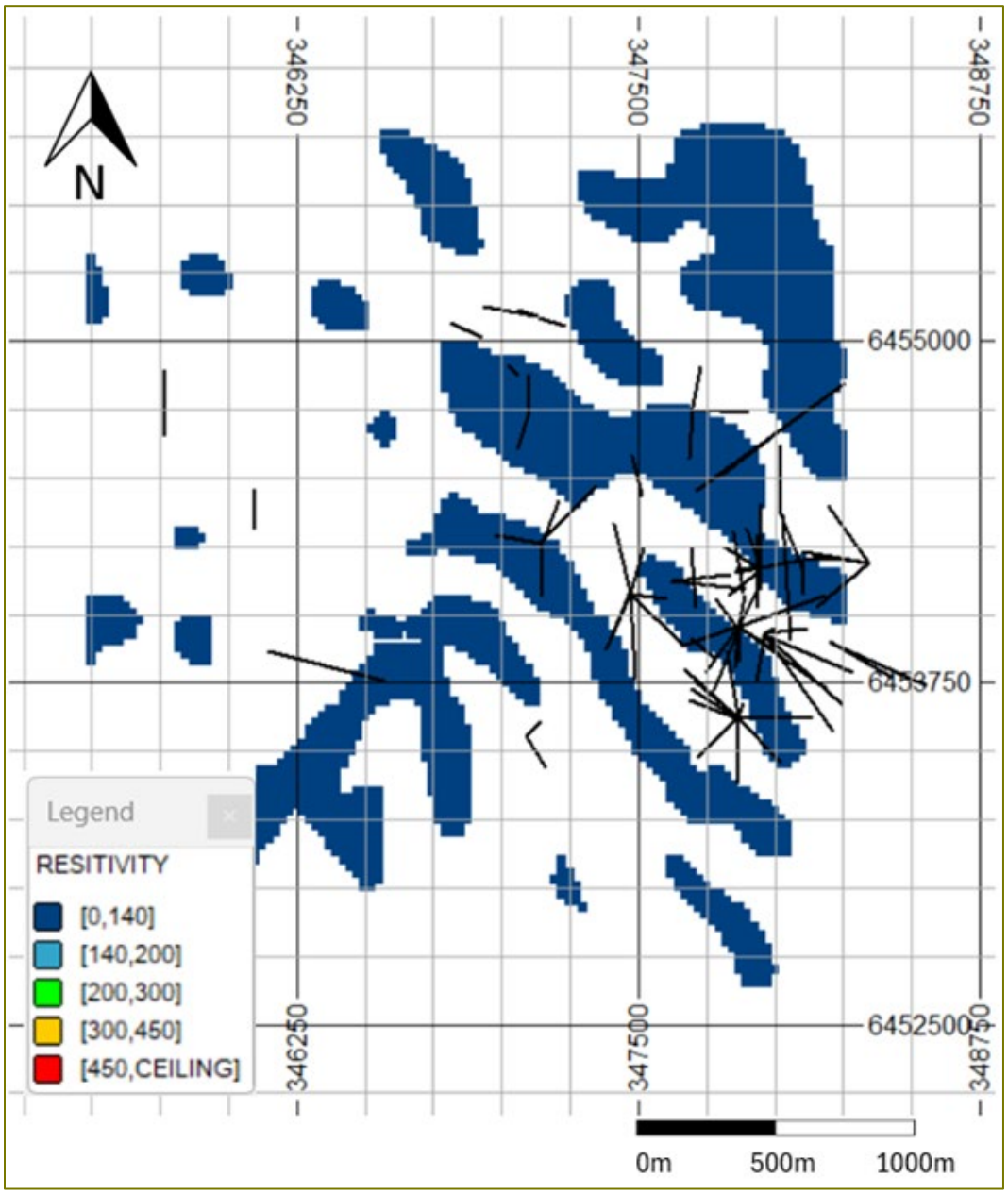


Figure 9-12: Elevation 700 masl; Resistivity Lows (<140 Ohm m) Outlining Conductive Domains Associated with Sulphide Mineralization

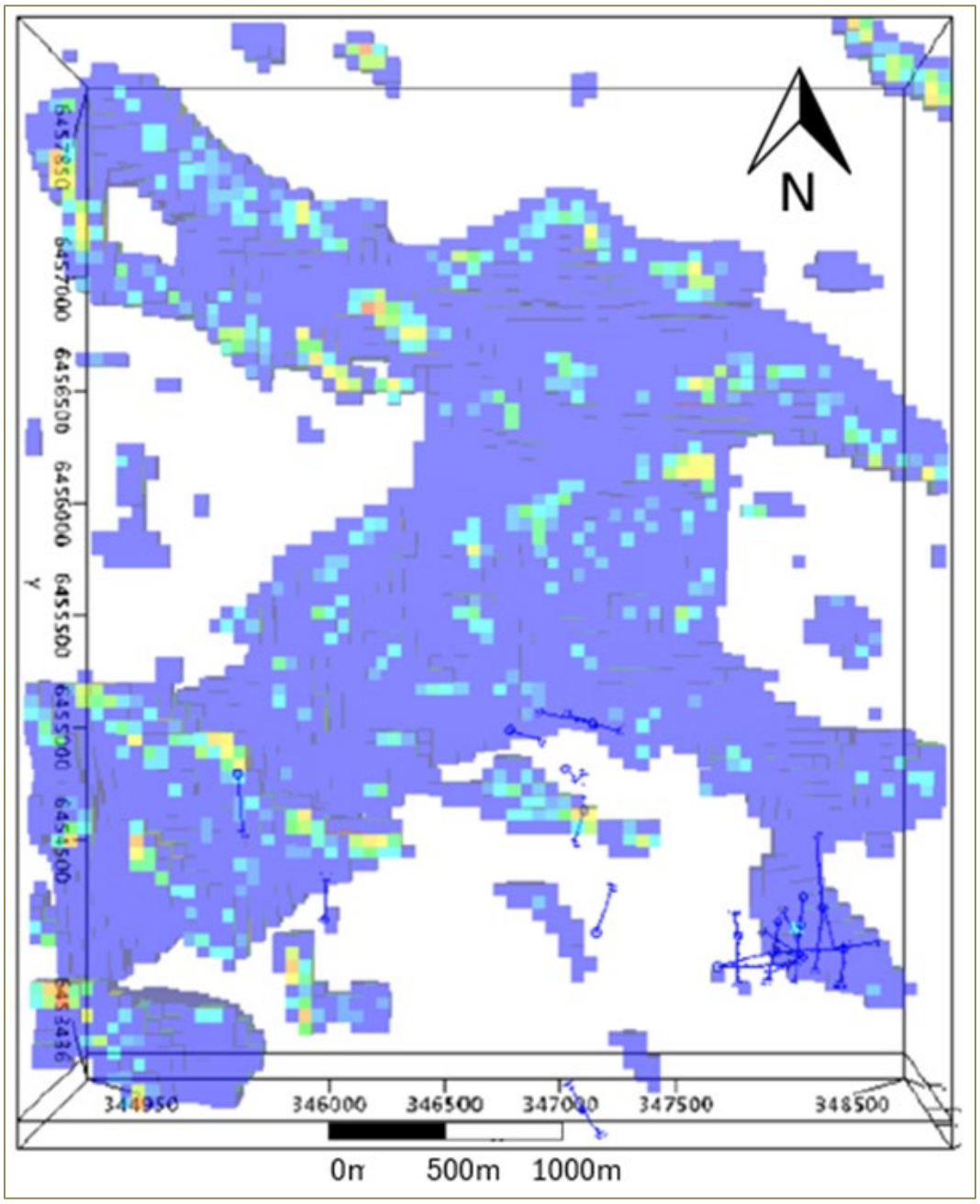


Figure 9-13: Unconstrained Magnetics Inversion Clipped >0.01 SI Detects Magnetic Highs Observed in the Unmineralized Northern Diorite

10 DRILLING

Drilling completed at the Hat Project up to September 30, 2025, comprises 101 diamond drill holes, of which 97 constitute the geological and assay database, totaling 49,548 m. Doubleview, the owner and operator of the Hat Project, has been responsible for all exploration drilling campaigns since 2013. Four drill holes (H010, H044, H048, and H084) were not sampled and are therefore excluded from the database. The early drilling programs were designed to test geophysical anomalies, whereas the more recent campaigns focused on validating the 3-D geological model, confirming the presence of mineralization, and delineating the geometry of the porphyry copper-gold system. All drilling has been completed using NQ and HQ diamond core sizes.

The annual drilling progress summarized in Section 10.4 and reflects the progressive expansion of the exploration program, with significant increases in drilling productivity from 2023 onward.

10.1 Drilling Methods

Diamond drilling was conducted using HQ and NQ core sizes, both employing standard tubes. The bit and core diameters are shown in Table 10-1.

Table 10-1: Diamond Drill Rod and Core Diameter

Drill Size	Bit (mm)	Core (mm)
HQ	96.0	63.5
NQ	75.7	47.6

Mixed core diameters were first introduced in 2020 with the drilling of H035. Prior to this, all drill holes up to and including H034 were completed exclusively using NQ-diameter core. No drilling occurred between 2020 and 2025 when dual-core drilling was reinitiated; six additional drill holes were completed using a combination of HQ and NQ core sizes (Table 10-2). Across the entire Hat Project drill program, HQ core represents approximately 6.4% of the total drilled metreage, while within the mixed-diameter drilling times, the average proportion of HQ core was 58.5%.

Drilling was executed in accordance with planned collar coordinates, initial azimuth, and dip parameters to intersect the target zones at the required depths. Drill pads were constructed at proposed locations; however, due to uneven terrain, the final pad position occasionally deviated by several metres from the planned coordinates. These deviations were subsequently corrected through precise collar surveying.

Table 10-2: Summary of Drill Holes Completed with Multiple Core Diameters, Reporting Total Depth and Distribution of HQ and NQ Drilling

Hole ID	From (m)	To (m)	Length (m)	Year	HQ (m)	NQ (m)	% HQ
H035	0	495	495	2020	453.9	41.1	91.7
H092	0	741	741	2025	513.5	227.5	69.3
H095	0	762	762	2025	495.0	267.0	65.0
H096	0	922.5	922.5	2025	501.4	421.1	54.4
H097	0	639	639	2025	201.3	437.8	31.5
H100	0	840	840	2025	501.6	338.5	59.7
H101	0	1,015.45	1,015.45	2025	501.5	514.0	49.4
				Total	**3,168.1**	**2,246.9**	**58.5**

10.2 Collar and Downhole Survey

10.2.1 Horizontal UTM Coordinates

Accurate collar positioning is essential for reliable geological interpretation and resource modelling. The drill pads before 2023 were surveyed by handheld GPS and later by using both of a drone equipped with RTK georeferencing and handheld GPS when it's applicable. This method provides high-precision spatial control suitable for resource-grade drilling.

Georeferenced imagery from the drone survey is used to confirm the final collar location. Updated azimuth readings and collar coordinates are recorded and stored in a structured database containing:

- Hole ID
- Target depth
- Azimuth and inclination
- X, Y, Z collar coordinates
- Start and end dates
- Drilling contractor
- Logging geologist.

This workflow ensures traceability, consistency, and data integrity throughout the drilling program.

10.2.2 Downhole Survey

Downhole trajectory measurements are collected using a gyroscope-based survey system. For the Hat Project, the selected instrument is the Icefield Tools Corporation Gyro Shot. The device is a robust, non-north-seeking gyroscope requiring initial orientation using an external reference point.

Key characteristics include:

- Multi-shot operating mode
- High magnetic immunity, suitable for magnetically disturbed environments
- Recording of relative azimuth and inclination
- Internal memory storage for secure data capture.

Survey measurements are typically collected from the bottom upward along the drill hole. All readings are exported to an Excel spreadsheet and linked to the corresponding drill hole ID for integration into the master database.

A review of the survey database indicates that 64% of the total drilled metres have downhole survey coverage; 36% lack survey data, predominantly from the earlier drilling campaigns. A discrepancy analysis was completed to assess the potential impact of missing surveys. The results of this analysis are presented in Section 12.5.

10.3 Core Recovery and Rock Quality Designation Data Sets

Systematic core recovery (CR) and rock quality designation (RQD) data were collected for all diamond drill holes included in this program. For each interval, CR and RQD were recorded in metres, and as a percentage of the nominal run length (typically ~3 m). Interval numbering was maintained per hole so that depth sequences and transitions can be traced directly in the geotechnical model.

Across the data set, core recovery is generally very high. Most intervals report CR% in excess of 95%, and many record 100% recovery. Lower-recovery intervals do occur but are short and typically associated with highly fractured, blocky, or weathered zones. The combination of high overall recovery and detailed RQD logging provides a robust and internally consistent basis for assessing rock mass discontinuity intensity and structural condition.

In the master database, recovery data for 16 drill holes is currently absent: H003, H006 to H013, H017 to H022, and H030. These drill holes do exist, and their geological logging, photographic records, and assay data are all available. Given the value of recovery information for geotechnical interpretation and core quality assessment, the missing recovery data should be reconstructed and incorporated so that these drill holes become a complete and integral part of the database.

The RQD data show the expected variability for a structurally complex intrusive system. RQD values range from 0% to near 100%, with patterns that are coherent within and between holes. On this basis, the drilling can be subdivided into three groups that capture the main structural and rock mass condition domains observed in the data set: highly fractured domain; a transitional, moderately fractured domain; and a more massive, less fractured domain.

10.3.1 Group 1—Low Rock Mass Quality, Highly Fractured Domain

Group 1 comprises intervals characterized by reduced core recovery and low RQD, predominantly in the upper portions of several holes. These intervals are most extensively developed in the upper parts of holes H065, H066, H070, and H071, and as scattered sections within some other holes such as H060 and H076. Runs in this group commonly show CR between about 1.0 and 2.3 m (~60%–85%), with RQD typically below 40% and frequently below 25%. Zero-RQD intervals are recorded in places, usually over short lengths.

The RQD distribution for Group 1 is strongly skewed to low values: mean RQD is on the order of 25%–35%, with a substantial proportion of intervals below 20% and very few above 60%. Despite the low RQD, core recovery is generally adequate, which confirms that the low RQD values represent true rock mass fragmentation rather than core loss.

Geologically and mechanically, Group 1 is interpreted as a near-surface or structurally disturbed domain where weathering, shearing, and intensive jointing have degraded the rock mass. This domain is expected to govern local stability and support requirements in areas where it is exposed or intersected by excavations. It forms a thin to moderate-thickness cap that grades downward into better-quality rock.

10.3.2 Group 2—Transitional, Moderate Rock Mass Quality Domain

Group 2 comprises intervals with generally high core recovery but moderate and variable RQD. It is the dominant rock mass condition in the data set and occurs through much of H060, H065–H072, H075, and H076, particularly in their mid-depth sections. In these intervals, CR is typically between 2.4 and 2.8 m (~90%–98%), while RQD ranges from about 30% to 70%, with mean values in the 50%–60% range.

The RQD distribution for Group 2 is more balanced than in Group 1: low-RQD intervals (<30%) are present but less frequent, and many intervals fall in intermediate categories (40%–70%). High-RQD intervals (>80%) occur but are not continuous over long lengths. This pattern indicates a rock mass that is moderately fractured, intersected by a moderate density of joints, fractures, and small faults.

Group 2 is interpreted as a transitional mechanical domain between the highly fractured, near-surface or faulted zones and the deeper, more massive intrusive units. It represents intact to moderately fractured rock, but fracture frequency and orientation must still be considered in design. This domain is likely to be the main contributor to overall slope and excavation behaviour in much of the deposit area.

10.3.3 Group 3—High Rock Mass Quality Domain

Group 3 includes intervals with uniformly excellent recovery and high RQD, representing the strongest and most structurally intact rock mass in the drilling. This domain is best expressed in the deeper parts of H065–H072 and in holes such as H075, H076, H090–H092, and other deeper or structurally less-disturbed holes. Intervals in this group typically record CR close to the full run length (2.8–3.0 m, ~98%–100%), and RQD commonly falls in the 70%–95% range, with frequent values above 80%.

The RQD distribution for Group 3 is strongly clustered at high values: mean RQD in this domain is on the order of 80%–90%, with only occasional intervals dropping below 60%. Low-RQD values, where present,

are short and isolated, and they almost always occur with high CR%, again indicating that they reflect genuine localized fracturing rather than sampling bias.

Group 3 is interpreted as a competent intrusive domain where the rock mass is dominated by long intact blocks, widely spaced joints, and limited structural disturbance. This domain is expected to support relatively steep slopes and to exhibit good stand-up time in underground openings, subject to local structural conditions, stress environment and joint orientation, and to local modification by faults and other structures.

10.3.4 Implications for Geotechnical Characterization

The three-group framework provides a practical basis for geotechnical characterization by distinguishing rock mass domains with different degrees of fracturing and structural disturbance:

- Group 1 represents the most-fractured rock mass condition and is expected to control conservative design considerations where present, particularly in near-surface or structurally disturbed zones, influencing slope geometry, bench configuration, and local support requirements.
- Group 2 represents the dominant rock mass condition across the deposit and will underpin most domain-scale rock mass classification (e.g., RMR and Q-system) and parameter selection. In this domain, excavation and slope behavior are governed primarily by fracture frequency, joint orientation, and joint condition rather than intact rock strength alone.
- Group 3 defines the least fractured and most structurally intact rock mass condition, where favourable stand-up time and steeper slope geometries or reduced support demands may be achievable, subject to local structural features and in situ stress conditions.

Given the consistently high core recovery, the RQD data are considered reliable indicators of rock mass discontinuity intensity. RQD is used as a supporting input to rock mass classification and geotechnical modelling, and is interpreted in conjunction with geological logging, joint condition, intact rock strength, and structural data in the derivation of design parameters.

10.4 Drilling Results

Exploration drilling has successfully delineated a large porphyry copper–gold system. The mineralized volume extends approximately 2 × 2 km laterally and to 900 m deep, consistent with a disseminated porphyry mineralization style. Drilling results have progressively refined the geological model, confirmed the continuity of mineralized zones, and provided the basis for Mineral Resource estimation presented in Section 14.

Figure 10-1 illustrates the spatial distribution of drill holes across the Lisle Zone, the central area of the Hat Project.

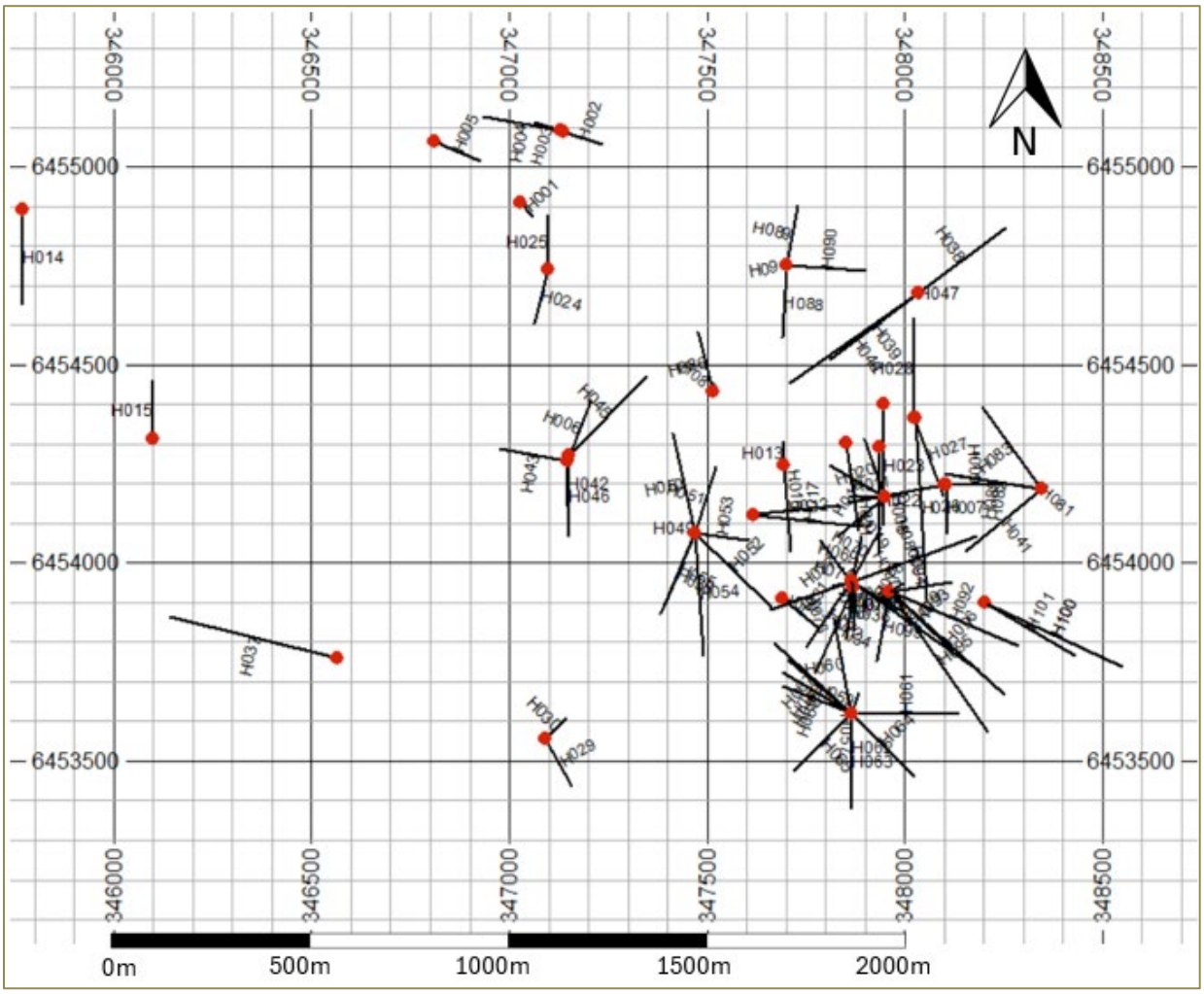


Figure 10-1: Plan View Showing Drill-Hole Locations Within the Hat Project (UTM Zone 9N, NAD 83)

Table 10-3 presents drill-hole assay results on a year-by-year basis to illustrate the expansion of the exploration program and the increasing geological understanding of the Hat Project. It is important to note that drilling campaigns were not always executed in an optimal configuration for geological modelling or Mineral Resource estimation. Due to permitting constraints, logistical limitations, and the high cost of relocating drill rigs, drilling was frequently conducted in a fan pattern, with multiple drill holes of varying azimuths and dips collared from the same drill pad. While operationally efficient, this approach results in uneven spatial distribution of drill-hole orientations and variable sampling density, factors that must be considered when interpreting grade continuity and constructing mineralized domains.

Table 10-3: Yearly Drilling Summary

Year	Drill Holes	Total Length (m)
2013	10	2,704
2014	12	5,125
2015	2	553
2016	5	2,345
2019	4	1,843
2020	3	2,249
2021	4	2,476
2022	8	3,433
2023	20	10,494
2024	18	9,800
2025	12	8,523

Between 2013 and 2025, drilling advanced from small reconnaissance-scale campaigns to large, systematic programs exceeding 10,000 m in 2023. Early drilling (2013–2016) focused on testing geophysical anomalies and establishing the initial geological framework, while later campaigns (2019–2025) targeted delineation of the porphyry system and refinement of the 3-D mineralized volume. The substantial increase in drilling from 2023 onward reflects improved operational capacity and a more advanced stage of exploration. The spatial distribution and temporal progression of drilling activities are illustrated in Figure 10-2 and Figure 10-3, where all drill holes are colour-coded by drilling year.

Across all years, the total drilled length is 49,885.6 m, with annual contributions ranging from 553 m in 2015 to a maximum of 10,494 m in 2023. The total length of drill holes considered in the mineral resource estimation and classification is 49,548 m. The cumulative increase in drilling density from 2013 to 2025, as summarized in Table 10-3, has materially enhanced spatial coverage and supports robust statistical analysis of grade distributions and the delineation of mineralized domains.

The collar coordinates and hole-orientation parameters for all drill holes included in the mineral resource estimation and classification are provided in Table 10-4.

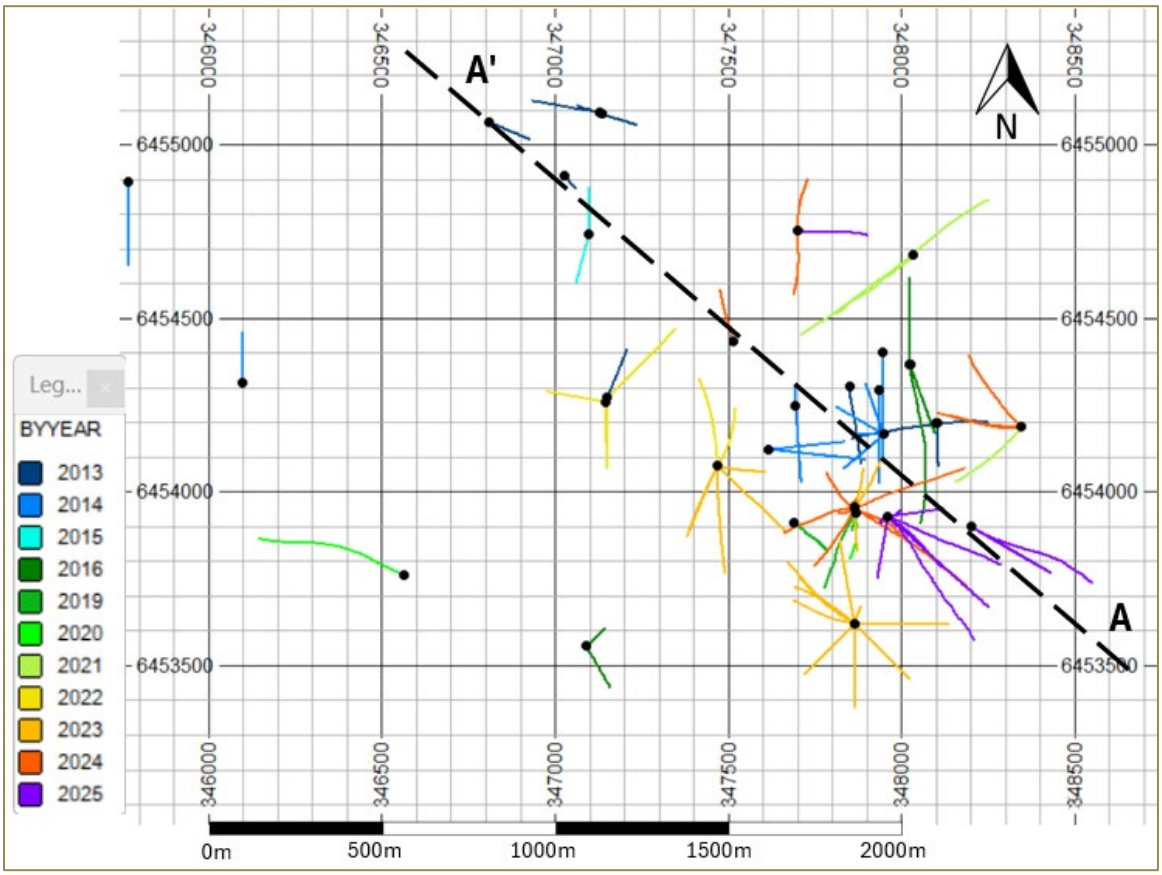


Figure 10-2: Plan View of All Drill Holes from the Hat Project, by Year Drilled (UTM Zone 9N NAD 83)

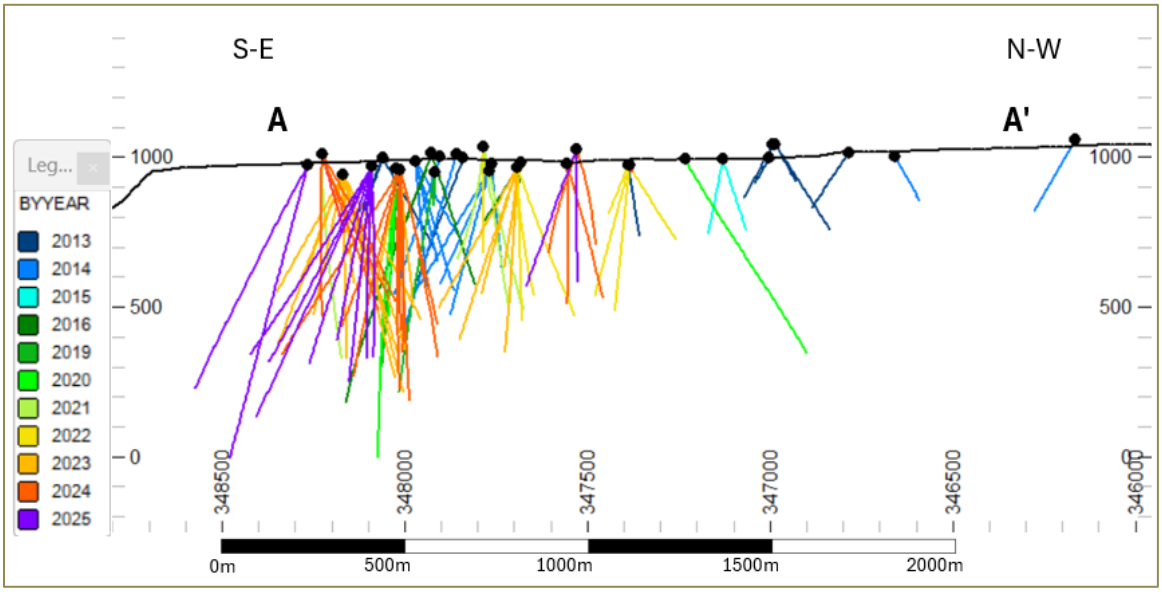


Figure 10-3: View Southwest, Cross-Section Displaying 2-D-Projected Drill Holes for the Hat Project (UTM Zone 9N NAD 83)

Table 10-4: Drill-Hole Collar Coordinates and Hole Parameters (UTM Zone 9N NAD 83)

Hole	Easting (m)	Northing (m)	Elevation (m)	Azimuth (°)	Dip (°)	Depth (m)	Drill Year
H001	347028.0	6454908.0	997.5	137	60	102	2013
H002	347137.0	6455085.0	1,041.5	107	60	205	2013
H003	347131.0	6455090.0	1,041.5	287	60	142	2013
H004	347131.0	6455090.0	1,042.0	280	55	349	2013
H005	346814.0	6455062.0	1,010.5	113	55	218	2013
H006	347148.0	6454257.0	972.7	22	55	288	2013
H007	348105.0	6454195.0	995.8	180	50	231	2013
H008	348104.0	6454195.0	995.8	260	55	404	2013
H009	348104.0	6454195.0	995.8	88	60	301	2013
H011	347852.0	6454300.0	993.8	180	55	463	2013
H012	347697.0	6454245.0	975.5	176	65	549	2014
H013	347697.0	6454245.0	975.5	0	80	350	2014
H014	345770.0	6454890.0	1,057.0	180	45	337	2014
H015	346100.0	6454310.0	1,001.5	0	45	209	2014
H016	347620.0	6454120.0	953.0	84	60	437	2014
H017	347620.0	6454120.0	953.0	96	55	482	2014
H018	347951.0	6454164.0	985.1	300	60	319	2014
H019	347951.0	6454164.0	985.1	230	70	446	2014
H020	347951.0	6454164.0	985.1	340	70	458	2014
H021	347951.0	6454164.0	985.1	270	75	346	2014
H022	347939.0	6454289.0	998.0	180	60	527	2014
H023	347949.0	6454400.0	1,008.0	180	63	665	2014
H024	347100.0	6454740.0	990.0	195	60	282	2015
H025	347100.0	6454740.0	990.0	0	60	271	2015
H026	348028.0	6454365.0	1,011.5	180	65	953	2016
H027	348026.0	6454365.0	1,011.5	160	65	500	2016
H028	348026.0	6454365.0	1,011.5	0	60	505	2016
H029	347093.0	6453553.0	979.0	150	55	236	2016
H030	347093.0	6453553.0	979.0	45	60	151	2016
H031	347692.0	6453908.0	947.0	120	80	741	2019
H032	347692.0	6453908.0	947.0	190	85	111	2019
H033	347870.0	6453938.0	959.0	190	80	289	2019
H034	347870.0	6453938.0	959.0	220	70	701	2019
H035	347870.0	6453938.0	959.0	175	85	495	2020
H036	347870.0	6453938.0	959.0	180	80	972	2020
H037	346567.6	6453758.0	991.0	294	60	782	2020
H038	348035.0	6454681.0	1,034.0	45	58	463	2021
H039	348035.0	6454681.0	1,034.0	222	61	615	2021
H040	348035.0	6454681.0	1,034.0	233	55	675	2021
H041	348348.0	6454186.0	1,008.0	217	70	723	2021

Hole	Easting (m)	Northing (m)	Elevation (m)	Azimuth (°)	Dip (°)	Depth (m)	Drill Year
H042	347148.0	6454257.0	972.5	180	55	201	2022
H043	347148.0	6454257.0	972.5	280	55	304	2022
H045	347151.0	6454269.0	972.0	45	60	562	2022
H046	347151.0	6454269.0	972.0	180	65	482	2022
H047	348035.0	6454681.0	1,034.0	0	90	352	2022
H049	347471.0	6454073.0	964.0	0	87	513	2022
H050	347471.0	6454073.0	964.0	0	59	558	2022
H051	347471.0	6454073.0	964.0	30	68	462	2022
H052	347471.0	6454073.0	964.0	135	61	540	2023
H053	347471.0	6454073.0	964.0	105	71	442	2023
H054	347471.0	6454073.0	964.0	180	59	655	2023
H055	347471.0	6454073.0	964.0	208	70	645	2023
H056	347471.0	6454073.0	964.0	210	56	399	2023
H057	347866.0	6453619.0	938.8	270	85	189	2023
H058	347866.0	6453619.0	938.8	311	73	750	2023
H059	347866.0	6453619.0	938.8	20	85	614	2023
H060	347866.0	6453619.0	938.8	350	70	717	2023
H061	347866.0	6453619.0	938.8	90	65	642	2023
H062	347866.0	6453619.0	938.8	180	70	495	2023
H063	347866.0	6453619.0	938.8	180	55	420	2023
H064	347866.0	6453619.0	938.8	135	60	453	2023
H065	347866.0	6453619.0	938.8	225	60	414	2023
H066	347866.0	6453619.0	938.8	280	71	621	2023
H067	347866.0	6453619.0	938.8	300	60	549	2023
H068	347866.0	6453619.0	938.8	310	69	618	2023
H069	347866.4	6453953.8	956.5	18	74	459	2023
H070	347866.4	6453953.8	956.5	48	70	462	2023
H071	347865.9	6453951.5	956.5	175	71	414	2023
H072	347865.9	6453951.5	956.5	212	75	761	2024
H073	347865.9	6453951.5	956.5	225	87	771	2024
H074	347865.9	6453951.5	956.5	262	85	609	2024
H075	347865.9	6453951.5	956.5	0	88	537	2024
H076	347865.9	6453951.5	956.5	262	69	660	2024
H077	347865.9	6453951.5	956.5	300	75	531	2024
H078	347865.9	6453951.5	956.5	120	75	708	2024
H079	347865.9	6453951.5	956.5	120	58	732	2024
H080	347865.9	6453951.5	956.5	70	56	624	2024
H081	348348.0	6454186.0	1,008.0	0	88	537	2024
H082	348348.0	6454186.0	1,008.0	270	65	558	2024
H083	348348.0	6454186.0	1,008.0	315	59	552	2024
H085	348348.0	6454186.0	1,008.0	260	63	543	2024
H086	347518.0	6454430.0	976.0	345	71	471	2024

Hole	Easting (m)	Northing (m)	Elevation (m)	Azimuth (°)	Dip (°)	Depth (m)	Drill Year
H087	347518.0	6454430.0	976.0	0	87	465	2024
H088	347703.0	6454749.0	1,025.0	180	62	390	2024
H089	347703.0	6454749.0	1,025.0	0	62	351	2024
H090	347703.0	6454749.0	1,025.0	90	65	501	2025
H091	347703.0	6454749.0	1,025.0	0	88	441	2025
H092	347963.0	6453927.0	966.0	120	62	741	2025
H093	347963.0	6453927.0	966.0	127	72	693	2025
H094	347963.0	6453927.0	966.0	77	75	597	2025
H095	347963.0	6453927.0	966.0	137	57	762	2025
H096	347963.0	6453927.0	966.0	132	65	923	2025
H097	347963.0	6453927.0	966.0	348	86	639	2025
H098	347963.0	6453927.0	966.0	251	87	633	2025
H099	347963.0	6453927.0	966.0	190	76	738	2025
H100	348203.0	6453897.0	972.0	120	61	840	2025
H101	348203.0	6453897.0	972.0	120	75	1,015	2025

10.4.1 Hat Deposit Relevant Intercepts

Porphyry copper–gold deposit, its geological nature and inherent variability do not allow barren gaps to be left unexamined, nor mineralization to be extended beyond its natural limits. To ensure the required transparency and materiality, the QP provides all necessary displays and supporting evidence with the purpose of leaving no room for doubt.

Database Regularization

The raw sample intervals were regularized into a drill-hole database in which all samples, both mineralized and barren were composited to a uniform length of 10 m. Each 10 m composite already incorporates and quantifies internal waste, and the reported grade reflects the combined effect of both mineralized and barren material within that interval.

This regularized length-grade dataset is the basis for domain modelling and Mineral Resource estimation. Because internal waste is already included during compositing, the process does not selectively remove barren material, nor does it artificially widen mineralized zones. The influence of any individual raw sample, including isolated high-grade values, is constrained by the 10 m composite length.

Drilling Results

Four metals—copper, gold, silver, and cobalt—are considered the primary economic contributors to the Hat Project. Their mineralized domains vary spatially and do not always coincide along the same downhole intervals. To avoid divergence in reporting separate intercepts for each metal, a unified economic indicator was applied. Relevant intercepts are therefore presented using Copper Equivalent (CuEq), which consolidates the four metal components into a single comparable metric.

CuEq was calculated using the following formula:

CuEq% = (((Ag × 46 × 0.68)/31.1035) + ((Au × 3200 × 0.89)/31.1035) + 0.0001 × (Co × 20 × 0.78 × 22.0462) + 0.0001 × (Cu × 4.8 × 0.85 × 22.0462))/(4.8 × 22.0462 × 0.85)

All metal inputs are expressed in ppm, and CuEq is reported in percent (%). Metal prices and metallurgical recoveries correspond to the assumptions used in the current PEA.

Drilling Dataset and Selection of Relevant Intercepts

The drilling results presented in this PEA correspond to the campaign conducted between July 2024 and September 2025. These data were used to update the Mineral Resource estimate for the Hat Project. The previous Mineral Resource estimate is dated July 17, 2024.

A total of 28 drill holes were added to the previously existing database. Of these, 22 drill holes returned continuous mineralization of sufficient length and grade to be considered relevant for reporting. The remaining six drill holes intersected only short, discontinuous mineralized intervals and were therefore not included among the significant intercepts.

The Hat mineralization is hosted within intrusive rocks, resulting in irregular mineralized domains that follow the geometry of the intrusion. The dominant strike direction of the system is approximately 150° azimuth.

Along this trend, the position and dip of the mineralized zones vary:

- In the northwestern sector, the mineralized body is sub-horizontal, dipping 20°–30° to the southeast.
- Toward the southeastern sector, the same mineralized trend steepens significantly, reaching dips of up to 60° to the southeast.

As a result, drill holes targeting the southeastern portion of the system are materially deeper than those drilled in the north and northwest as shown in Figure 10-3.

Reporting Format—Significant Intercepts with Nested Sub-Intervals

Drill results are presented as significant intercepts with nested “including” intervals. Each hole reports:

- A primary mineralized interval, representing the full continuous CuEq-bearing zone, and
- Nested higher-grade sub-intervals, which highlight internal grade variability and economically meaningful zones.

Some drill holes contain multiple levels of nested intervals. All intervals incorporate internal waste as captured in the composited dataset, and the reported grades represent true average grades over the stated lengths. This approach communicates both the overall mineralized thickness and the internal higher-grade structure without implying selective sampling.

Hat Project Drilling Campaign (July 2024–September 2025)

Table 10-5 presents a summary list of relevant intervals of the July 2024–September 2025 drilling campaign.

Table 10-5: Summary of Relevant Intervals

BHID	From (m)	To (m)	Length (m)	Copper Equivalent (wt %)	Copper (wt ppm)	Gold (wt ppm)	Silver (wt ppm)	Cobalt (wt ppm)
H072	233	565	332	0.69	3,802.1	0.27	0.51	81.05
including	338	504	166	1.14	6,261.2	0.45	0.87	115.58
including	429	550	121	1.20	6,798.2	0.46	1.03	112.43
including	429	504	76	1.77	9,874.6	0.69	1.46	158.95
H073	156	741	585	0.23	1,125.4	0.09	0.20	53.32
including	306	426	120	0.51	2,712.7	0.20	0.44	80.80
H074	188	489	301	0.26	1,245.8	0.10	0.21	71.73
including	218	338	120	0.28	1,252.6	0.12	0.22	81.29
including	368	489	120	0.28	1,501.4	0.10	0.23	59.37
including	368	413	45	0.47	2,365.9	0.19	0.35	98.73
H075	97	385	288	0.50	2,667.3	0.19	0.51	78.48
including	127	264	137	0.59	2,959.2	0.25	0.60	80.91
including	203	264	61	1.04	5,784.6	0.41	1.10	94.06
including	279	385	106	0.50	3,117.7	0.15	0.49	71.75
H076	155	378	223	0.36	1,687.9	0.15	0.20	84.80
including	155	199	45	0.37	1,727.5	0.12	0.43	190.38
including	244	378	134	0.41	2,048.6	0.18	0.16	61.65
H077	39	516	477	0.33	1,567.1	0.13	0.41	96.60
including	39	218	179	0.35	1,517.4	0.13	0.71	146.00
including	248	397	149	0.41	1,857.1	0.18	0.17	90.38
including	248	397	149	0.41	1,857.1	0.18	0.17	90.38
including	442	516	75	0.39	2,282.7	0.14	0.48	44.98
H078	125	633	508	0.69	3,701.3	0.27	0.58	92.39
including	200	409	209	0.98	6,079.7	0.31	1.05	114.84
including	200	350	149	1.22	7,611.2	0.39	1.32	127.88
including	454	499	45	0.53	3,061.0	0.18	0.33	104.68
including	559	633	75	0.70	3,589.2	0.31	0.30	55.00
H079	113	732	619	0.55	2,645.0	0.25	0.34	71.23
including	173	430	257	0.25	996.2	0.12	0.15	72.57
including	475	732	257	0.98	5,099.2	0.43	0.59	66.38
including	521	687	166	1.22	6,442.2	0.53	0.74	72.92
including	596	732	136	1.40	7,210.7	0.63	0.94	69.59
H080	109	609	500	0.23	1,118.2	0.09	0.13	66.42
including	109	351	242	0.33	1,581.6	0.13	0.18	93.81
including	230	351	121	0.43	2,183.6	0.17	0.19	94.76
including	230	276	45	0.60	3,251.3	0.23	0.23	90.12

BHID	From (m)	To (m)	Length (m)	Copper Equivalent (wt %)	Copper (wt ppm)	Gold (wt ppm)	Silver (wt ppm)	Cobalt (wt ppm)
H081	107	418	312	0.38	1,714.1	0.18	0.23	60.74
including	122	226	104	0.64	2,551.2	0.35	0.44	73.91
including	255	404	148	0.29	1,543.4	0.11	0.14	57.69
including	255	329	74	0.35	1,866.0	0.13	0.18	76.15
H082	108	483	375	0.37	2,325.0	0.11	0.41	63.19
including	258	453	195	0.57	3,934.4	0.14	0.67	79.83
including	348	453	105	0.69	5,333.9	0.11	0.99	82.88
H083	127	355	228	0.41	1,918.4	0.19	0.25	63.83
including	173	355	182	0.45	2,140.0	0.20	0.24	64.95
including	203	264	61	0.96	4,864.3	0.42	0.49	100.17
H085	36	468	433	0.22	1,090.7	0.09	0.19	50.57
including	304	468	164	0.35	1,998.9	0.12	0.28	60.40
H088	39	329	290	0.23	1,216.9	0.07	0.66	71.69
including	85	131	46	0.33	1,913.1	0.09	1.09	104.04
including	207	314	107	0.30	1,738.2	0.09	0.98	70.66
including	268	314	46	0.43	2,542.0	0.13	1.22	90.95
H092	52	741	689	0.50	2,371.1	0.23	0.36	73.87
including	292	741	449	0.65	3,329.5	0.28	0.46	83.36
including	367	651	284	0.87	4,426.4	0.37	0.60	104.12
including	382	442	60	0.66	3,815.0	0.21	0.39	165.04
including	487	651	165	1.12	5,622.7	0.51	0.82	84.22
including	546	651	105	1.43	6,935.5	0.68	1.08	85.12
H093	409	693	284	0.58	3,437.4	0.21	0.44	49.55
including	499	633	134	1.08	6,454.1	0.39	0.82	62.89
including	544	633	90	1.52	9,128.4	0.55	1.19	77.74
H094	233	597	364	0.22	1,216.5	0.08	0.13	53.50
including	294	552	258	0.23	1,241.6	0.08	0.12	49.10
including	445	552	106	0.34	1,889.2	0.13	0.17	47.56
including	491	552	61	0.36	1,996.4	0.14	0.18	40.49
H097	21	549	528	0.19	882.1	0.08	0.16	57.34
including	112	488	377	0.23	1,091.3	0.09	0.18	64.04
including	127	307	181	0.25	1,176.4	0.10	0.23	81.95
including	202	307	106	0.26	1,381.8	0.09	0.28	78.15
including	368	443	75	0.28	1,546.8	0.10	0.18	53.21
H098	128	544	416	0.28	1,568.2	0.09	0.23	77.90
including	202	440	238	0.36	2,247.7	0.10	0.33	87.71
including	247	440	193	0.41	2,556.5	0.11	0.35	88.90
including	380	440	59	0.51	3,087.2	0.16	0.28	95.74
including	410	440	30	0.70	4,385.8	0.21	0.37	104.64
H099	200	738	538	0.39	2,046.0	0.16	0.24	61.89
including	305	723	418	0.44	2,442.1	0.17	0.28	62.28
including	350	529	179	0.44	2,771.6	0.13	0.33	73.97

BHID	From (m)	To (m)	Length (m)	Copper Equivalent (wt %)	Copper (wt ppm)	Gold (wt ppm)	Silver (wt ppm)	Cobalt (wt ppm)
including	379	484	105	0.63	3,966.7	0.19	0.41	81.93
including	574	648	75	0.90	3,927.5	0.47	0.28	59.61
H100	76	750	674	0.25	1,026.0	0.12	0.12	67.22
including	226	630	405	0.31	1,411.7	0.13	0.14	79.80
including	406	630	225	0.36	1,866.5	0.14	0.18	74.88
including	465	555	90	0.41	2,162.4	0.16	0.21	83.98
including	525	630	105	0.25	1,288.6	0.10	0.16	56.22
H101	218	1,015	797	0.35	1,726.4	0.15	0.18	68.06
including	218	910	692	0.38	1,883.8	0.16	0.19	72.15
including	218	594	376	0.49	2,343.6	0.21	0.23	94.74
including	805	910	105	0.58	2,992.6	0.25	0.29	66.17

Note: BHID = borehole ID; wt = weighted average.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

This section describes the sampling procedures, sample preparation protocols, analytical methods, and security measures implemented for the Hat Deposit. All practices follow the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) *CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines* (CIM, 2019) and the requirements of NI 43-101.

Doubleview personnel completed sampling, preparation, and analytical work at the Hat Project site, assisted by accredited commercial laboratories, including AGAT Laboratories (Calgary, A.B.), ALS (North Vancouver, B.C.), MS Analytical (Langley, B.C.), and Bureau Veritas (formerly Acme Analytical Laboratories, Vancouver, B.C.). All laboratories used in the program are ISO/IEC 17025 accredited for the analytical methods employed.

The procedures outlined here integrate:

- Doubleview's 2025 Core Logging Guideline and QA/QC Procedure
- Laboratory protocols from AGAT Laboratories, MSALab, ALS Minerals, and Acme/Bureau Veritas
- The QA/QC framework described in Section 11.5.

These procedures ensure that all analytical data used in the Mineral Resource estimate are reliable, reproducible and verifiable.

11.1 Quality Assurance and Quality Control Overview

The Hat Project applies a proactive, multi-stage chain-of-control system that monitors data quality from the moment core is removed from the ground to the final analytical validation at head office. Each checkpoint is designed to detect errors early, prevent mislabeling, and maintain full transparency throughout the sampling and analytical workflow. This structured approach ensures that all data entering the resource model meet NI 43-101 and CIM Best Practice standards.

11.1.1 Summary of QA/QC Stages

QA/QC #1—Geologist (Hat Site: Pre-Cutting Control)

This first control point occurs immediately after logging and before core photography and cutting. The geologist verifies all sample tags, intervals, and depth references on the core boxes. Any inconsistencies—such as incorrect intervals, misplaced tags, or missing samples—are corrected before the core is cut. This step ensures that sampling intervals are geologically valid and properly documented.

QA/QC #2—Geologist (Hat Site: Post-Cutting and Sample Preparation)

After cutting, all samples are brought to the core shack for verification. The geologist checks that sample numbers match the log file, weighs each sample, assigns rice-bag numbers, and prepares the shipment

documentation. This stage ensures that physical samples and digital records are synchronized before leaving the site.

QA/QC #3—Geologist (Hat Site: Final File Verification)

Before sending the log file to head office, the geologist performs a full data validation. This includes verifying collar information, reviewing survey comments, confirming shipment dates, and ensuring all samples have been received by the laboratory. This step guarantees that the dataset is internally consistent and complete.

QA/QC #4—Laboratory and Head Office (Shipment and Receipt Control)

At the laboratory, staff confirm that all samples and rice bags listed in the shipment manifest have been received intact. Any damaged or missing samples are reported to head office for corrective action. Work orders confirmations are issued and recorded. This stage ensures the integrity of the chain of custody between site and laboratory.

QA/QC #5—Head Office (Analytical Verification and Database Integration)

Once analytical results are received, head office performs a detailed QA/QC review. Certified standards are checked against accepted limits, blanks are evaluated for contamination, and duplicates are assessed for precision. Thompson–Howarth plots are produced for key elements. Only after all checks passes are the results entered into the master database. This final stage ensures analytical accuracy and long-term data reliability.

11.2 Sampling

11.2.1 Core Handling and Logging

Upon retrieval from the drill, core boxes are transported to the Hat Project core facility where they are:

- Washed and aligned in their original orientation.
- Measured for core recovery and RQD (Section 10.3)
- Logged for lithology, alteration, mineralization, and structure following the standardized colour-coded system:
  - Lithology—blue
  - Alteration—red
  - Mineralization—yellow
- Measured for magnetic susceptibility using a KT-10 m
- Sample intervals defined based on geological criteria (mineralization, alteration, lithological boundaries, structural zones).

Figure 11-1 presents a schematic workflow from drilling to data readiness.

	DRILLING
1	**Core Transport to Site**
2	**Core Receiving and Washing**
3	**Core Reassembly and Checks**
	(Depth blocks, box numbers, CR/RQD)
4	**Geological Core Logging**
	• Lithology (blue)Alteration (red) • Mineralization (yellow) • Structures • Magnetic susceptibility (KT-10/KT-20) • Density measurements (KT-20)
5	**Definition of Sample Intervals**
	(Based on lithology, alteration, mineralization, structures)
6	**QA/QC #1—Pre-Sampling Check**
	(Tag placement, interval verification, corrections)
7	**Core Cutting**
	(Half-core for assay, half retained)
8	**Sample Bagging and Weighing**
	(Sample tags inserted; weights recorded; rice bags prepared)
9	**QA/QC #2—Sample Verification**
	(Weights, bag numbers, tag checks, sample list validation)
10	**Shipment Preparation and Security**
	(Rice bags sealed, labelled, stored securely)
11	**QA/QC #3—Sample Shipment**
	(Commercial carrier; chain-of-custody documentation)
12	**QA/QC #4—Laboratory Sample Reception**
	(Lab verifies counts, condition, integrity)
13	**Laboratory Preparation and Analysis**
	• Drying, crushing, splitting, pulverizing • Fire assay (Au) • ICP-AES / ICP-MS multi-element analysis • Ore-grade Cu analysis • Internal lab QA/QC (blanks, certified reference materials [CRM], duplicates)
14	**Laboratory QA/QC Review**
	(Internal CRMs, blanks, duplicates, drift checks)
15	**Certificate of Analysis Issued**
16	**QA/QC #5—Head Office Review**
	• CRM accuracy checks • Blank contamination checks • Duplicate precision (relative percent difference, Thompson–Howarth plots) • Batch acceptance / rejection
17	**Database Update and Validation**
18	**Data Ready for Resource Modelling**

Figure 11-1: Sampling, Sample Preparation, and QA/QC Workflow

11.2.2 Sample Interval Selection

Sampling is continuous from collar to end-of-hole. Intervals typically range from 0.5 to 3.0 m, shortened where necessary to honour:

- Lithological contacts
- Alteration boundaries
- Mineralized zones
- Faults, gouge, or breccia
- High-grade or visually heterogeneous intervals.

Intervals are marked directly on the core using the colour-coded system described in Section 11.2.1 and are verified during the execution of the QA/QC protocol at Stage 1 (QA/QC #1), as outlined in Section 11.1.1.

11.2.3 Core Cutting and Sample Preparation

Core is cut using a diamond saw, with half submitted for analysis and half retained for reference. The following procedures are applied:

- Duplicate samples are generated by quartering the core
- Density samples are marked with a “D” and measured using a KT-20 density module
- Sample tags (three-part) include:
  - Hole ID
  - From–To interval
  - QA/QC designation—standard, blank, duplicate (DUP).

A structured QA/QC program was implemented to ensure the reliability of all geochemical data. Control samples were inserted at defined intervals to monitor accuracy, precision, and potential contamination throughout the sampling and analytical workflow. The program includes routine primary samples, field duplicates, CRMs, blanks, and laboratory-generated duplicates, each serving a specific verification function. A summary of the sample types, their purpose, and insertion frequency is provided in Table 11-1.

Table 11-1: Summary of QA/QC Sample Types, Purpose, and Insertion Frequency

Sample Type	Purpose	Frequency	Notes
Routine Half-Core	Primary geochemical analysis	Continuous	Standard sampling
Field Duplicate (Quarter-Core)	Precision assessment	1 in 60	Marked DUP
Certified Reference Material (CRM)	Accuracy assessment	1 in 60	Multiple grades used
Blank	Contamination detection	1 in 60	Glass or barren rock
Coarse Reject Duplicate	Secondary precision check	As required	Sent to secondary lab
Pulp Duplicate	Laboratory precision	Internal to lab	Reported in certificates

Doubleview inserts QA/QC samples at a nominal rate of 1 in 20 samples, consisting of:

- CRMs
- Blank samples (dolomite barren material)
- Field duplicates (quarter-core splits).

Then:

- QA/QC samples are inserted at a frequency of one of three control-sample types for every 20 routine samples, corresponding to an overall insertion rate of about 5%. As a result, each type of control sample is appearing once every 60 routine samples.

The total QA/QC insertion rate (5%) meets minimum CIM expectations. The per-type insertion rate (1.7%) is low compared to industry norms, which are as follows:

- CRM: typically, 1 in 20 (5%)
- Blank: typically, 1 in 20–30 (3%–5%)
- Duplicate: typically, 1 in 20–30 (3%–5%).

With the purpose of improving QA/QC program, the QP recommends increasing CRM and blank insertion frequency and ensuring duplicates are inserted at a rate sufficient to evaluate sampling heterogeneity and precision.

11.2.4 Sample Weighing and Bagging

Each sample is weighed (±0.01 kg) and recorded in the logging database. Samples are placed in plastic sample bags, sealed with zip ties, and grouped into rice bags weighing 15–20 kg.

Rice bags are labelled with:

- Bag number
- Laboratory name
- “From: Doubleview Gold Corp.”

No hole number or location information is written on the exterior, consistent with security protocols.

11.3 Sample Security

Sample security is maintained from drill site to laboratory through a multi-stage chain-of-custody system.

11.3.1 On-Site Security

- Samples remain in the custody of Doubleview personnel at all times.
- Weighted samples placed in rice bags are stored in a locked, access-controlled facility.
- Sample books and other physical documentation are archived and shipped to head office at season end.

11.3.2 Transportation

- Samples are transported by commercial carriers directly to the laboratory.
- Chain-of-custody forms accompany each shipment.
- Head office verifies the receipt of all bags generated through the Stage 5 (QA/QC #5) procedure, as described in Section 11.1.1.

11.3.3 Laboratory Security

All laboratories used are accredited facilities with internal security systems, including:

- Controlled access
- Sample tracking barcodes
- Secure storage of pulps and rejects.

Pulps and rejects are returned to Doubleview or stored at the laboratory per contractual agreement.

11.4 Analytical Techniques

All laboratories the Hat Project has used are ISO/IEC 17025 accredited.

11.4.1 Laboratory Sample Preparation

A standardized sample preparation workflow follows a consistent set of technical steps presented in Table 11-2, regardless of the specific analytical code selected. Workflows typically include drying, crushing, splitting, pulverizing, and producing a representative pulp suitable for the requested analytical method. Although individual method codes may differ in digestion type, analytical finish, or detection limits, the underlying preparation procedures remain broadly similar across laboratories and analytical packages.

Table 11-2: Laboratory Sample Preparation Steps

Step	MSA Code	Acme Code	Description
Drying	PRP-910	R200-250	Dry core, crush to 70% passing 2 mm
Crushing	PRP-910	R200-250	Split 250 g subsample
Pulverizing	PRP-910	R200-250	Pulverize to 85% passing 75 µm (200 mesh)

11.4.2 Gold Analysis

Fire assay with atomic absorption spectrophotometer (AAS) finish (FAS-121 / Au-AA23)

- 30 g or 50 g fusion
- Detection limit: 0.005 ppm
- Over-limit (>10 ppm) re-assayed by gravimetric finish.

11.4.3 Multi-Element Analysis

Four-acid digestion ICP- atomic emission spectroscopy (AES) (ICP-230 / ME-ICP61):

- 0.2 g sample
- Near-total digestion
- Suitable for copper, cobalt, silver, molybdenum, zinc, lead, iron, aluminum, magnesium, calcium, phosphorus and other variables of interest.

Ore-grade copper (ICF-6Cu / Cu-OG62):

- Four-acid digestion
- ICP-AES finish
- Used for samples >10,000 ppm Cu.

11.4.4 Precious Metals (Platinum Group Elements)

Palladium, platinum, and rhodium are analyzed by ICP-OES or INAA depending on the laboratory.

11.4.5 Density and Magnetic Susceptibility

- Density measured using KT-20 density module (air/water displacement)
- Magnetic susceptibility measured using KT-10 meter (average and maximum per run).

11.4.6 QA/QC Review

QA/QC results are reviewed at three levels described in Section 11.1.1:

1. Field—QA/QC #1–#3
2. Laboratory—QA/QC #4
3. Head Office—QA/QC #5

Analytical accuracy is assessed using CRMs available in Section 11.5.1 and Appendix 11.1, while precision and heterogeneity is evaluated using duplicate pairs and Thompson–Howarth plots presented in Section 11.5.3.

11.4.7 Check Assays

In accordance with Doubleview, pulps are periodically sent to a secondary laboratory for independent verification. Any material discrepancies trigger reanalysis and investigation.

11.5 QA/QC Program

The QA/QC program complements the samples submitted to the analytical laboratory. Sample selection for analytical testing is carried out during the geological logging process. The average analytical sample length is 2.07 m, with a maximum of 15.5 m and a minimum of 0.2 m.

QA for chemical analysis in the Hat Project is implemented through the insertion of control samples. These control samples consist of CRMs, pulp blanks, and field duplicate samples. The insertion frequency for each type of control sample is presented in Table 11-3.

Table 11-3: Control Sample and Insertion Rates Used in the Core Drilling Quality Assurance Program

Sample Type	Sample Subtype	Insertion Rate (%)
Duplicate	Field	1.7
	Coarse	0
	Pulp	0
CRM	14 various CRMs	1.7
Blank	Coarse	0
	Pulp	1.7
Umpire Check	Assay validation	5 (2024)
Total		**10.1**

The three types of control samples are inserted at a steady rate of approximately 5%. In practice, one control sample is added for every 20 routine samples, with the type of control sample rotated to ensure balanced coverage across the QA/QC program.

The QA/QC program meets the minimum requirements necessary to support the credibility and reliability of the analytical database. An annual program of submitting selected samples for reanalysis at an umpire laboratory further complements and completes the QA/QC framework.

11.5.1 Certified Reference Material

CRMs are used in QA/QC programs to monitor and control analytical accuracy. Since the first drilling campaign in 2013, various CRMs have been used (Figure 11-4. The Project applies clearly defined acceptance criteria based on the certified best value (CBV) and associated standard deviations (SD). Results falling within CBV ±2 SD are considered acceptable, those between ±2 SD and ±3 SD are treated as warnings, and results outside ±3 SD are classified as failures.

In practice, CRM results within ±3 SD of the certified value are generally regarded as acceptable, as this interval corresponds to a nominal 99.8% confidence level. This represents a relatively broad tolerance. Under normal laboratory performance, and assuming proper sample handling, fewer than 0.2% of CRM results would be expected to fall outside this range.

Table 11-4: Certified Reference Materials Used in the Hat Exploration Programs

CRM	Variable	Expected Average	±2 SD	±3 SD	Source	CRM Used
Cu 187	Cu %	0.38	0.0264	0.0396	WCM Minerals	3
	Au ppm	0.514	0.047	0.0705		
	Ag ppm	12.02	1.57	2.355		
Cu 193	Cu %	0.54	0.018	0.027	WCM Minerals	4
	Au ppm	0.48	0.021	0.0315		
	Ag ppm	3.0	0.32	0.48		
CM 24	Cu %	0.365	0.02	0.03	CDN Resource	5
	Au ppm	0.52	0.056	0.084		
	Ag ppm	4.1	0.4	0.6		
CM 30	Cu %	0.73	0.034	0.051	CDN Resource	5
	Au ppm	1.3	0.12	0.18		
	Ag ppm	15.9	1.3	1.95		
Cu 165	Cu %	0.31	0.008	0.012	WCM Minerals	24
	Au ppm	1.42	0.046	0.069		
	Ag ppm	31	1.28	1.93		
Cu 189	Cu %	0.9	0.036	0.054	WCM Minerals	20
	Au ppm	0.84	0.033	0.05		
	Ag ppm	13	1.1	1.65		
CM 32	Cu %	0.23	0.01	0.015	CDN Resource	60
	Ag ppm	1.4	0.3	0.45		
CM 33	Cu %	0.35	0.015	0.023	CDN Resource	59
	Ag ppm	2.3	0.3	0.45		
CM 40	Cu %	0.56	0.032	0.048	CDN Resource	84
	Au ppm	1.31	0.12	0.18		
	Ag ppm	18	2	3		
CM 47	Cu %	0.72	0.028	0.042	CDN Resource	47
	Au ppm	1.13	0.11	0.165		
	Ag ppm	69	6	9		
ME 1311	Cu %	0.468	0.022	0.033	CDN Resource	53
	Au ppm	0.839	0.066	0.099		
	Ag ppm	44.9	2.2	3.3		
ME 1414	Cu %	0.219	0.01	0.015	CDN Resource	51
	Au ppm	0.284	0.026	0.039		

CRM	Variable	Expected Average	±2 SD	±3 SD	Source	CRM Used
	Ag ppm	18.2	1.2	1.8		
ME 1812	Cu %	0.989	0.042	0.063	CDN Resource	56
	Au ppm	7.86	0.66	0.99		
	Ag ppm	97	5	7.5		

The Hat Project uses two CRM providers: CDN Resource Laboratories Ltd. and WCM Minerals.

Table 11-5: Certified Reference Material Providers

Company	Address
CDN Resource Laboratories Ltd.	#2, 20148–102nd Ave, Langley, B.C., Canada, V1M 4B4
WCM Sales Ltd. (WCM Minerals)	7729 Patterson Avenue, Burnaby, BC, Canada V5J 3P4

For the Hat Project, CRM analyses were completed for approximately 95% of all inserted standards. Five standards were not analyzed by the QP for QA/QC analytical-validation purposes: CM 24, CM 30, CM 187 and CM 193, representing 23 of the 471 total insertions up to the data cut-off date. Across the historical datasets reviewed to date, 79 CRM results for gold, silver, or copper fall outside the ±3 SD acceptance limits (Table 11-6). Of these, 11 results are lower than the certified value, while 68 are higher.

Table 11-6: Distribution of CRM Failures by CRM Type

CRM	CRM Failures (Outside ±3 SD)
CDN-CM-32	4
CDN-CM-33	4
CDN-CM-40	20
CDN-CM-47	12
CDN-ME-1311	3
CDN-ME-1414	5
CDN-ME-1812	9
Cu 165	21
Cu 189	1
Total	**79**

Where reanalysis was required, and the review confirmed that the original assay should be replaced, the corrected value was incorporated into the database. Outdated assay values were removed to ensure that only the most current and reliable data are retained. In cases where no corrective action was necessary, the results were documented within the internal QA/QC records.

The database Doubleview provided for Mineral Resource estimation represents the finalized version and incorporates all required corrections across its various components.

An example of laboratory accuracy performance is shown in Figure 11-2 to Figure 11-4, which illustrate the behaviour of the CDN-CM-40 CRM. The complete sequence of analytical figures is presented in Appendix 11.1.

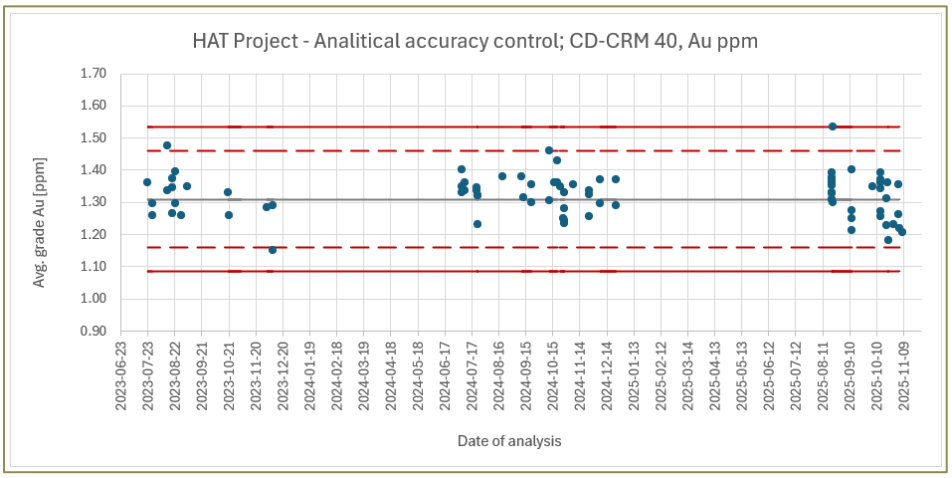


Figure 11-2: Analytical Performance of CRM CDN-CM-40 for Gold (Au)

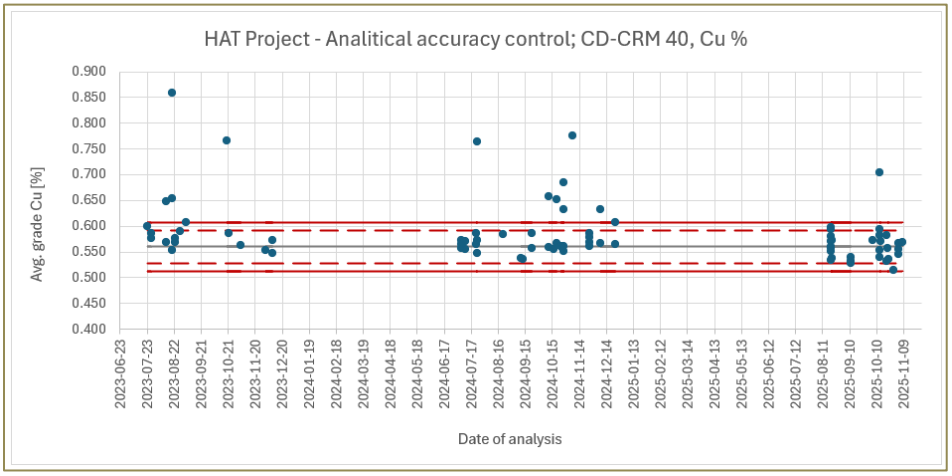


Figure 11-3: Analytical Performance of CRM CDN-CM-40 for Copper (Cu)

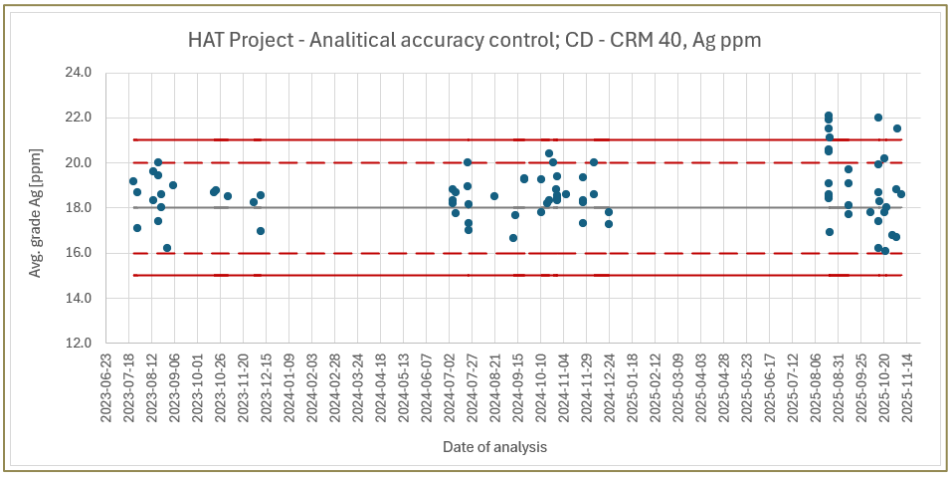


Figure 11-4: Analytical Performance of CRM CDN-CM-40 for Silver (Ag)

The results are plotted against the actual sample dispatch dates, highlighting the relationship between submission timing, mechanical preparation, and analytical performance. Certified Reference Material CD-CRM 40 was incorporated into the QA/QC program during three distinct periods: July to December 2023, July to December 2024, and August to November 2025, reflecting its periodic use in monitoring analytical accuracy throughout the drilling campaigns.

The frequency and timing of sample submissions to the laboratory have a direct influence on how reliably QA/QC performance can be evaluated. When samples and CRMs are dispatched at regular intervals, the resulting data reflect the laboratory's normal operating conditions, and allow trends such as drift, calibration changes, or shifts in analytical behaviour to be monitored over time.

In contrast, when samples are submitted only occasionally, or in large, infrequent batches, QA/QC results represent isolated snapshots rather than continuous performance. This limits the ability to detect long-term analytical trends and may cause CRM failures to cluster around short-term laboratory conditions rather than indicating a persistent analytical bias. Irregular submissions also reduce the statistical power of the QA/QC program, making subtle issues more difficult to identify.

For the Hat Project, logistical constraints meant that sample dispatch could not always occur on a routine schedule. Despite this limitation, each batch contained sufficient control materials to evaluate analytical performance at the time of processing. Where CRM results indicated potential issues, reanalysis was undertaken and corrections were applied as required. The final assay database incorporates all validated and corrected results and is considered suitable for use in Mineral Resource estimation.

The full set of analytical figures for the CRMs analyzed in this study is provided in Appendix 11.1.

11.5.2 Blanks

Blank samples are used to monitor potential contamination within the analytical circuit, where large numbers of samples are processed sequentially and even trace carryover can compromise data quality. By confirming

that equipment and preparation stages remain free of the elements being analyzed, blanks provide assurance that reported concentrations reflect true sample values rather than laboratory-introduced contamination.

Coarse blanks were not available for monitoring potential contamination during mechanical sample preparation. As a result, contamination control relied on pulp blanks, which are expected to return values less than or equal to twice the analytical detection limit.

A threshold must be established to enable effective detection of contamination. A common approach in contamination monitoring is to set this threshold at twice the laboratory detection limit.

Because the Hat Project used multiple laboratories, additional consideration was required. Each laboratory employs different analytical protocols and maintains its own set of detection limits. To present the blank results in a consistent and comparable format, upper outliers were capped where necessary to reflect the practical uncertainty associated with each laboratory's detection capabilities.

Table 11-7 summarizes the twice-detection-limit thresholds applied to evaluate pulp blank performance across all laboratories.

Table 11-7: Threshold Values Used to Monitor Potential Analytical Contamination

Variable	2 x Detection Limit (ppm)
Au	0.01
Ag	0.1
Cu	2
Co	2
Sc	2

Due to clustering and overlap in the dispatch dates, samples in Figure 11-5 to Figure 11-9 are shown in chronological order from earliest to latest work. Analytical contamination is inferred when a blank sample plots above the detection-limit line.

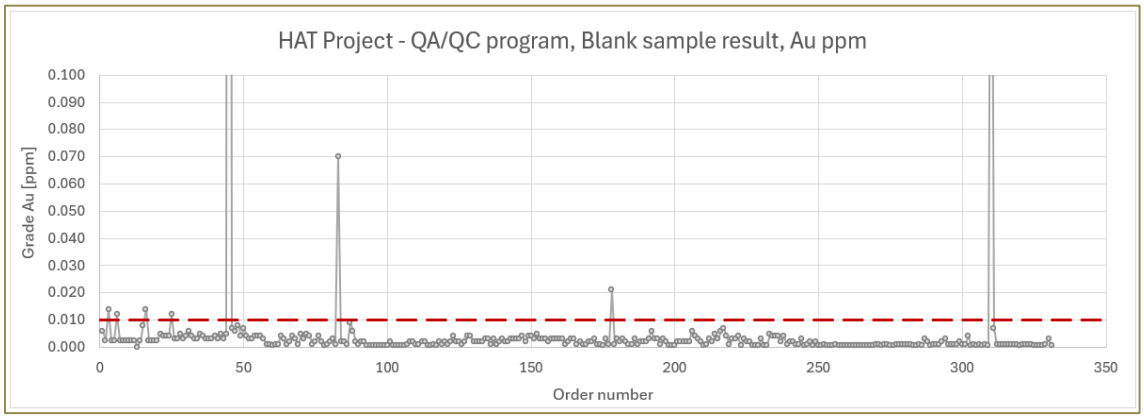


Figure 11-5: Summary of Pulp Blank Performance for Gold (Au)

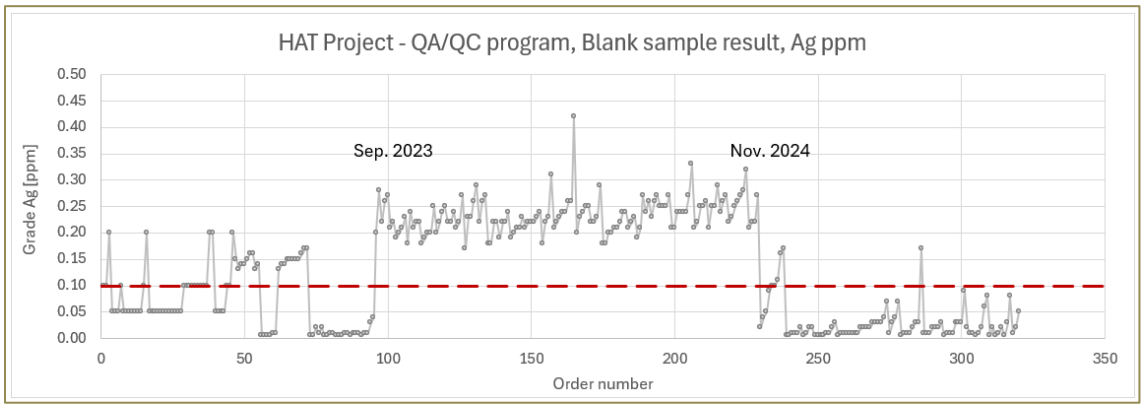


Figure 11-6: Summary of Pulp Blank Performance for Silver (Ag)

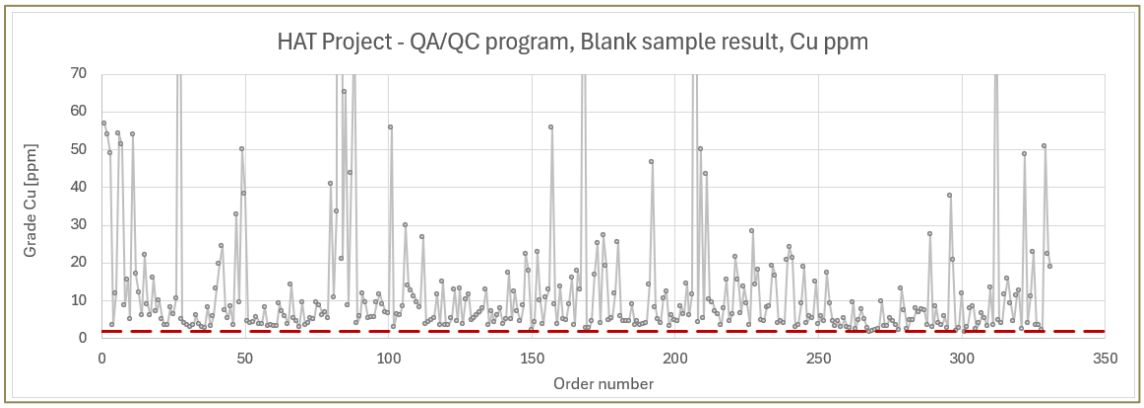


Figure 11-7: Summary of Pulp Blank Performance for Copper (Cu)

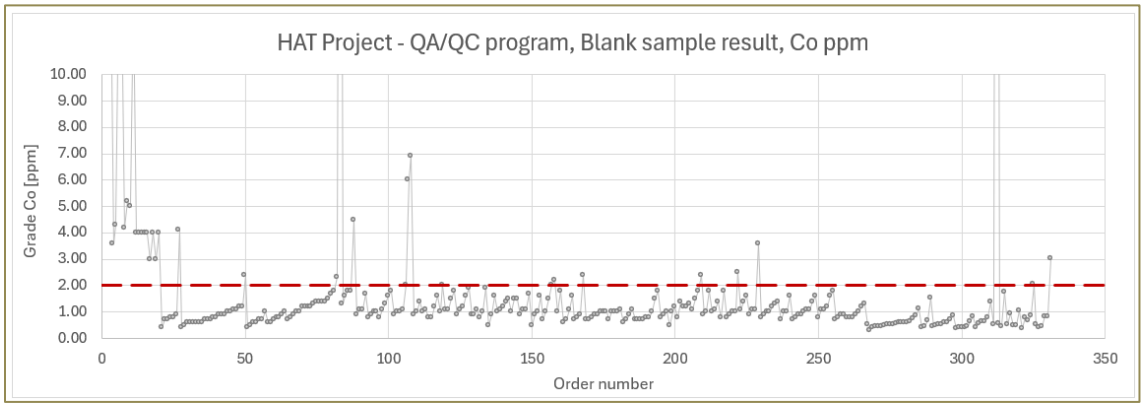


Figure 11-8: Summary of Pulp Blank Performance for Cobalt (Co)

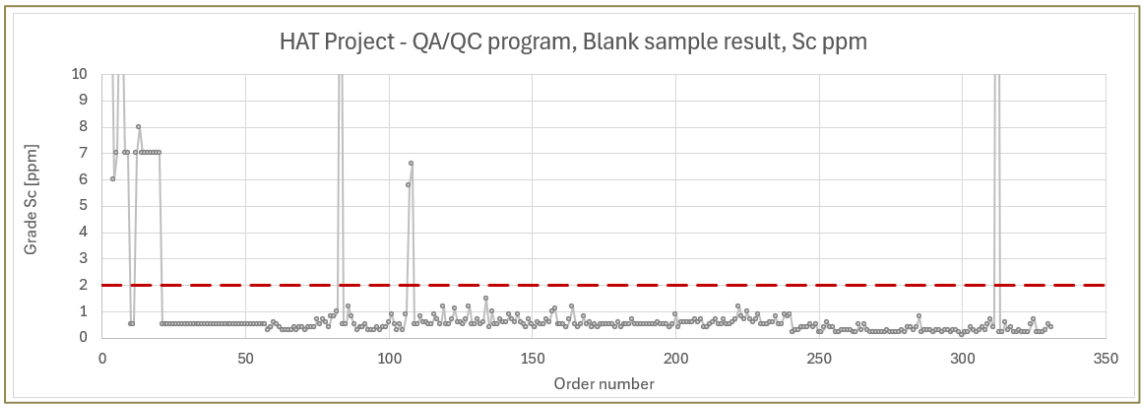


Figure 11-9: Summary of Pulp Blank Performance for Scandium

The pulp blank used to assess analytical contamination is a dolomite material for which no certification is available. Review of its performance indicates that this material is not suitable for use as a blank. While occasional—yet recurrent—contamination is observed for scandium, gold, and cobalt, the results for silver and copper are highly inconsistent, to the extent that they undermine the intended purpose of the material labelled as dolomite.

Other materials used as pulp blanks are not considered here, as they represent only 23 cases out of 331 and were employed during the very early stages of Project development. For the purposes of this evaluation, the only relevant pulp blank material is dolomite.

As previously noted, the role of a pulp blank is to monitor analytical or instrumental contamination, events that should occur only at very low probability. The standardized analytical protocols used by the laboratories supporting the Hat Project ensure that any elevated results are far more likely to reflect impurities in the blank material itself rather than true instrumental contamination. Consequently, the use of this incorrect material renders this component of the QA/QC program unsuitable for drawing meaningful conclusions regarding contamination.

11.5.3 Duplicate Samples

Duplicate samples can be generated at any stage in the sampling process where the sample mass is reduced (i.e., when the sample is split and a reject is created). During the early phases of establishing sampling, sample preparation, and analytical protocols, duplicates should be collected frequently at all relevant stages to generate sufficient data for estimating the variance introduced by each step.

In the Hat Project, the primary sample is obtained by splitting the drill core in half. One half is submitted as the primary analytical sample. The duplicate is produced from the remaining half-core by splitting it again to obtain a quarter-core, which is then submitted as the duplicate sample. It is important to note that using different sample masses for primary and duplicate samples is not considered best practice, as it introduces unnecessary variability into the comparison.

When two sample populations are derived from the same material but differ in sample mass, the population composed of smaller samples typically exhibits a distribution with a longer "tail"—that is, it becomes more skewed and displays greater variability. If sampling is performed correctly and the dataset is sufficiently large, both populations should have similar means; however, the spread of grades will differ. This effect is commonly referred to as the change of support: increasing sample support (mass or volume) reduces grade variability, while decreasing support increases it.

Empirical evidence from porphyry-style deposits indicates that differences in sample mass between primary and duplicate samples can degrade analytical precision by as much as 10%—or inflate the inherent variance of the mineralization—purely due to the change-of-support effect rather than laboratory performance.

Establishing an appropriate pass/fail threshold for duplicate performance is not always straightforward, as the reliability of duplicate pairs varies between deposits. A tolerance considered stringent in one geological setting may be easily achieved in another. Nevertheless, several industries' "rules of thumb" are commonly referenced when evaluating duplicate precision:

- Coarse reject duplicates are generally expected to fall within ±20% relative difference for at least 90% of the population.
- Pulp duplicates are typically expected to fall within ±10% relative difference for at least 90% of the population.

For coarse and pulp duplicates—materials that are inherently more homogeneous—standardized expectations for analytical precision are well established. In contrast, field duplicates reflect natural geological variability, and therefore the same numerical criteria cannot be applied directly.

Interpretation of field duplicate results must consider the geological context, sample type, and expected heterogeneity of the material being tested. The Hat Project relies on field duplicates, which are intended to capture the natural variability of the mineralized material rather than laboratory precision alone.

Scatter Plot

A scatter plot of duplicate assay results is presented in Figure 11-10 to Figure 11-14, and proportion of duplicate pairs in Table 11-8, and Table 11-9 where the number of data points lying beyond the tolerance limits is quantified. This is a standard diagnostic tool for evaluating duplicate performance. In this plot, the duplicate assay values are plotted against the corresponding primary assay values, with the X = Y line and the upper and lower tolerance limits superimposed. This presentation allows differences between the two datasets to be assessed directly, without the need for data transformations.

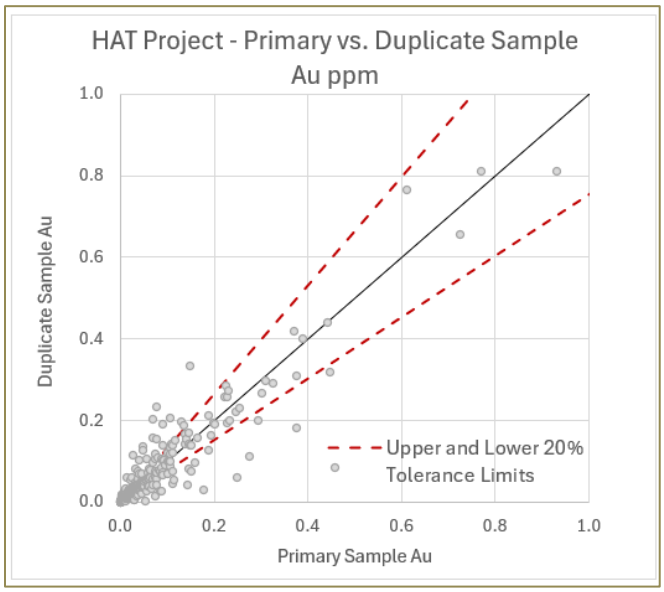


Figure 11-10: Scatter Plot of Primary and Field Duplicate Samples for Gold (Au)

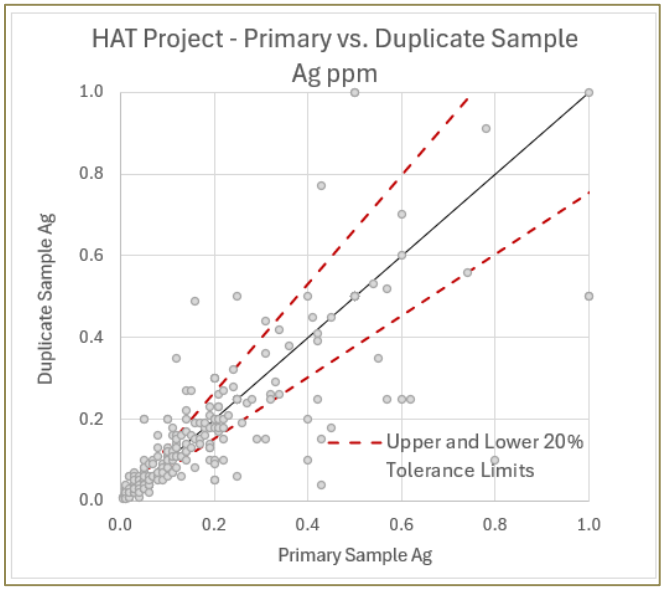


Figure 11-11: Scatter Plot of Primary and Field Duplicate Samples for Silver (Ag)

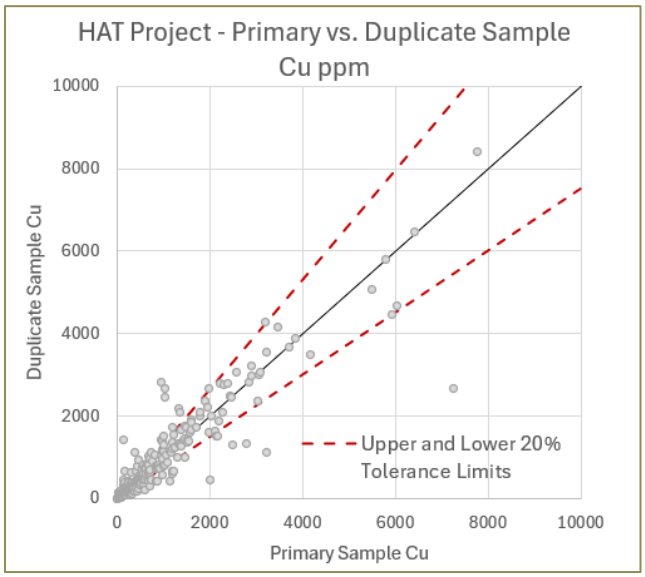


Figure 11-12: Scatter Plot of Primary and Field Duplicate Samples for Copper (Cu)

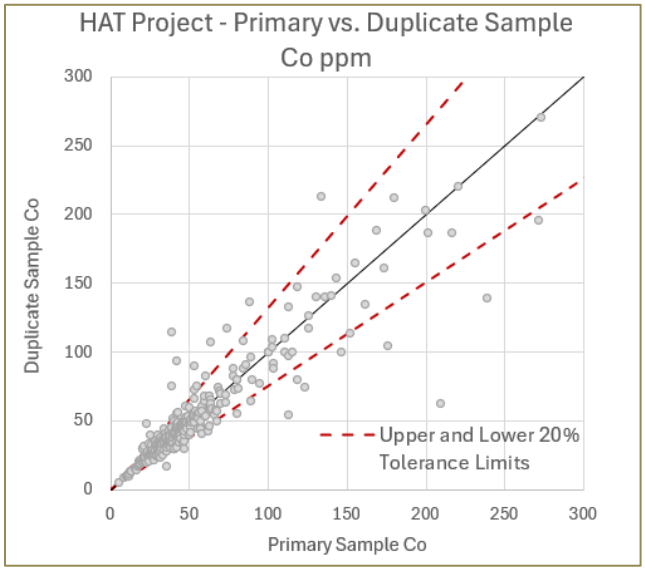


Figure 11-13: Scatter Plot of Primary and Field Duplicate Samples for Cobalt (Co)

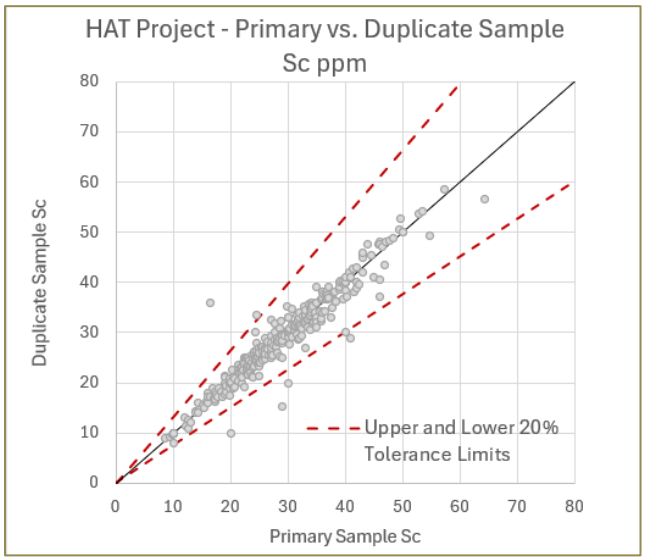


Figure 11-14: Scatter Plot of Primary and Field Duplicate Samples for Scandium (Sc)

Table 11-8: Proportion of Duplicate Pairs Within 20% Tolerance limits

Variable	Proportion of Duplicate Pairs Within ±20% Tolerance
Ag	62.2
Au	55.6
Cu	67.7
Co	86.1
Sc	97.9

Among the analyzed commodities, gold, silver, and copper exhibit the highest natural variability, with a substantial proportion of duplicate pairs falling outside the ±20% tolerance limits. In contrast, cobalt and scandium show very few pairs exceeding the 20% threshold, indicating a more stable and homogeneous distribution for these elements.

When the tolerance limits are relaxed to ±30%, the proportion of duplicate pairs falling within the acceptable range increases for all commodities. However, gold, silver, and copper still fall short of the 90% benchmark, as summarized in Table 11-9.

Table 11-9: Proportion of Duplicate Pairs Within 30% Tolerance limits

Variable	Proportion of Duplicate Pairs Within ±30% Tolerance
Ag	74.9
Au	72.2
Cu	77.6
Co	94.9
Sc	99.1

These observations are consistent with the Relative Difference Distributions for Field Duplicates presented in continuity of this section, in which the absolute relative difference between primary and duplicate samples is evaluated. The results confirm that variability is commodity-dependent, and that certain elements inherently display greater dispersion at the scale represented by field duplicates.

Relative Difference

The relative difference is defined as the absolute difference between the pair divided by the pair mean.

The Thompson–Howarth approach is effective in assessing field heterogeneity by plotting the sorted absolute value of relative percent differences against percentile rank in Figure 11-15 to Figure 11-18; (Thompson and Howarth, 1976). To make this plot, the absolute value of the difference is divided by the mean of the pair, as in the formula:

$$\text{Relative Difference} = \text{Absolute Value}\,[2 \times (A - B) / (A + B)]$$

This analysis is particularly important because it quantifies how much the grade can vary over a very short distance, essentially the width of the drill core under the assumption that sample preparation and analytical protocols remain consistent over time.

Given the inherent geological heterogeneity and the absence of prior knowledge regarding deposit-scale variability, a more permissive threshold such as ±30% relative difference may be considered reasonable as an initial benchmark. Even with this relaxed criterion, some variables still exceed the hypothetical threshold.

More than the expected 10% of duplicate pairs fall outside the tolerance limits for gold, silver, and copper. This indicates that the variability observed during early data exploration is consistently reflected in the QA/QC program and represents a characteristic feature of the dataset.

The analysis is done twice for variables where the whole population and cut-off population are presented. The cut-off population corresponds to probabilistic analysis of mineralized volume based on the cut-off to separate background from probable economic mineralization, discussed in Sections 14.4 and 10.4.1.

Thompson–Howarth Relative Difference Distributions for Field Duplicates, illustrating the magnitude and percentile distribution of absolute relative differences for each element Figure 11-15 to Figure 11-18. The plots highlight short-range geological heterogeneity and identify duplicate pairs exceeding the ±30% (±10% Sc) relative-difference threshold.

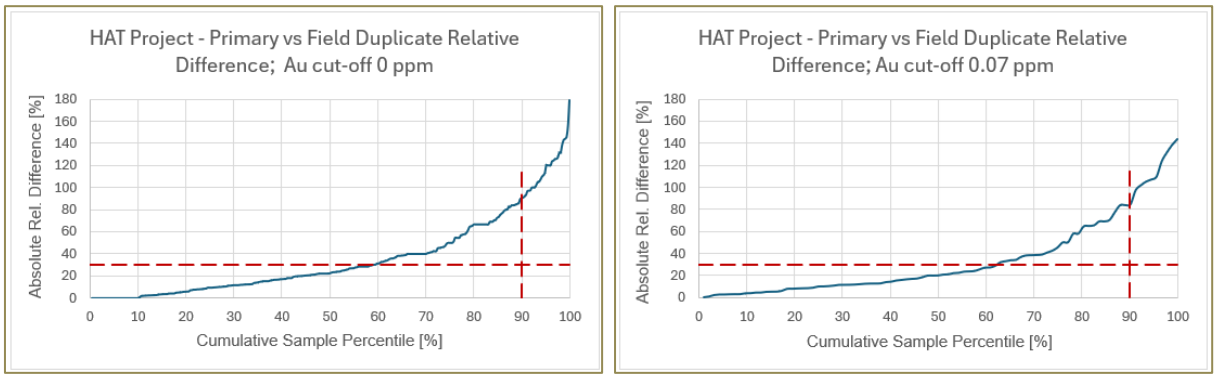


Figure 11-15: Percentile Distribution of Absolute Relative Differences for Gold (Au)

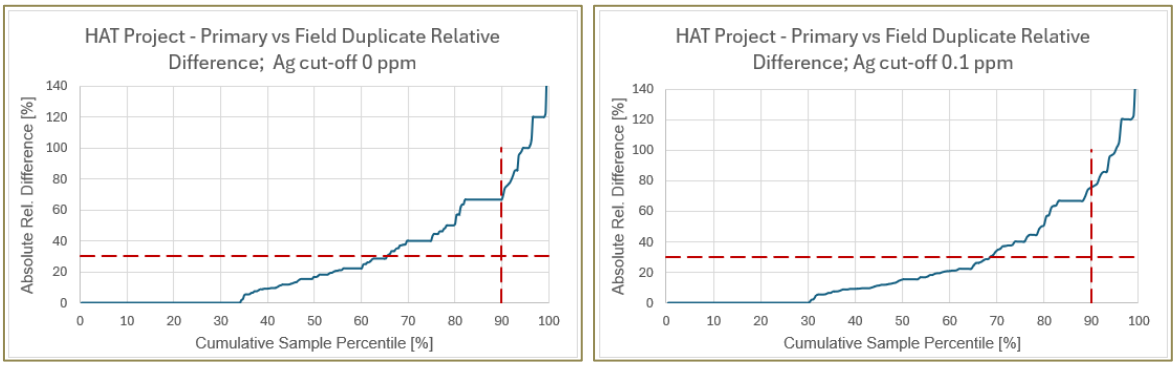


Figure 11-16: Percentile Distribution of Absolute Relative Differences for Silver (Ag)

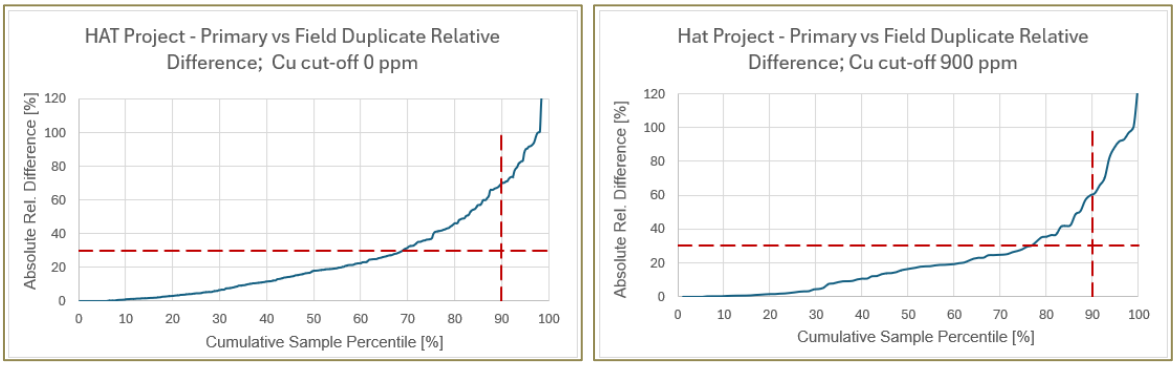


Figure 11-17: Percentile Distribution of Absolute Relative Differences for Copper (Cu)

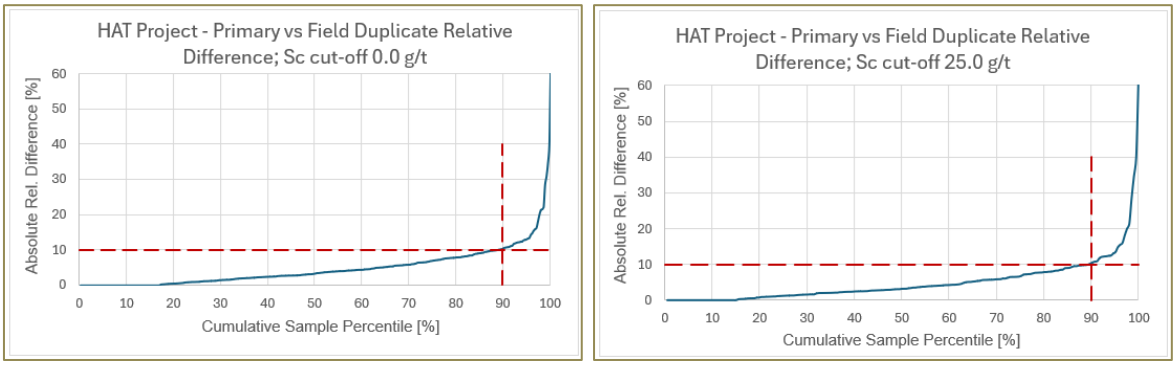


Figure 11-18: Percentile Distribution of Absolute Relative Differences for Scandium (Sc)

Comments on Absolute Relative Difference

Scandium duplicates are straightforward to interpret. The absolute relative difference between the primary and field duplicate behaves similarly to what is typically observed for pulp duplicates, with 90% of the population falling within ±10% relative difference.

Cobalt also shows strong performance. The absolute relative difference between primary and field duplicates meets the ±30% relative difference threshold for 90% of the population, which is the hypothetical target adopted for this Project.

The remaining variables exhibit substantially higher variability, and the hypothetical ±30% absolute relative difference threshold for field duplicates is not achieved. Even after accounting for the difference in sample mass between primary and duplicate samples (half-core versus quarter-core), it is evident that silver, gold, and copper display unexpectedly high local variability:

- Silver falls more than 25% short of achieving 90% of pairs within ±30% relative difference.
- Gold reaches the ±30% threshold at only 60% of the cumulative population, rather than the expected 90%.
- Copper reaches 70% of the cumulative population at ±30% relative difference. When applying the Project's analytical cut-off, copper performance improves slightly but still remains approximately 10% below the 90% target.

Although duplicate analysis in the Hat Project QA/QC program is not a decision-making criterion, it provides valuable insight into the inherent mineralization variability of each commodity. This information is essential for understanding grade continuity, sampling performance, and the geological behavior of the deposit.

To fully benefit from the duplicate database, the number and type of duplicate samples should be expanded to include coarse and pulp (fine) duplicates. When used together with field duplicates, these additional duplicate types would provide variance control at each subsampling stage and allow quantification of the overall sampling error.

11.6 Internal Laboratory QA/QC Programs

Doubleview implements a QA/QC program that meets the minimum requirements recommended by the CIM (2019) best practice guidelines and the disclosure expectations of NI 43-101. The program includes the routine insertion of CRMs, blanks, and field duplicates into the primary sample stream, providing a baseline level of analytical control that is consistent with industry norms for early- to mid-stage exploration projects.

In addition to Doubleview's internal QA/QC measures, the Hat Project benefits from the use of professional, ISO-accredited commercial laboratories listed in Table 11-10. These laboratories operate under rigorous internal quality systems that include method blanks, CRMs, preparation and analytical duplicates, calibration checks, and continuous monitoring of instrument performance. Although Doubleview's field-based QA/QC program is not yet at an "excellence" level in terms of insertion frequency or diversity of control materials, the laboratory QA/QC framework provides an important complementary layer of quality assurance.

Given the combined effect of company-level and laboratory-level controls, the QP considers it essential to include a dedicated section describing the internal laboratory QA/QC programs. This ensures that readers understand that the analytical dataset supporting the Mineral Resource estimation is underpinned by robust laboratory quality systems, and that the overall data quality is sufficient for the classification and reporting of Mineral Resources in accordance with NI 43-101 and CIM standards.

11.6.1 Laboratory Accreditation and Quality Systems

Each laboratory Doubleview uses maintains ISO-based quality systems, typically including the standards listed in Table 11-10.

Table 11-10: Accredited Analytical Laboratories and Applicable Quality Standards

Laboratory	Accreditation	Relevant Standards
AGAT Laboratories	ISO 9001, ISO/IEC 17025	Certified quality management system; competence of testing and calibration laboratories
ALS Minerals	ISO/IEC 17025	Analytical testing and calibration
Bureau Veritas (formerly Acme)	ISO 9001, ISO/IEC 17025	Quality management and analytical competence; certified quality management system
MS Analytical	ISO/IEC 17025	Competence of testing and calibration laboratories

These accreditations require documented procedures for sample handling, chain of custody, analytical calibration, instrument maintenance, and data verification.

11.6.2 Laboratory Sample Preparation

A standardized sample preparation workflow, regardless of the specific analytical code selected, follows a consistent set of technical steps listed in Table 11-11.

Table 11-11: Standardized Sample Preparation Workflow

Step	MSA Code	Acme Code	Description
Drying	PRP-910	R200-250	Dry core, crush to 70% passing 2 mm
Crushing	PRP-910	R200-250	Split 250 g subsample
Pulverizing	PRP-910	R200-250	Pulverize to 85% passing 75 µm (200 mesh)

11.6.3 Analytical Methods

Table 11-12 lists the analytical methods routinely used.

Table 11-12: Analytical Methods

Element(s)	Method	Description
Au	FAS-121	50 g fire assay, AAS finish
Cu (Ore Grade)	ICF-6Cu	4-acid digestion, ICP-AES
Multi-Element Suite	ICP-230	4-acid digestion, ICP-AES
Precious Metals (Pd, Pt)	INAA or ICP-OES	Depending on turnaround and detection limits

11.6.4 Internal QA/QC Components

Internal laboratory QA/QC programs typically include analytical blanks, CRMs, and laboratory duplicates.

Analytical Blanks

Laboratories insert method blanks at regular intervals within each analytical batch. These blanks detect:

- Contamination during digestion
- Carryover between samples
- Contamination introduced by reagents or equipment.

A blank returning values above the method detection limit triggers an internal investigation and may require reanalysis of affected samples.

Certified Reference Materials

Laboratories insert their own in-house CRMs into each batch to monitor:

- Instrument calibration
- Analytical drift
- Digested efficiency.

These CRMs are independent of the client's standards and are used to verify that the analytical system is performing within acceptable tolerance ranges.

If a CRM result falls outside the certified ±2 SD range, the batch is flagged and may be re-run.

Laboratory Duplicates

Laboratories routinely perform duplicate analyses to assess analytical precision. These include:

- Pulp Duplicates
  - Same pulp split analyzed twice
  - Measures instrument precision.
- Crush Duplicates
  - Two subsamples taken after crushing
  - Measures sample homogeneity and preparation precision.
- Coarse Reject Duplicates
  - Two subsamples taken at the reject stage
  - Measures of variability introduced during crushing.

Precision is evaluated using relative percent difference or Thompson–Howarth plots.

11.6.5 Instrument Calibration and Drift Correction

Laboratories perform:

- Initial calibration using multi-element calibration standards
- Continuous drift monitoring using check standards
- Recalibration if drift exceeds tolerance.

ICP-AES and ICP-MS instruments typically require drift checks every 10–20 samples.

11.6.6 Method Repeats and Over-Range Analysis

If a sample exceeds the upper detection limit of a method, laboratories automatically:

- Re-run the sample using a higher-range method (e.g., ore-grade copper)
- Perform additional dilutions
- Apply gravimetric or titration methods where appropriate.

This ensures accurate reporting of high-grade intervals.

11.6.7 Internal Sample Tracking and Chain of Custody

Laboratories maintain:

- Barcoded sample tracking

- Electronic laboratory information management systems
- Time-stamped custody logs
- Photographic documentation for some preparation stages.

This ensures traceability from receipt to reporting.

11.6.8 Batch-Level QA/QC Review

Before releasing a Certificate of Analysis, laboratories perform:

- Automated QA/QC screening
- Manual review by senior analysts
- Verification of blanks, CRMs, and duplicates
- Confirmation of method codes and detection limits.

Only batches passing internal QA/QC thresholds are released to the client.

Example of Internal QA/QC Inserts in a Typical Batch

A typical analytical batch of 40–60 samples includes introducing the items and frequency per sample for the various purposes listed in Table 11-13.

These internal controls are in addition to Doubleview's own QA/QC insertions (i.e., standards, blanks, field duplicates).

Table 11-13: Internal QA/QC Frequency and Purpose

QA/QC Type	Frequency	Purpose
Method Blank	1 per batch	Detect contamination
Laboratory CRM	1–2 per batch	Verify accuracy
Pulp Duplicate	1 per 20 samples	Assess precision
Crush Duplicate	1 per 20–40 samples	Assess preparation precision
Over-range repeats	As required	Ensure correct high-grade reporting

11.6.9 Laboratory QA/QC Reporting

Laboratories provide the following documentation:

- Raw analytical data
- QA/QC summary tables
- CRM performance plots
- Blank performance tables
- Duplicate precision statistics.

Doubleview's head office reviews these results as part of QA/QC before accepting data into the master database.

11.6.10 Summary

The internal QA/QC programs of the laboratories listed in Table 11-10 used for the Hat Project meet or exceed industry standards for mineral exploration. These combine:

- Method blanks
- CRMs
- Duplicates
- Calibration checks
- Drift monitoring
- Batch-level review.

These programs ensure that analytical results used in the Hat Project Mineral Resource estimation are reliable, reproducible, and defensible under NI 43-101 and CIM (2019) Best Practice Guidelines.

12 DATA VERIFICATION

The data verification program was designed to independently assess the reliability, completeness, and technical integrity of the information supporting this Mineral Resource estimate. Verification activities were conducted in accordance with NI 43-101 requirements and industry best practices and include both field-based observations and detailed analytical reviews. The objective of the verification process is to confirm that the data used in geological interpretation and Mineral Resource modelling are accurate, traceable, and representative of the exploration work completed to date.

A key component of the verification program was the site visit, which provided the opportunity to directly inspect the exploration infrastructure, drilling operations, sampling procedures, core handling facilities, and data management practices. The visit also allowed for an evaluation of the operational protocols in place, including health and safety, sample security, chain of custody, and QA/QC implementation. These observations form an essential foundation for assessing the reliability of the supporting datasets and for understanding the practical context in which the exploration data were generated.

Subsequent verification steps included:

- Assay verification, involving comparison of laboratory certificates with the assay database
- Re-assay of selected pulp samples (2024) to evaluate analytical reproducibility
- Verification of horizontal easting and northing coordinates to confirm spatial accuracy of drill-hole collar locations
- Review of survey measurements to ensure consistency between field survey data and the database.

Together, these components provide a comprehensive evaluation of the data quality and support the QP's conclusion regarding the suitability of the dataset for Mineral Resource estimation.

12.1 Site Visit

Independent QP, Tomasz Wawruch (Mineit), visited the site from October 1 to 3, 2025; he has no affiliation with the issuer responsible for the disclosure of Mineral Resources. The visit was supported by Doubleview's geology lead, who provided access to all relevant exploration facilities, datasets, and operational procedures. The purpose of the visit was to evaluate the exploration infrastructure, review field practices, and verify that data collection, sampling, and geological documentation are consistent with industry standards and suitable for Mineral Resource estimation. The scope of the visit included the full chain of data acquisition and management—drilling operations, geological logging, sampling governance, density determination, data security, QA/QC implementation, and inspection of both legacy and active drill sites.

12.1.1 Drilling Execution and Surveying Protocol

Diamond drilling at the Project is carried out using HQ and NQ drill sizes, both employing standard tubes. HQ drilling produces a 63.5 mm-diameter core, while NQ drilling yields one 47.6 mm in diameter. Drilling is executed according to planned collar coordinates, azimuth, and dip parameters designed to intercept target zones at the intended depths.

Drill pads (Figure 12-1) are constructed at proposed locations; however, due to irregular topography, the final pad position may deviate slightly from the planned coordinates. In recent years, to establish the definitive collar location, each pad is surveyed using a drone equipped with RTK georeferencing, anchored to a fixed base-station receiving satellite corrections. All coordinates are referenced to UTM Zone 09.



Figure 12-1: Photo Taken on 3/10/25. Active Drill Pad at Drill-Hole H102 (UTM 9N 348203, 6453897; Elevation 972 m)

Georeferenced imagery is used to confirm the final drill-hole position. Updated collar azimuths and all associated metadata are recorded in structured Excel files to ensure full traceability and data integrity. Metadata comprise hole ID, target depth, azimuth, inclination, X, Y, Z coordinates, start and end dates, drilling contractor, and logging geologist.

12.1.2 Drill-Hole Trajectory

Downhole trajectory measurements are collected using a Gyro Shot non-north-seeking gyroscope. The tool operates in multi-shot mode, offers high magnetic immunity, and records relative azimuth and

inclination essential for accurate 3-D reconstruction of drill-hole paths. Measurements are typically collected from the bottom upward, stored in the instrument's internal memory, exported to Excel, and integrated into the Project database.

12.1.3 Core Recovery and Handling

Core recovery procedures follow standard industry practices. Drill core is placed into wooden core boxes, each labeled with the drill-hole ID and box number shown in Figure 12-2. Wooden depth markers are inserted at the end of each run. Core boxes are transported from the drill site to the storage area using 4×4 quad vehicles or in basket by helicopter.



Figure 12-2: Core Recovery Process and Placement of Recovered Core in the Wooden Box

The storage area is adjacent to the core shack (Figure 12-3), which serves as the central facility for geological and geotechnical data collection. The core shack supports the following activities:

- QA/QC verification of recovery, sample integrity, and chain of custody
- Detailed geological logging
- Systematic core photography
- Sampling for assays, density, petrography, and other analyses
- Organized storage of core boxes by hole ID, depth, and date.



Figure 12-3: Core Shack Facilities and Close-Up of Core Box Storage Area

The facility includes LED lighting, core tables, and racks (capacity ~30 boxes), a scale, water supply, and dedicated core saws in a separate preparation area (Figure 12-4). Controlled access is maintained to ensure sample security.



Figure 12-4: Interior of the Core Shack and Core-Cutting Facility

12.1.4 Geological Logging

Geological logging is conducted using structured Excel templates and includes documentation of:

- Core recovery
- Lithology

- Mineralization
- Alteration
- Structural features
- Density measurements
- Geotechnical parameters (including RQD at 10 cm intervals)
- Geophysical readings (magnetometer measurements with standardized calibration)
- Sample tag information.

This workflow ensures consistent and traceable geological, geotechnical, and geophysical data collection.

12.1.5 Sampling

Sampling respects lithological, alteration and mineralization boundaries to preserve geological integrity. Sample intervals are marked with paper tags placed precisely at the split point. Maximum sample length is 3 m, with an average of ~2 m. Each tag contains a barcode and unique reference number, ensuring full traceability within the QA/QC framework. Primary samples consist of half-core. The cores are cut with a diamond saw in the cutting facility near the core shack Figure 12-4.

12.1.6 Core Density Determination

Bulk density is measured using the dry/wet weighing method based on Archimedes's principle, without wax coating. Density samples are collected at a rate of two per three core boxes (~12 m of drilling). Selected samples are marked with a yellow pen and the letter "D."

The procedure includes:

- Dry weight (W_d)
- Wet (buoyant) weight (W_w)
- Volume calculation: $V = (W_d - W_w)/\rho_{water}$)
- Bulk density: $\rho_{bulk} = W_d / V$

This method is appropriate for competent, low-porosity rocks and provides reliable density values for Mineral Resource modelling. A small proportion of samples should ideally be paired with wax-coated measurements for QA validation.

12.1.7 Petrography Samples

A suite of thin-section petrographic samples supports detailed mineralogical and textural analysis. Thin-section studies document mineral assemblages, paragenesis, crystallization history, metamorphic textures, deformation mechanisms, and hydrothermal alteration (Precision Petrographic Ltd., 2024). These observations refine interpretations of lithology, alteration zoning, and mineralization styles, contributing directly to geological modelling.

12.1.8 Photographic Documentation

High-resolution photographs are taken after logging and verification of all depth markers and sample tags. Core trays are photographed three boxes at a time, and images are stored in drill-hole–specific folders, providing a systematic visual record.

12.1.9 Data Storage and Security

All electronic geological and analytical files are stored on the company's primary server in Vancouver. Data stored on the server is backed up regularly as per the established protocol, ensuring secure and consistent data retention. A full backup is also maintained on a portable hard drive on site as a precaution. Data security protocols are appropriate for the Project's scale and stage.

12.1.10 QA/QC Program

The QA/QC program includes duplicates, blanks, and CRMs. Control samples are inserted at a frequency of one every 20 primary samples, resulting in each control type being introduced every 60 samples (~5% of total samples). Although the full program was not audited during the site visit, its structure and implementation were reviewed in the context of logging and sampling workflows.

12.1.11 Sample Dispatch Procedure

Each sample is placed in a transparent plastic bag, weighed, and recorded in a dedicated tracking spreadsheet. Samples are grouped into rice bags and arranged sequentially. Each drill hole is assigned a distinct color-coded adhesive tag, affixed to the rice bags to ensure clear identification during laboratory intake.

The lead geologist maintains direct communication with the laboratory and conducts periodic visits to oversee sample handling and preparation, ensuring alignment with QA/QC standards.

12.1.12 Site Inspection

The QP inspected nine drill pad areas, representing approximately 53 drill-hole traces, including both early and recent drilling campaigns (Figure 12-5). These observations confirm the continuity and authenticity of the Project's exploration record.



Figure 12-5: Two Vintage Drilling Sites with Remaining Rod Tubes and Anchor Traces

The inspection concluded with a visit to the active drill rig, where live core extraction was observed. Freshly recovered cores were subsequently reviewed at the core shack (Figure 12-6), including boxes 38 and 39 (165–172 m) and box 45 (196–200 m). These intervals display visible mineralization characterized by strong potassic alteration.



Figure 12-6: Borehole H102 Displayed Core Boxes—45 (Right), 38 and 39 (Left)

Overall, the site inspection demonstrates that the Project is technically well executed and professionally managed. Field operations, data collection practices, and sample handling procedures reflect strong commitment to data integrity and support the reliability of the dataset used for Mineral Resource estimation.

12.2 Re-Assay of Selected Pulp Samples

Micon International Limited conducted an independent audit in July 2024, including a site visit from July 20 to 23. During the data validation process, Micon's QP selected 59 pulp samples for re-assay (Table 12-1).

Table 12-1: Summary of Pulp Samples Selected for Re-Assay

Assay Laboratory	Drill-Hole ID	From (m)	To (m)	Sample Tag Number
Bureau Veritas	H036	231.2	234.2	B0029515
Bureau Veritas	H036	271.3	274.3	B0029532
Bureau Veritas	H036	329.0	330.0	B0029556
Bureau Veritas	H037	16.0	19.0	B0029867
Bureau Veritas	H037	155.5	157.6	B0029922
Bureau Veritas	H037	655.4	656.6	A0009626
Bureau Veritas	H038	70.0	72.0	3824023
Bureau Veritas	H038	123.8	125.0	3824044
Bureau Veritas	H039	107.8	108.3	3824242
Bureau Veritas	H039	261.2	264.2	3824306
Bureau Veritas	H039	550.0	551.0	3824445
Bureau Veritas	H039	607.0	609.3	3824468
Bureau Veritas	H040	55.9	57.0	3824491
Bureau Veritas	H040	223.3	226.2	3824561
Bureau Veritas	H040	292.0	293.0	3824592
Bureau Veritas	H040	403.2	404.2	3824644
Bureau Veritas	H041	243.5	246.5	3824876
Bureau Veritas	H041	314.7	317.7	3824912
ALS	H043	160.0	161.0	3850290
ALS	H058	314.5	315.5	C0052447
ALS	H058	359.0	360.0	C0052473
ALS	H058	577.0	578.0	C0052621
ALS	H059	54.0	57.0	C0052766
ALS	H059	118.0	119.6	C0052801
ALS	H060	74.7	76.0	C0053076
ALS	H060	76.0	77.0	C0053078
ALS	H061	48.4	51.0	H655516
ALS	H061	64.3	66.0	H655523
ALS	H061	66.0	69.0	H655524
ALS	H062	129.0	130.0	H655808
ALS	H062	324.0	327.0	H655951

Assay Laboratory	Drill-Hole ID	From (m)	To (m)	Sample Tag Number
ALS	H063	141.0	142.5	H650609
ALS	H063	162.0	165.0	H650618
ALS	H063	177.0	179.0	H650625
ALS	H063	302.0	303.0	H650680
ALS	H064	45.4	48.0	H650767
ALS	H064	55.0	56.0	H650774
ALS	H065	54.0	55.0	H655019
ALS	H065	56.0	57.0	H655022
ALS	H066	29.6	30.7	H655209
ALS	H066	45.0	47.6	H655215
ALS	H067	18.0	19.7	H656054
ALS	H067	60.0	63.0	H656072
ALS	H067	300.0	301.0	H656215
ALS	H068	12.0	15.0	H656351
ALS	H068	99.6	101.4	H656386
ALS	H068	550.0	551.0	H651157
ALS	H068	563.0	564.0	H651167
ALS	H069	55.9	58.0	H651221
ALS	H069	74.0	75.0	H651228
ALS	H069	375.0	376.0	H651407
ALS	H070	42.0	45.0	H651515
ALS	H070	159.0	162.0	H651582
ALS	H071	46.0	48.0	H651766
ALS	H071	102.0	105.0	H651787
ALS	H071	132.0	135.0	H651802
ALS	H071	302.0	303.0	H651901
ALS	H071	308.0	309.0	H651907
ALS	H071	396.0	398.0	H651959

These samples were resubmitted to the same two laboratories that conducted the original assays: Bureau Veritas and ALS. A comparison of original and re-assay results, illustrated in scatter plots (Figure 12-7 to Figure 12-9), show strong, unbiased concordance between datasets. Accordingly, the original assay data are considered reliable for use in the Mineral Resource estimate.

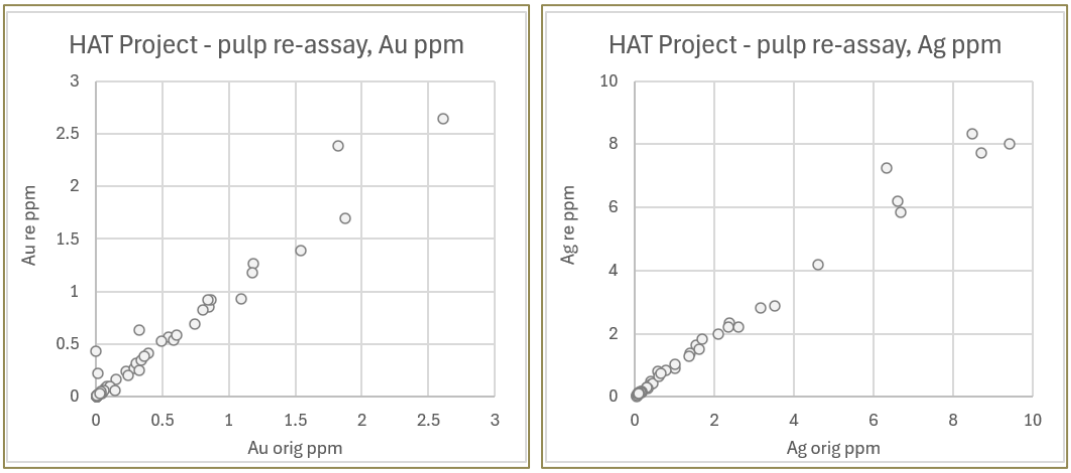


Figure 12-7: Unbiased Scatter Plot for Gold (Au) and Silver (Ag) Comparing Original and Re-Assayed Datasets

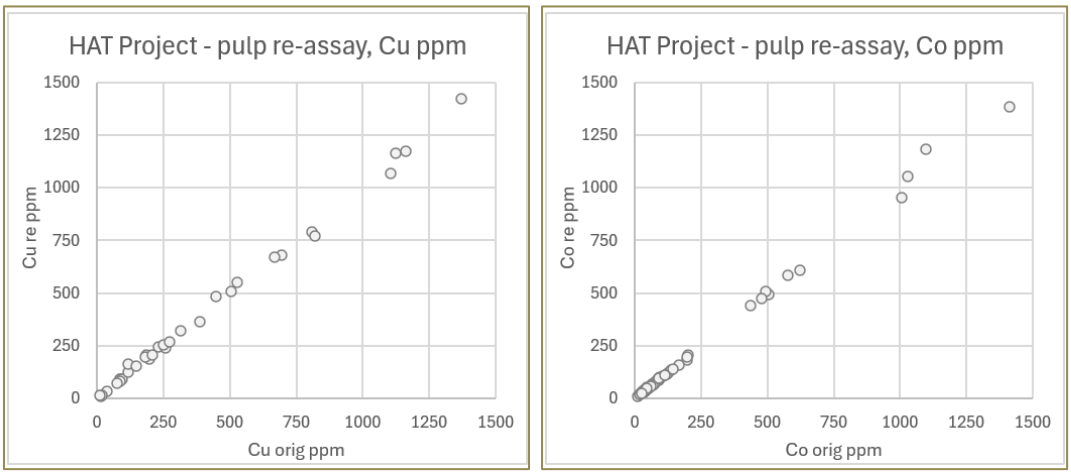


Figure 12-8: Unbiased Scatter Plot for Copper (Cu) and Cobalt (Co) Comparing Original and Re-Assayed Datasets

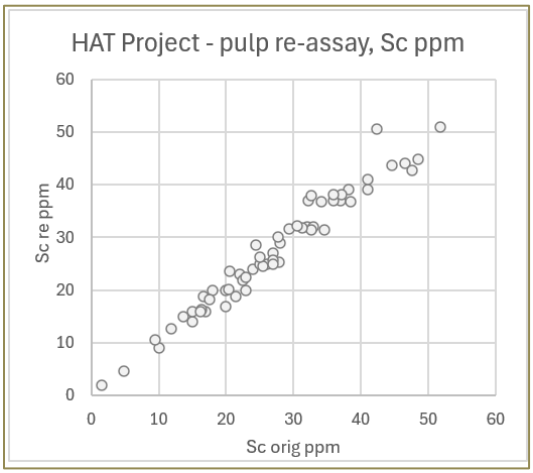


Figure 12-9: Unbiased Scatter Plot for Scandium (Sc) Comparing Original and Re-Assayed Datasets

Table 12-2 summarizes the details for the selected re-assay sample pulps, along with the original and re-assay information for copper, gold, silver, cobalt, and scandium.

Table 12-2: Comparison of Numerical Outputs for Original and Re-Assayed Samples

Sample Tag Number	Original (ppm)					Re-Assay (ppm)				
	Au	Ag	Co	Cu	Sc	Au	Ag	Co	Cu	Sc
B0029515	0.155	0.3	122.2	1,371.1	22	0.165	0.3	115.1	1,423.3	23
B0029532	0.031	<0.1	34.8	385.9	15	0.031	<0.1	32.6	365.4	14
B0029556	1.538	4.6	198.6	27,110	10	1.389	4.2	183.8	>10,000	9
B0029867	0.009	<0.1	44.5	14.4	41	0.005	<0.1	45.1	12.7	39
B0029922	0.007	<0.1	25.6	86.2	18	0.006	0.1	26.3	93.9	20
A0009626	0.231	0.4	69.1	3,625.3	20	0.247	0.5	70.9	3,699.8	20
3824023	0.019	<0.1	70.9	37.2	37	0.024	<0.1	68.3	34.6	37
3824044	0.01	<0.1	34	88.3	41	0.007	<0.1	32.6	93.6	41
3824242	0.008	<0.1	31.1	18	27	0.005	<0.1	29.1	15.1	27
3824306	0.069	0.1	38	808.9	26	0.08	0.1	35.3	788.5	25
3824445	0.849	2.1	1,005.7	12,300	32	0.858	2	952.9	>10,000	32
3824468	0.004	<0.1	12.8	13.3	33	<0.002	<0.1	11.6	17.5	32
3824491	1.184	2.6	51.5	5,381.2	36	1.262	2.2	50.7	5,387.9	37
3824561	0.044	0.4	37.3	696.1	24	0.034	0.4	34.6	682	24
3824592	0.287	6.6	503.3	6,121.1	25	0.272	6.2	492.9	6,235.6	25
3824644	1.178	9.4	494.5	22,860	17	1.177	8	507.8	>10,000	16
3824876	0.027	<0.1	40	231.5	28	0.028	<0.1	40.8	246.5	29
3824912	0.008	<0.1	19.1	96.4	15	0.006	<0.1	18.8	94.1	16
3850290	0.543	6.33	622	22,200	13.6	0.565	7.25	609	22,400	15
C0052447	0.08	0.08	126.5	669	51.8	0.099	0.09	119.5	671	50.9
C0052473	0.395	1.02	93	5,320	25.5	0.413	0.9	86.3	5,140	24.6
C0052621	0.582	1.54	479	10,500	29.3	0.535	1.65	47.4	10,500	31.6
C0052766	0.242	0.46	52.7	820	44.6	0.207	0.42	54.2	770	43.6
C0052801	0.324	0.32	90.1	1,105	38.1	0.634	0.27	99.1	1,070	39
C0053076	1.875	3.51	201	5,470	32.6	1.7	2.89	205	5,680	31.4
C0053078	1.82	1.39	1,095	1,965	16.7	2.38	1.37	1,185	2,180	18.8
H655516	0.016	0.05	64	119.5	42.4	0.22	0.05	66.9	128	50.5
H655523	0.604	0.59	95.8	1,780	32.2	0.587	0.82	101.5	1,755	36.9
H655524	0.303	0.18	138	258	32.6	0.318	0.18	136	240	37.9
H655808	0.746	0.61	106.5	2,030	46.5	0.696	0.64	106	2,150	44.1
H655951	0.329	0.06	32	252	20	0.248	0.04	31.6	256	1,680[1]
H650609	0.341	0.12	78.3	316	22.5	0.346	0.13	80.8	323	21.8
H650618	0.866	3.16	1,030	7,560	9.4	0.924	2.83	1,055	7,410	10.6
H650625	0.845	1.62	578	4,960	16.4	0.917	1.52	583	4,850	16.4

Sample Tag Number	Original (ppm)					Re-Assay (ppm)				
	Au	Ag	Co	Cu	Sc	Au	Ag	Co	Cu	Sc
H650680	18.4	1.37	2,330	18,700	4.8	>10.0	1.3	2,410	18,900	4.7
H650767	0.492	0.31	39.8	503	34.6	0.53	0.29	40.4	509	31.4
H650774	6.78	2.37	168.5	4,010	38.4	6.76	2.35	159.5	3,750	36.8
H655019	2.61	8.7	77.2	16,050	27.9	2.64	7.71	74.3	16,450	25.3
H655022	7.46	19.65	146	33,800	21.4	7.36	19.3	139	34,400	18.8
H655209	0.025	0.09	45	117.5	37.1	0.026	0.12	53.9	162.5	38.2
H655215	0.013	0.18	68.3	526	48.5	0.015	0.18	65.4	550	44.8
H656054	0.011	0.18	40	195	47.6	0.012	0.14	40.7	190.5	42.7
H656072	0.004	0.05	17.4	85.6	20.6	0.005	0.02	18.9	82.1	23.5
H656215	15.05	6.68	1410	133,500	1.5	15.4	5.85	1,385	131,000	1.9
H656351	0.008	0.15	34.9	196.5	34.1	0.009	0.15	39.7	190	36.7
H656386	0.005	0.08	31.5	77	25	0.008	0.04	36	72.4	26.2
H651157	1.095	1.7	92.8	7,900	31.3	0.934	1.82	97.2	8,340	31.8
H651167	0.808	8.48	435	57,600	11.8	0.823	8.33	442	58,500	12.6
H651221	0.021	0.16	47.3	187	27.7	0.022	0.18	44.1	206	30.1
H651228	0.05	1.02	91.3	1,125	24.4	0.05	1.03	97.4	1,165	28.6
H651407	0.103	0.15	41.3	1,160	23	0.102	0.14	39.4	1,175	19.9
H651515	0.06	0.11	26	181.5	17.6	0.063	0.13	28.2	196	18.3
H651582	0.04	0.08	35	275	16.2	0.031	0.1	35.2	268	15.9
H651766	0.011	0.11	30.1	149.5	22.9	0.011	0.12	30.2	155.5	22.5
H651787	0.039	2.35	61.2	1,595	35.9	0.052	2.21	59.3	1,585	38.2
H651802	0.148	0.09	46.9	209	30.5	0.057	0.1	47.7	209	32.2
H651901	<0.001	0.78	196	3,490	27	0.436	0.83	200	3,710	25.6
H651907	0.364	0.67	113	4,200	26.9	0.385	0.75	113	4,410	24.9
H651959	0.032	0.09	26.2	446	20.4	0.03	0.12	26.6	483	20.2

Note: [1]The Scandium value of 1680 ppm, along with all other grades generated during the 2024 re assay program conducted by Micon, originates from an independent exercise performed for the purposes of their own MRE study. The objective of that program was limited to assessing the reproducibility of original assay grades by re assaying a selection of pulp samples using the same laboratories that analyzed the primary samples. Micon's overall conclusion—that original grades show satisfactory reproducibility across the five variables tested (Au, Cu, Ag, Co, Sc)—is acknowledged. However, the individual re assay values produced by Micon, including localized discrepancies between original and re assayed grades, do not form part of the laboratory certificates nor the validated assay database used for the 2026 Mineral Resource Estimation supervised by Mineit. These re assay results were not provided as part of the dataset for the current PEA, and they are not used in any estimation or classification procedures supporting the 2026 study. For this reason, the Micon 2024 re assay outputs are not considered relevant to the 2026 Mineral Resource Estimation and do not influence the results or conclusions of the current work.

12.3 Assay Verification

12.3.1 Purpose of Assay Verification

The objective of the assay verification process is to confirm that the assay database is complete, internally consistent, and accurately reflects the analytical results reported by the commissioned laboratories. Verified assays form the basis for Mineral Resource estimation and classification; therefore, ensuring fidelity between laboratory certificates and the database is essential.

12.3.2 Data Sources and Traceability

Doubleview provided all analytical certificates in both PDF and XLS formats. The assay database is structured with good traceability: each record in Table 12-3 clearly displays the sample tag and the corresponding certificate identification code. In all, 202 certificates were registered in the database. Of these, 187 certificates were examined in detail and compared directly against the database entries.

Table 12-3: Schematic Representation of Traceability within the Hat Project Database

BHID	Sample Tag	From (m)	To (m)	Interval	Certificate ID	Date Finalized
H001	2642528	40	41.8	1.8	MA0110-MAR16	2016-03-29
H001	2642529	42.5	45	2.5	MA0110-MAR16	2016-03-29
H001	2642530	45	48	3	YVR1610196	2016-11-21
H001	2642531	48	51	3	YVR1610196	2016-11-21
H001	2642533	51	52.6	1.6	YVR1610196	2016-11-21
H001	2326751	52.6	54	1.4	SMI13000044	2013-06-10
H001	2326752	54	56.4	2.4	SMI13000044	2013-06-10
H001	2326753	56.4	58.5	2.1	SMI13000044	2013-06-10
H001	2642534	58.5	61	2.5	YVR1610196	2016-11-21
H001	2642535	61	64	3	MA0110-MAR16	2016-03-29
H001	2642536	64	67	3	MA0110-MAR16	2016-03-29
H001	2642537	67	70	3	MA0110-MAR16	2016-03-29
H001	2642538	70	73.2	3.2	MA0110-MAR16	2016-03-29
H002	2620454	14	17	3	YVR1610196	2016-11-21

Note: BHID = borehole ID.

12.3.3 Certificate Extraction and Processing

Extraction of analytical results from certificates was time-consuming due to the structure of the laboratory reporting protocols, which are not numerically straightforward to parse. Several operational challenges were encountered during extraction and were resolved during the validation process.

The database under review reflects primary assay registrations. However, it does not include a traceability mechanism to track assay updates, re-assays, or changes arising from QA/QC procedures. The certificates

used in this verification represent the "current" or primary certificates forming the main analytical stream. Consequently, occasional or seasonal laboratory checks performed outside the main analytical workflow may have introduced changes in the operational database that are not reflected in the validated certificate set. It was therefore anticipated that some discrepancies would remain between the operational database and the validated dataset at the end of the verification process.

An ideal assay database should maintain version control and annotation, documenting analytical stages, re-assays, and administrative updates. Such structure would facilitate interpretation of differences encountered during verification.

Doubleview provided the most recent database, closed on November 9, 2025; that version was used for verification.

Observations on Analytical Certificates and their Impact on Verification

The following observations relate to laboratory reporting protocols. These issues influenced the QP's actions and affected the comparison process between laboratory certificates and the assay database.

- Handling of "less-than" values
  Laboratories commonly report below-detection results using the "<" symbol. Most software does not recognize "<" as a numeric character. A standard convention is to replace "<" with a minus sign "–", and this should be explicitly documented in the certificate metadata.
- Non-Analyzed or missing samples
  Certificates often include entries such as "n.a.", "nss", "not received" or "missing."
  A consistent convention is to replace these with "–99.9", a value unlikely to coincide with any analytical detection limit. This ensures that non-analyzed samples are clearly distinguishable during processing.
- Variable detection limits in silver analyses
  Silver assays were reported with several detection limits (2, 1, 0.5, and 0.1 ppm).
  If below-detection values are replaced with half the detection limit, results from different analytical groups processed with significantly lower detection limits may overlap numerically, creating ambiguity. To avoid confusion, all below-detection silver values were standardized to 0.01 ppm. As a result, some low-end values differ from Doubleview's internal interpretation.
- Changes in reporting units over time
  Gold assays in older certificates were reported in ppb, whereas more recent certificates use ppm. Because verification was not performed chronologically, this unit inconsistency introduced additional complexity during data processing.
- High-grade over-limit assays
  Over-limit assays requiring secondary calibration are reported in separate columns. Laboratory approaches to handling over-limit values vary, and reconciling these differences requires careful attention. This step is prone to error and demands high concentration during verification.

General Comments on Laboratory Reporting Practices

The issues noted above could be mitigated through improved communication between laboratories and exploration and mining companies, including standardized reporting conventions and clearer metadata. QA/QC professionals typically follow laboratory protocols closely; however, during periods of high analytical demand, inconsistencies or delays in reporting can introduce additional technical challenges.

12.3.4 Verification Outcome

A total of 17,698 records were compared between the laboratory-certified analytical protocols and the Hat Project database; in the tabulated results, the "Lab" and "Geo" columns denote the laboratory assays and the geological database entries, respectively. Table 12-4 gives an example of the data used for the comparison, leaving out Sample IDs 10 through 17,688. The database contains 23,366 records as of the cut-off date of November 9, 2025. Accordingly, the verification covered 75.7% of all assay entries.

Table 12-4: Comparative Summary of Raw Analytical Inputs, Cleaned Database Outputs, and Recorded Differences

Record Number	Sample ID	Ag Lab	Ag Geo	Ag Diff	Au Lab	Au Geo	Au Diff	Co Lab	Co Geo	Co Diff	Cu Lab	Cu Geo	Cu Diff	Sc Lab	Sc Geo	Sc Diff
1	H652651	0.04	0.04	0	0.011	0.011	0	16.3	16.3	0	65.5	65.5	0	24	24	0
2	H652652	0.13	0.13	0	0.009	0.009	0	25.1	25.1	0	207	207	0	23	23	0
3	H652653	0.2	0.2	0	0.013	0.013	0	50.6	50.6	0	226	226	0	28	28	0
4	H652654	0.07	0.07	0	0.02	0.02	0	39	39	0	173	173	0	29.8	29.8	0
5	H652655	0.09	0.09	0	0.024	0.024	0	48	48	0	338	338	0	34.3	34.3	0
6	H652656	1.02	1.02	0	0.27	0.27	0	57.4	57.4	0	2580	2580	0	31.2	31.2	0
7	H652657	0.12	0.12	0	0.051	0.051	0	35.6	35.6	0	460	460	0	27.7	27.7	0
8	H652658	0.04	0.04	0	0.026	0.026	0	28	28	0	123	123	0	25.6	25.6	0
9	H652659	0.03	0.03	0	0.039	0.039	0	12.4	12.4	0	91.5	91.5	0	24.8	24.8	0
10	H652660	0.04	0.04	0	0.034	0.034	0	14.4	14.4	0	127	127	0	23.9	23.9	0
Records 11 through 17688 omitted for example																
17689	Q461991	0.01	0.01	0	0.5	0.05	0.45	40	40	0	350	350	0	20	20	0
17690	Q461992	0.01	0.01	0	0.5	0.05	0.45	50	50	0	120	120	0	40	40	0
17691	Q461993	0.01	0.01	0	0.5	0.05	0.45	50	50	0	310	310	0	50	50	0
17692	Q461994	0.18	0.18	0	0.5	0.05	0.45	50	50	0	980	980	0	40	40	0
17693	Q461995	0.005	0.005	0	0.5	0.05	0.45	30	30	0	220	220	0	20	20	0
17694	Q461996	0.005	0.005	0	0.5	0.05	0.45	30	30	0	260	260	0	10	10	0
17695	Q461997	0.02	0.02	0	0.5	0.05	0.45	110	110	0	1530	1530	0	20	20	0
17696	Q461998	0.03	0.03	0	0.5	0.05	0.45	40	40	0	370	370	0	10	10	0
17697	Q461999	0.005	0.005	0	0.5	0.05	0.45	30	30	0	260	260	0	20	20	0
17698	Q462000	0.01	0.01	0	0.5	0.05	0.45	90	90	0	120	120	0	40	40	0

Notes: Lab = laboratory assays; Geo = geological database values

A total of 160 record differences were identified during the comparative review between the primary analytical certificates and the assay database, representing approximately 0.9% of all verified assay

records. This small proportion reflects a high level of consistency between the laboratory certificates and the database entries.

These differences correspond to re-assays generated through the routine QA/QC process, where CRM performance triggers secondary certification. In these cases, the Hat Project QA/QC team requested re-analysis in accordance with established procedures, and the updated certified values were correctly incorporated into the assay database. The laboratory's internal review and corrective actions further confirm that the re-assay results represent the most reliable analytical values.

Overall, the validation results are satisfactory. The review confirms full agreement for the five validated analytical variables between the laboratory-certified results and the assay grades reported by Doubleview, demonstrating that both the laboratory's quality controls and the Hat Project QA/QC program are functioning effectively.

12.4 North and East Coordinates Verification

A field visit to the Hat exploration camp was completed as part of the data verification process. During the visit, a traverse of both historical (vintage) and recent drill pads was undertaken. At each visited pad location, geographic coordinates were recorded in degrees–minutes–seconds (DMS) format using a handheld GPS application.

The purpose of this exercise was not to verify the collar coordinates of individual drill holes one by one, but rather to obtain a representative coordinate for each drill pad area. This approach is more efficient and appropriate for a short field campaign, while still providing a meaningful check on the spatial reliability of the drilling database. In the official drilling database, all drill holes originating from the same pad are assigned identical collar coordinates, reflecting the standard practice used during exploration. This convention records the pad location rather than the precise collar position of each individual hole. The CP's verification approach, comparing field-measured pad coordinates against the average collar position of drill holes at each pad is therefore fully aligned with the exploration procedures used to build the database.

The recorded DMS coordinates were subsequently converted to decimal degrees and then transformed into UTM coordinates using UTM Zone 9N (NAD 83). The online tool coordinates-converter.com was used for these transformations.

For each pad, the transformed UTM coordinates were compared against the average collar position calculated from all drill holes located at that pad. This method allows a direct assessment of whether the field-measured pad location is spatially consistent with the drill-hole collar data stored in the database.

A tolerance of 15 m was adopted as the threshold for acceptable horizontal deviation. The field measurement of pad locations carries an inherent degree of uncertainty, particularly for vintage drill holes where clear collar identification is often absent or obscured. The 15 m criterion corresponds to the nominal 15 m block size used in the Mineral Resource model and is therefore considered appropriate for determining whether any horizontal discrepancies are material. This tolerance also provided a practical and efficient framework for the two-day site visit, allowing the verification to focus on meaningful spatial

differences rather than minor positional noise. In total, 53 drill holes were included in this verification exercise, grouped according to their respective pad locations as summarized in Table 12-5. The Hat Camp coordinates were first recorded as a reference point, followed by measurements at nine pad locations, summarized in the following tables:

- Table 12-5 presents the average UTM easting and northing for drill-hole collars at each pad.
- Table 12-6 lists the independently measured DMS coordinates collected during the site visit.
- Table 12-7 provides comparative results, including calculated horizontal differences.
- Figure 12-10 illustrates the spatial distribution of the pads included in the horizontal coordinate verification.

The comparison between field-measured pad coordinates and the corresponding average drill-hole collar positions shows that all evaluated locations fall within the QP's established 15 m horizontal tolerance, which is considered reasonable for the purpose of this check. This horizontal-coordinate verification is not intended to question the operational procedures of the company responsible for collar surveying, but rather to rule out any major location errors that could displace drill holes beyond the property boundary or by hundreds of metres from their correct positions. No material discrepancies were identified, and the observed differences are consistent with expected positional variability. Overall, the verification confirms that the horizontal coordinates in the drilling database are reliable for use in Mineral Resource estimation and are consistent with the exploration practices employed at the Hat Project.

Table 12-5: Average UTM Coordinates (Easting and Northing) for Drill-Hole Collars by Pad

Ref. PAD Number	Drill-Hole Number per Pad	Mean Easting	Mean Northing
1	2	348203.0	6453897.0
2	8	347963.0	6453927.0
3	15	347867.1	6453948.2
4	2	347692.0	6453908.0
5	11	347866.0	6453619.0
6	8	347471.0	6454073.0
7	2	347620.0	6454120.0
8	2	347518.0	6454430.0
9	3	347703.0	6454749.0
Total	**53**		

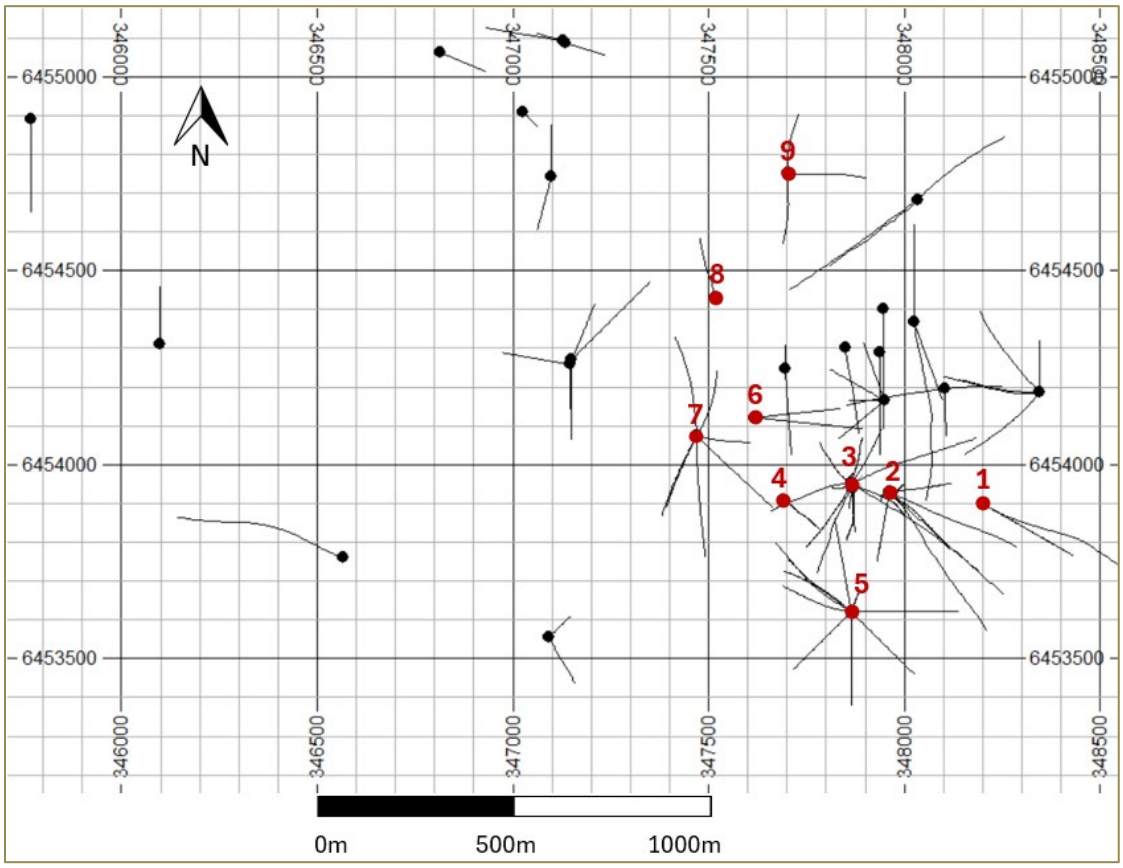


Figure 12-10: Spatial Distribution of the Pads Included in the Horizontal Coordinate Verification

Table 12-6: Independently Measured DMS Coordinates Collected During the Site Visit

Ref. PAD Number	Latitude	Longitude	UTM Easting	UTM Northing
1	58°12'0"N	131°34'59"W	348192.7	6453886.2
2	58°12'1"N	131°35'14"W	347949.1	6453926.5
3	58°12'2"N	131°35'19"W	347868.7	6453960.5
4	58°12'0"N	131°35'29"W	347703.1	6453905.0
5	58°11' 51"N	131°35'18"W	347864.3	6453610.8
6	58°12'11"N	131°35'57"W	347259.3	6454262.6
7	58°12'7"N	131°35'35"W	347613.5	6454125.1
8	58°12'17"N	131°35'41"W	347527.5	6454438.0
9	58°12'27"N	131°35'31"W	347702.6	6454740.9
Hat Camp	58°12'2"N	131°36'35"W	346628.5	6454007.8

Table 12-7: Numerical Comparison of Northing, Easting, and Horizontal Discrepancies

PAD	UTM East	UTM North	Mean East	Mean North	Delta East	Delta North	Horizontal Difference (m)
1	348192.7	6453886.2	348203.0	6453897.0	−10.27	−10.81	14.91
2	347949.1	6453926.5	347963.0	6453927.0	−13.88	−0.50	13.89
3	347865.1	6453956.2	347867.1	6453948.2	−1.99	7.96	8.20
4	347703.1	6453905.0	347692.0	6453908.0	11.13	−3.01	11.52
5	347864.3	6453610.8	347866.0	6453619.0	−1.72	−8.19	8.37
6	347473.1	6454075.2	347471.0	6454073.0	2.08	2.23	3.05
7	347613.5	6454125.1	347620.0	6454120.0	−6.47	5.14	8.26
8	347527.5	6454438.0	347518.0	6454430.0	9.53	8.04	12.47
9	347702.6	6454740.9	347703.0	6454749.0	−0.40	−8.11	8.12

12.5 Survey Measurement Verification

During the earliest drilling campaigns, downhole surveys were collected using a Reflex EZ-Trac instrument. As the program advanced, the survey procedure was standardized, and subsequent drill holes were surveyed exclusively with a gyroscope-based system.

To determine the three-dimensional orientation of drill-hole trajectories and the corresponding sample positions, the Hat Project employs a Gyro Shot tool. This instrument is a robust, non–north-seeking gyroscope that requires an initial setup using an external reference point to establish orientation prior to surveying. The tool is suitable for high-angle and deep drill holes and provides reliable azimuth and inclination measurements when properly configured.

A review of the survey database revealed that a significant number of drill holes lack downhole survey records. In several cases, the total drilled length or the number of survey stations is incomplete or inconsistent with standard industry requirements. These gaps indicate that survey data were not systematically collected or properly registered during certain phases of the drilling program.

A summary of the drill-hole survey coverage, including the number of surveyed holes, missing records, and distribution by campaign, is presented in Table 12-8 and Table 12-9.

The review of downhole survey data indicates that survey practices have evolved significantly over the life of the Project. Early drilling campaigns (2013–2016) show a high proportion of missing survey records, with several years relying entirely on unsurveyed trajectories. From 2019 onward, survey coverage improves, and by 2020–2025 most of the drilling was completed using Gyro Master tools, reflecting a transition to more reliable and industry-standard survey methods.

Table 12-8: Annual Summary of Downhole Survey Methods and Drilled Metres

Year	Survey Method	Partial Drilled Length (m)	Participation (%)	Total Drilled Length (m)
2013	Missing Survey Records	1,305	48	2,704
2013	REFLEX EZ-TRAC	1,399	52	
2014	Missing Survey Records	5,125	100	5,125
2015	Missing Survey Records	553	100	553
2016	Missing Survey Records	1,392	59	2,345
2016	REFLEX EZ-TRAC	953	41	
2019	Missing Survey Records	111	6	1,843
2019	REFLEX EZ-TRAC	1,732	94	
2020	Gyro Master	1,754	78	2,249
2020	Missing Survey Records	495	22	
2021	Gyro Master	2,476	100	2,476
2022	Gyro Master	1,019	30	3,433
2022	Missing Survey Records	2,414	70	
2023	Gyro Master	6,554	62	10,497
2023	Missing Survey Records	3,944	38	
2024	Gyro Master	9,800	100	9,800
2025	Gyro Master	5,847	69	8,523
2025	Missing Survey Records	2,676	31	

Table 12-9: Overall Downhole Survey Coverage and Data Totality

Survey Status	Summed Length (m)	Participation (%)
Missing Survey Records	18,014	36
Surveyed Trajectory	31,534	64
Total Drilled Length	**49,548**	

Across the full drilling database, a total of 49,548 m of drilling was evaluated. Of this, 31,534 m (64%) have a recorded downhole survey, while 18,014 m (36%) lack survey data. The distribution of missing surveys is not uniform: older campaigns account for most of the gaps, whereas recent drilling demonstrates near-complete survey coverage.

Overall, the dataset shows a clear improvement in survey quality and consistency through time. While the presence of unsurveyed intervals in earlier drilling introduces some uncertainty, the increasing use of gyroscope-based tools in later campaigns provides a reliable foundation for current geological interpretation and Mineral Resource modelling.

12.5.1 Quantification of Trajectory Uncertainty

Given the presence of drill holes without downhole survey data, a discrepancy analysis is required to evaluate the potential impact of missing surveys on drill-hole trajectories and sample positions. The objective is to determine whether the absence of survey measurements could introduce material deviations relative to the surveyed population.

To achieve this, all drill holes with existing gyroscope or Reflex EZ-Trac surveys are temporarily re-processed as if they were unsurveyed. For each hole, the trajectory will be recalculated using only the initial collar azimuth and inclination, following the same simplified approach applied to drill holes lacking survey data. The resulting unsurveyed trajectories will then be compared against the actual surveyed paths.

This method provides a direct and practical measure of the magnitude and frequency of deviations that would occur if surveyed holes were treated as unsurveyed. The analysis helps to establish whether the simplified trajectory assumption used for unsurveyed drill holes is reasonable, and whether any systematic or material discrepancies need to be considered in geological interpretation or Mineral Resource modelling.

With two distinct populations of drill holes—those with downhole surveys and those without—it is possible to parse corresponding trajectory intervals and measure the spatial deviation between surveyed and unsurveyed paths. Figure 12-11 is an example of the resulting comparison, including the reconstructed unsurveyed trajectory the actual surveyed drill-hole path, and the interval-by-interval distance vectors between the two trajectories, which form the basis of the deviation analysis. This comparison allows the impact of missing survey data to be quantified and provides a direct assessment of the potential positional uncertainty affecting the 36% of drill holes that lack downhole survey measurements.

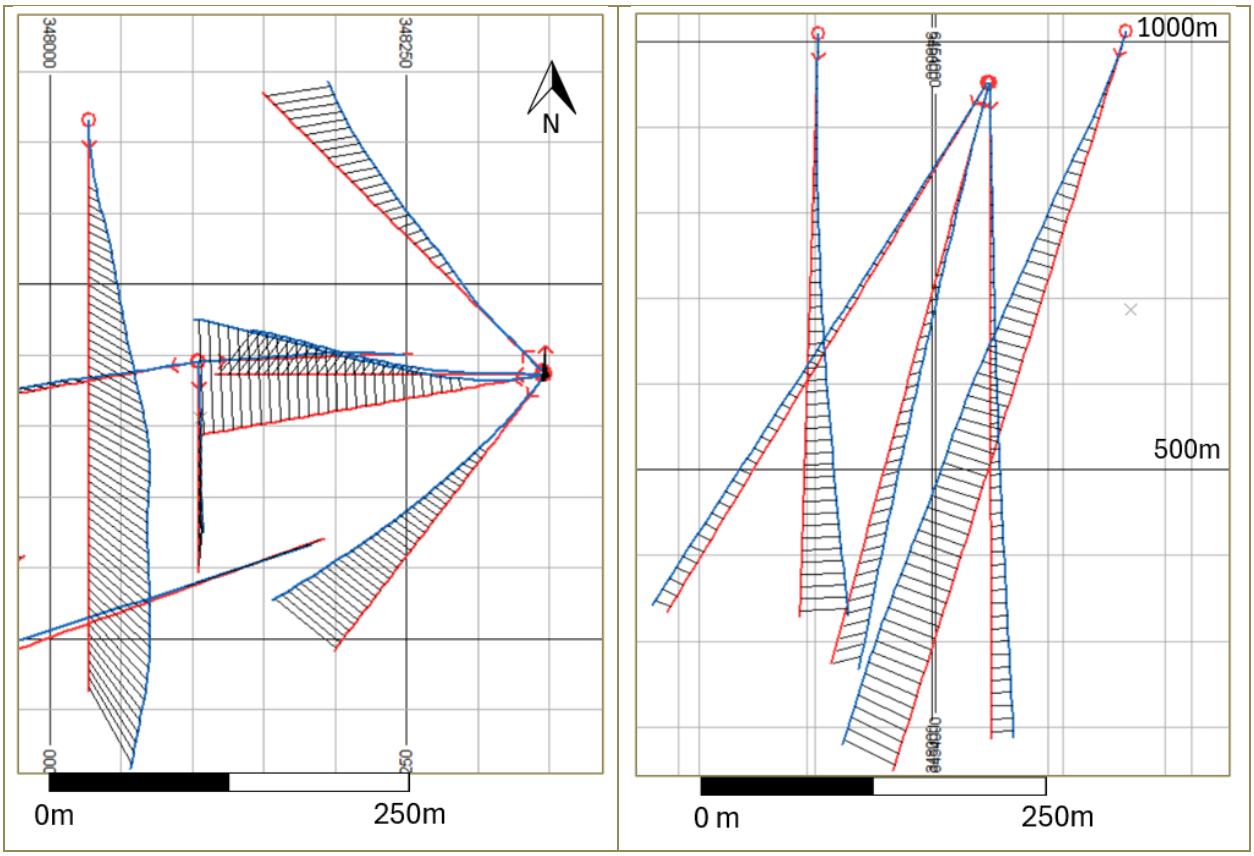


Figure 12-11: Comparative Drill-Hole Trajectories: Plan View (Left) and Vertical Section (Right) (Red Line = Unsurveyed Trajectory; Blue Line = Actual Surveyed Path; Black Line = Interval Distance)

12.5.2 Result Discussion

Figure 12-12 and Figure 12-13 illustrate the expected spatial drift between surveyed and unsurveyed drill-hole trajectories and provide a defensible buffer for sample-location uncertainty, as follows:

- At a depth of 600 m, the expected average horizontal deviation between surveyed and unsurveyed trajectories is 30 m, with a 90% confidence interval of 11 to 56 m.
- The corresponding vertical displacement at 600 m depth averages 6 m, with a 90% confidence interval of 0.1 to 12 m.

These results define a realistic range of positional uncertainty that can be expected when drill holes lack downhole survey data.

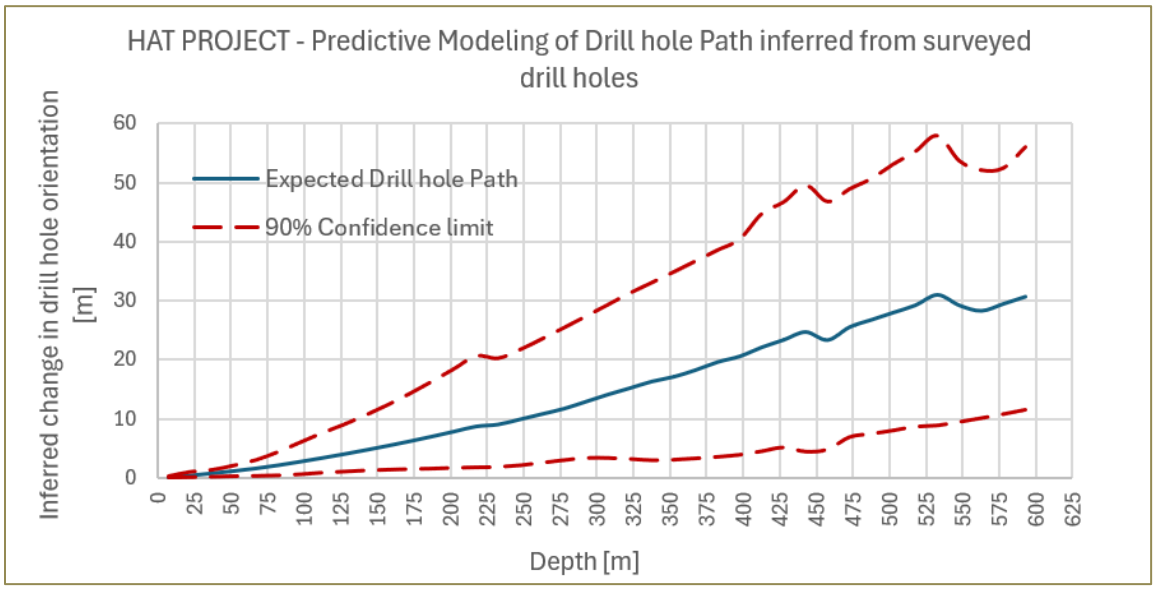


Figure 12-12: Expected Spatial Drift of Drill-Hole Trajectories (90% Confidence Limits)

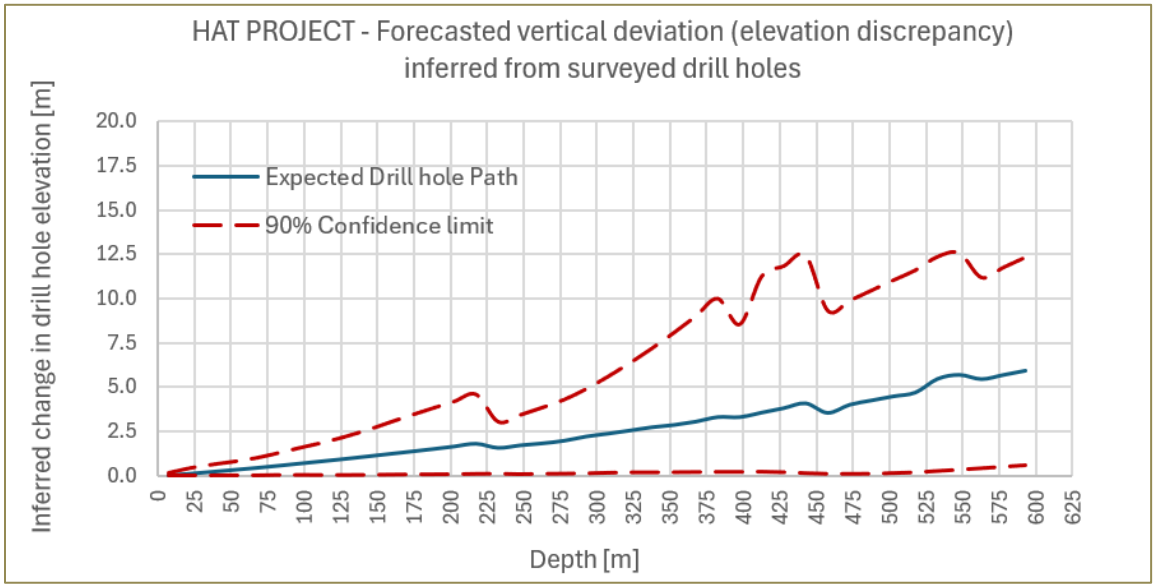


Figure 12-13: Expected Vertical Drift of Drill-Hole Trajectories (90% Confidence Limits)

Device Precision and Expected Drift

To contextualize the observed trajectory discrepancies, this section reviews the intrinsic measurement tolerances of high-precision survey tools routinely used in exploration campaigns.

Survey tools exhibit inherent deviation due to mechanical, geological, and procedural factors. A generalized drift rate of 1.3 cm/m is acceptable, yielding an expected positional uncertainty of ±7.8 m at

600 m depth (Industry Steering Committee on Wellbore Survey Accuracy, 2017). This aligns with industry norms and supports the validity of surveyed data within reasonable tolerances.

12.5.3 Geological Context and Mitigating Factors

The mineralized volume extends approximately 3 × 3 km laterally and to a depth of 900 m, characteristic of a porphyry copper–gold system with disseminated mineralization. Key mitigating factors include:

- Surface mineralization and outcrops, supporting open-pit development.
- Mineralization is disseminated through the host rocks. There are no clear lithological or structural boundaries separating mineralized from barren zones.
- Expected deviation at 300 m depth (~14 m) aligns with block sizes of 15 × 15 m preserving spatial integrity for early-stage planning.
- Further exploration drilling and future mining operations will require infill drilling, inherently resolving survey gaps at the exploration stage and during short- and long-term mine planning.

12.6 Conclusion

While the survey database is not ideal, it reflects the inherited constraints of the Project and the regulatory environment. Through quantified trajectory modelling and geological context, the current dataset is acceptable for preliminary Mineral Resource assessment, though not recommended as a long-term standard.

The limitations are acknowledged transparently, and mitigation strategies, including future infill drilling, are embedded in the Project's development plan. This approach ensures that Mineral Resource modelling remains defensible, auditable, and aligned with professional standards.

12.7 Data Verification Summary

The data verification program independently assessed the reliability of all geological, analytical, and spatial information supporting the Hat Project Mineral Resource estimate. Work followed NI 43-101 standards and included a site visit, inspection of drilling and sampling procedures, assay verification, re-assays, and evaluation of collar coordinates and drill-hole trajectories.

Field observations confirmed that drilling, core recovery, logging, sampling, density determination, and data management follow industry best practices. Drill pads are surveyed using RTK-enabled drone imagery, and core handling is systematic, with secure facilities for logging, photography, sampling, and storage. Sampling uses half-core with full QA/QC traceability, and density is determined using the dry/wet method.

A 2024 re-assay program of 59 pulp samples showed strong reproducibility, with no material bias relative to original assays.

Assay verification compared 17,688 laboratory results with the database, identifying only ~160 minor discrepancies (0.9%), the identified discrepancies are associated with re-assays submitted under

secondary certification protocols as part of the active QA/QC program monitoring CRM performance. The review confirms 100% agreement for the five validated analytical variables between the laboratory-certified results and the assay grades reported by Doubleview. The assay database is therefore considered reliable.

Horizontal coordinate verification compared field-measured pad locations (converted from DMS to UTM) with the mean collar coordinates stored in the database. A tolerance of 15 m was adopted as an appropriate threshold, and all measured pads fell within this limit. This confirms that no material horizontal discrepancies exist and that the collar coordinates consistently fall within a narrow and reliable spatial range.

Drill-hole trajectory verification identified a significant limitation: approximately 36% of drill holes lack downhole survey measurements. Existing surveys, collected with a Gyro Shot non-north-seeking gyroscope, are of good quality, but the absence of trajectory data for over one-third of the holes reduces confidence in the spatial accuracy of deeper intercepts and may affect local geometry in the Mineral Resource model. This gap represents the most material technical issue identified during verification.

However, the mitigation measures proposed are appropriate and technically sound. These include implementing mandatory downhole surveying for all future drill holes and integrating improved survey QA/QC protocols into drilling contracts. Together, these actions will progressively strengthen the spatial reliability of the dataset and ensure that future drilling fully supports the accuracy of the Mineral Resource model.

In addition, the large areal extent of the deposit, its disseminated mineralization style, and the planned open-pit mining scenario collectively reduce the practical impact of missing survey records. Although positional uncertainty increases with depth, even at 300 m downhole the potential deviation remains within the 15 m block size used for Mineral Resource modelling. As the drilling program advances and mandatory surveying is applied consistently, the spatial fidelity of the 3-D dataset will continue to improve, further mitigating this limitation.

Overall, the verification work shows that field practices, analytical data, and collar coordinates are robust and well-documented, while the lack of downhole surveys for a substantial portion of drill holes remains the primary data limitation requiring future correction.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

Doubleview Gold Corporation requested that Mineit Inc. conduct a PEA-level study of the Hat Project, which proposes to mine and process 120,000 t/d during 350 d/a of porphyritic chalcopyrite and pyrite-bearing material which also contains potentially economically extractable quantities of gold, cobalt, and scandium.

Micon International Ltd (Micon) (2024, August) completed a Mineral Resource estimate (MRE) for the Hat Project. Scoping-level testwork results from Sepro Laboratories, Langley, B.C., were used as the basis for the anticipated process design parameters used in the MRE study. The results of the testwork program were reported in June 2021. Since the Micon report was issued, further metallurgical testing to recover scandium from the copper-gold-cobalt tailings was also completed, leading to a proposed process design for producing scandium oxide in addition to the anticipated conventional recovery of copper, gold, cobalt, and silver by flotation methods. Sepro undertook no comminution characterization tests.

13.1 Historical Test Program

Micon (2024, August) discussed the flotation testwork program, which included a gravity concentration test for gold recovery, and several rougher and cleaner flotation scoping tests to determine the feasibility of producing a high-gold-containing copper concentrate and a low-gold-content, high-cobalt pyrite concentrate. A portion of the flotation tailings would be treated for scandium recovery. A mineralogical examination of two flotation products was also performed.

The flotation strategy for copper concentrate production was designed to maximize gold recovery reporting to the copper concentrate, while concurrently optimizing cobalt recovery to the pyrite concentrate.

A flow diagram of the flotation test procedure Sepro adopted is discussed in Section 13.2.2.

13.1.1 Head Sample

The testwork was conducted on sample material selected and composited from the drill cores stored at the Project site, using the assay database to compile the appropriate sample for testing based on the information from the geological database. Table 13-1 lists Sepro's head-grade results for copper, cobalt, iron, total sulphur, gold, and silver.

Table 13-1: Metallurgical Sample Head Grades

Element	Unit	Head Grade
Copper	%	0.22
Cobalt	ppm	199
Iron	%	11.5
Sulphur, Total	%	2.40
Gold	g/t	0.37
Silver	g/t	2.30

13.1.2 Mineralogy

Early microscopic investigations had indicated that relatively coarse gold in association with pyrite was present in the test material. This finding led to a gravity concentration test being conducted ahead of the flotation testwork to ascertain whether gravity concentration could be utilized in the gold recovery. The results are presented in Section 13.2.1.

Mineralogical evaluations were also conducted on two flotation products to determine the respective liberation characteristics and to identify the gangue minerals associated with the valuable minerals. These results are discussed in Section 13.2.2 together with the outcomes obtained from the flotation tests.

The copper mineralization in the deposit is characterized mainly by the mineral chalcopyrite, which is associated with variably occurring pyrite, and with trace amounts of covellite, chalcocite, and bornite. The gangue minerals present include chlorite, epidote, tremolite, and calcite, with minor amounts of mica minerals, quartz, and other oxide minerals. The mineralogical analysis of two flotation products is presented in Section13.2.3.

Platinum group elements (PGE), mainly palladium, have been reported in low levels from drill-hole intercepts in the deposit. An analysis of a head sample and two flotation products was performed; it revealed that palladium occurs in the deposit and appears to be closely associated with the copper mineralization in localized areas. However, to date, assays for PGE and palladium values do not exceed 35 and 22 parts per billion (ppb) respectively. Two flotation products from the cleaner flotation test were analyzed to determine whether there was an indication that PGE upgrading had occurred, and the results indicate that limited upgrading had occurred. These results are also discussed in Section 13.2.4.

13.2 Metallurgical Tests

13.2.1 Gravity Concentration

Indications of the presence of relatively coarse gold in association with pyrite was noted in the test material, and this finding led to a gravity concentration test being conducted ahead of the flotation testwork to ascertain whether gravity concentration could be utilized in the recovery of gold. The results from the test showed that 10.9% of the gold present in the sample material could be recovered when

ground to the target grind particle size of 80% passing (P_{80}) 75 µm. Since this gold recovery value was relatively low, gravity concentration was not tested again.

13.2.2 Flotation Tests

A set of basic scoping-level flotation tests was performed in order to evaluate the response of the sample material to basic flotation conditions and to determine the feasibility of preparing two flotation products, namely a high-grade copper concentrate and a cobalt-bearing, pyrite-based concentrate. For each potential product, the tests consisted of producing a rougher flotation concentrate that was subjected to several cleaning (upgrading) stages. A test was also performed where the rougher concentrate was reground prior to the cleaning flotation stage. Typical copper mineral flotation reagents and conditions were chosen for the first flotation stage targeting copper recovery, while typical pyrite flotation reagents and conditions were selected for producing the second pyritic concentrate. These test conditions and reagents used are listed in Table 13-2, and the flowsheet followed is shown in Figure 13-1. The test material consisted of the tailings from the gravity concentration test as well as a parallel sample prepared for flotation testwork only. The grind and regrind particle sizes selected were typical of a copper-porphyry flotation recovery process. Similarly, the reagents used in the test program were those used typically for both the copper and pyrite recovery flotation process.

Table 13-2: Flotation Test Conditions

Test Conditions and Reagents	Unit	Description or Value
Concentrate Type		A copper-gold and a cobalt-pyrite concentrate
Flotation Circuit per Concentrate		Rougher stage, regrind, cleaner stage
Rougher Stage Grind Size (P_{80})	µm	75
Regrind Particle Size (P_{80})		
Copper-Gold Rougher Concentrate	µm	30 to 33 (target 30)
Cobalt-Pyrite Rougher Concentrate	µm	30 to 43 (target 30)
Reagents: Copper-Gold Flotation Stage		
Lime		Control slurry pH value between 10.5 and 11.0
Sodium Cyanide		Pyrite depression
SIPX		Collector reagent
A3477		Collector reagent
MIBC		Frother reagent
Reagents: Cobalt-Pyrite Flotation Stage		
Sulphuric Acid		Control slurry pH value to 6.6
Copper Sulphate		Pyrite activation
PAX		Collector reagent
MIBC		Frother reagent

Notes: A3477 = sodium dialkyl dithiophosphate; MIBC = methyl isobutyl carbinol; PAX = potassium amyl xanthate; SIPX = sodium isopropyl xanthate.

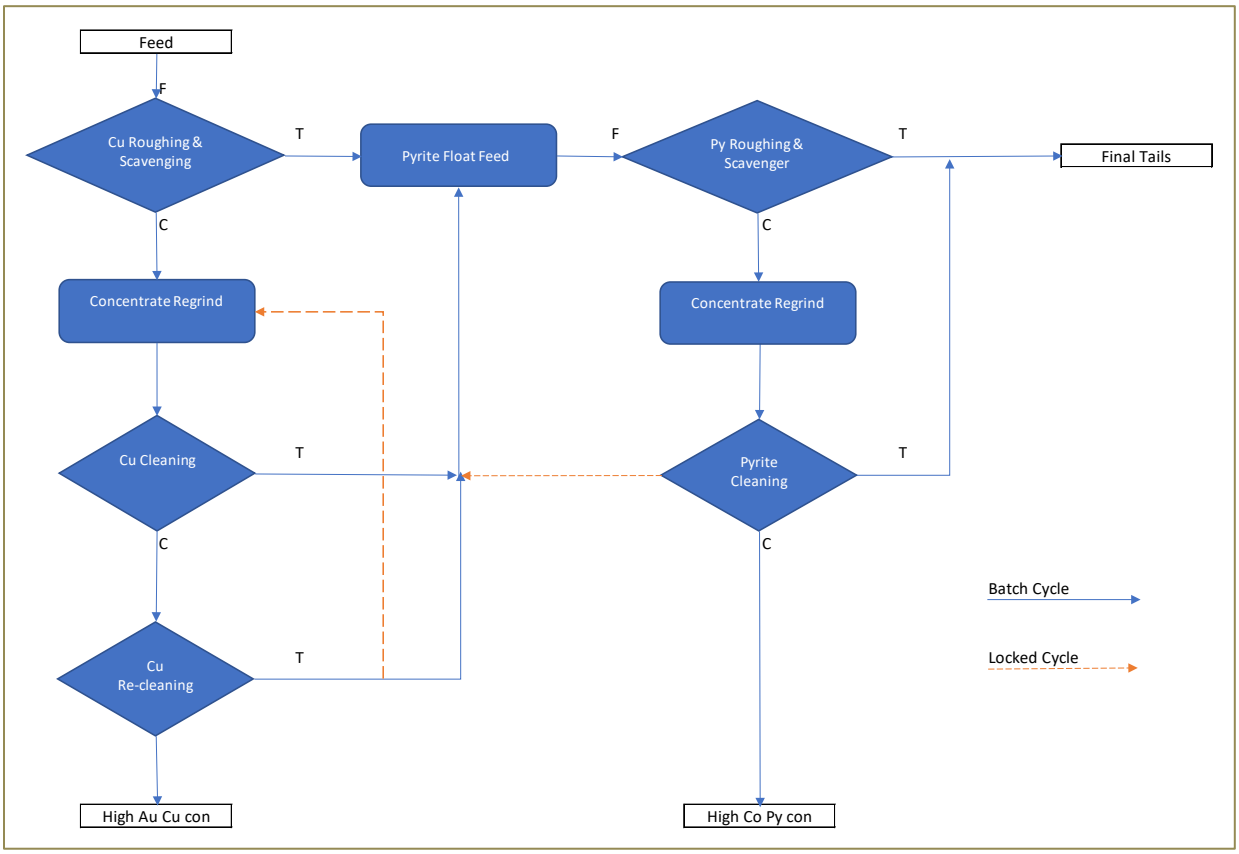


Figure 13-1: Flotation Test Flow Diagram

Although the flotation procedure for the two rougher and cleaner tests was the same, including the regrind stages, the results obtained differed somewhat. Table 13-3 presents the average recovery and concentrate grade values.

Table 13-3: Rougher and Cleaner Stage Flotation Results (Average Values)

Product	Grade			Recovery		
	Copper (%)	Cobalt (ppm)	Gold (g/t)	Copper (%)	Cobalt (%)	Gold (%)
Head Grade	0.21	132	0.34	-	-	-
Rougher Copper-Gold	1.07	200	1.39	78.7	22.7	59.0
Cleaner Copper-Gold	20.6	815	9.54	34.2	5.1	22.2
Rougher Cobalt-Pyrite	0.24	1,505	1.75	5.1	48.1	21.6
Cleaner Cobalt-Pyrite	0.14	2,575	2.57	0.4	21.3	8.1
Tailings	0.04	44	0.05	14.9	27.3	12.8

The results indicate that a saleable-grade copper concentrate and a high cobalt-pyrite concentrate can potentially be produced, since the cleaner test stage of one of the tests resulted in a maximum copper grade of 27.3%, while the cobalt-pyrite concentrate resulted in an average cobalt grade of 2,575 g/t from a feed grade of 132 g/t. Although the recovery values associated with the respective concentrates are not high, the flotation test procedure was not optimized with respect to grind size and reagent additions. Nevertheless, the tests served to demonstrate that producing a saleable high-grade copper concentrate was achievable, and that upgrading the cobalt into a pyrite concentrate that can be used for cobalt recovery purposes was possible. Based on the final tailings assay values, the following recovery values for both concentrates appear to be attainable, subject to the various factors listed in Section 26. Note that these assumptions cannot be proven without a more-detailed testwork program being performed. Table 13-4 gives the potentially attainable concentrate grade and recovery values.

Table 13-4: Indicated Attainable Grade and Recovery Values

Product	Grade				Recovery			
	Copper (%)	Cobalt (ppm)	Gold (g/t)	Silver (g/t)	Copper (%)	Cobalt (%)	Gold (%)	Silver (%)
Head Grade	0.21	132	0.34	2.9	-	-	-	-
Copper-Gold Concentrate	25	1,160	12	68	80	30	66	53
Cobalt-Pyrite Concentrate	0.30	1,605	2	8	5	48	23	15
Combined Concentrates	-	-	-	-	85	78	89	68
Tailings	0.05	40	0.05	1.0	15	22	11	32

13.2.3 Mineralogical Evaluation

To characterize the nature of the copper and cobalt losses, a mineralogical analysis was performed as a quantitative evaluation of materials by scanning electron microscope (QEMSCAN) analysis and a specific mineral search for cobalt minerals on two flotation tailings samples. The two samples selected for the investigation were the cobalt-pyrite rougher-scavenger tailings and the copper-gold first cleaner tailings samples. The results obtained indicated that both samples consisted of predominantly the following gangue minerals, namely amphibole/pyroxene, chlorite feldspars, and lesser amounts of carbonates, iron oxides, quartz, and epidote, while the copper-bearing sulphide minerals identified included chalcopyrite, with trace amounts of bornite, chalcocite, and covellite. Cobalt was found to be associated within cobalt sulphide minerals (which were not identified), and within the lattice structure of pyrite presumably substituting for iron within the pyrite mineral structure, It was determined that a significant proportion of the cobalt minerals, namely 22% and 42%, respectively, were liberated particles finer than 14 µm. Copper losses in both samples were primarily in the form of coarse binary products with non-sulphide mineral particles, while significant losses of copper occurred with ultrafine liberated copper sulphide particles, namely 42% and 53%, respectively. These aspects should be investigated further in the future test programs.

13.2.4 Platinum Group Elements Investigation

Very-low levels of PGEs, mainly palladium, were reported to be present in the Hat Deposit. Since this could be a potential source of credit for the copper concentrate, an analysis of the head sample and two flotation products from the rougher-cleaner test was performed to determine whether notable upgrading of any of the constituent elements of the PGE had occurred. The two samples analyzed were the copper-gold cleaner concentrate and the final tailings, as it was anticipated the PGE would be associated and recoverable with the copper minerals (Table 13-5).

Table 13-5: PGE Assay Data

Details	Os (ppb)	Ir (ppb)	Ru (ppb)	Rh (ppb)	Pt (ppb)	Pd (ppb)
Head Sample	<2	0.6	5	0.8	10	12
Copper-Gold Cleaner Concentrate	<2	3.0	< 5	3.9	6	22
Final Tailings	<2	1.9	<5	3.2	6	9

Notes: Key: Os = Osmium, ppb = parts per billion, Ir = Iridium, Ru = Ruthenium, Rh = Rhodium, Pt = Platinum, Pd = Palladium

The results indicate that some upgrading had in fact occurred for palladium, while iridium and rhodium also showed upgrading, although the final tailings grade value appears to indicate a contradictory responsive behaviour. Ruthenium and osmium also appear to be non-responsive or below the detection limit for assaying. When assessing the final tailings value, the results for platinum appear to show a behaviour similar to that of palladium, and that a small degree of upgrading had also occurred, but not with respect to the copper concentrate. Since it is not known whether the PGE occurs as discrete minerals or combinations of substitution elements in a host mineral, and since the total amount of PGE in the concentrate sample does not exceed 30 to 35 ppb, it is recommended that additional testwork be conducted before any specific conclusions are drawn regarding the recovery of PGE.

13.2.5 Head-Grade Values

The head-grade values reported during the metallurgical investigation were found to vary, with cobalt and silver showing the greatest variation (Table 13-6). For purposes of this Technical Report and ease of comparison, the accepted head-grade values as shown in Table 13-1 are also given in Table 13-6.

Table 13-6: Variation in Head-Grade Values

Origin of Analysis	Copper (%)	Cobalt (ppm)	Gold (g/t)	Silver (g/t)
Head Grade	0.22	199	-	-
ICP Analysis	0.20	137	0.37	2.90
Head Grade from Rougher Flotation Test	0.21	132	0.34	-
Head Grade from Rougher-Cleaner Test	0.20	136	0.37	2.30
Head Grade as given in Table 13-1	0.22	199	0.37	2.30

13.3 Hydrometallurgical Testwork

13.3.1 Scandium

The initial head analyses and testwork showed that geologic materials derived from the Hat project contained scandium at levels elevated above those normally associated with crustal rocks. As scandium is designated a critical mineral by governments worldwide and is of increasing strategic and economic interest, it was decided to evaluate the potential for scandium extraction and recovery as part of the Hat project development program. As testwork showed that scandium reported to flotation tailings, a preliminary testwork program was implemented in October 2023, which was focussed on the treatment of flotation tailings for the recovery of scandium into an intermediate product suitable as feedstock for a scandium recovery process.

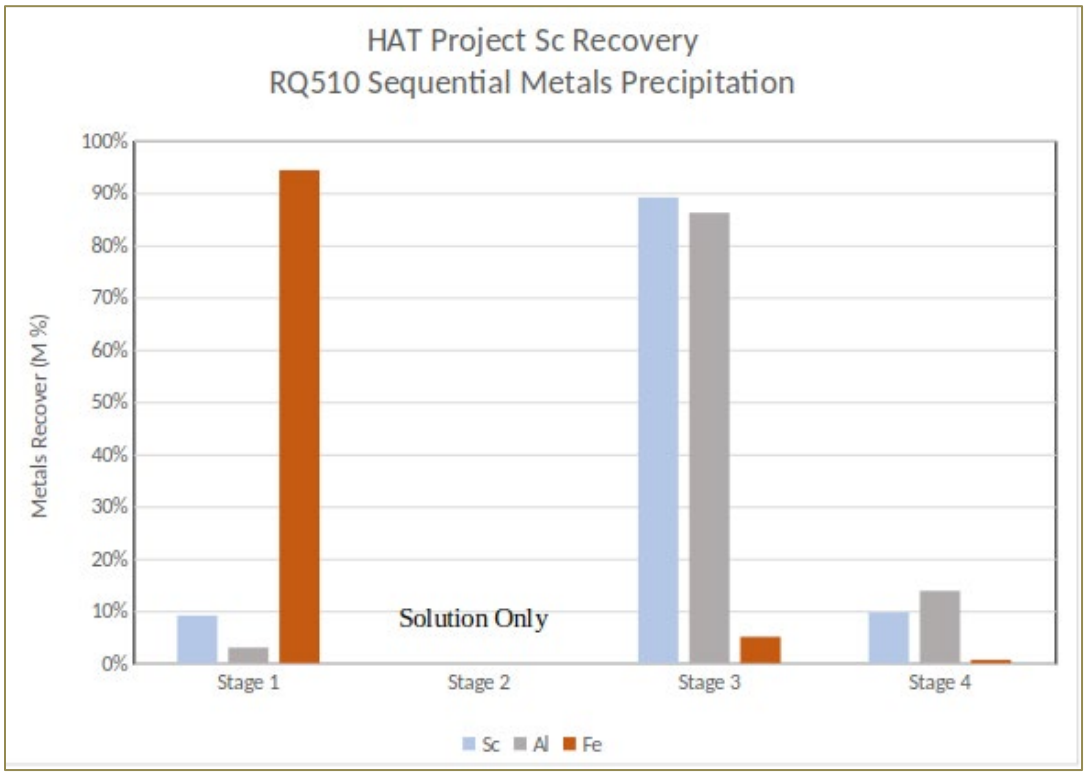
Scandium was recovered from flotation tailings material by extraction in sulphate media at elevated temperatures using a leach aid, and was followed by removing metals by stage-wise precipitation as hydroxides or phosphates. Results for test RQ510 are shown in Table 13-7.

Table 13-7: Hat Preliminary Scandium Recovery Data

Element	Head Grade (ppm)	Recovery (%)				Metal Balance (%)
		Stage 1	Stage 2	Stage 3	Stage 4	
Scandium	10.30	9.10	0.00	89.15	9.71	107.96
Aluminum	10,800	2.97	0.00	86.19	13.81	102.97
Iron	24,750	94.34	0.00	5.02	0.63	100

The testwork demonstrated that 94% of the iron could be rejected as a primary precipitate product and that scandium and aluminium could be recovered into a secondary phosphate precipitate. Total scandium recovery into secondary products was greater than 90%. Metals accountability was generally good with 108% Sc, 103% Al, and 100% Fe. Figure 13-2 shows the distribution of the primary metals to the various products.

At the time it was thought that scandium could be recovered to a secondary precipitate, which would be treated to recover scandium to a final product such as an oxide; the precipitate would be redissolved in acid media—typically sulphate or chloride. The scandium could then be separated either by ion exchange (IX) or solvent extraction (SX). The scandium would then be recovered from the strip solution of IX or SX by precipitation as a salt such as a hydroxide or carbonate. The hydroxide or carbonate would then be converted to the oxide by calcination.



Note: M% = metal percent.

Figure 13-2: Preliminary Scandium Extraction and Precipitation Data



Figure 13-3: Scandium Separation Precipitates

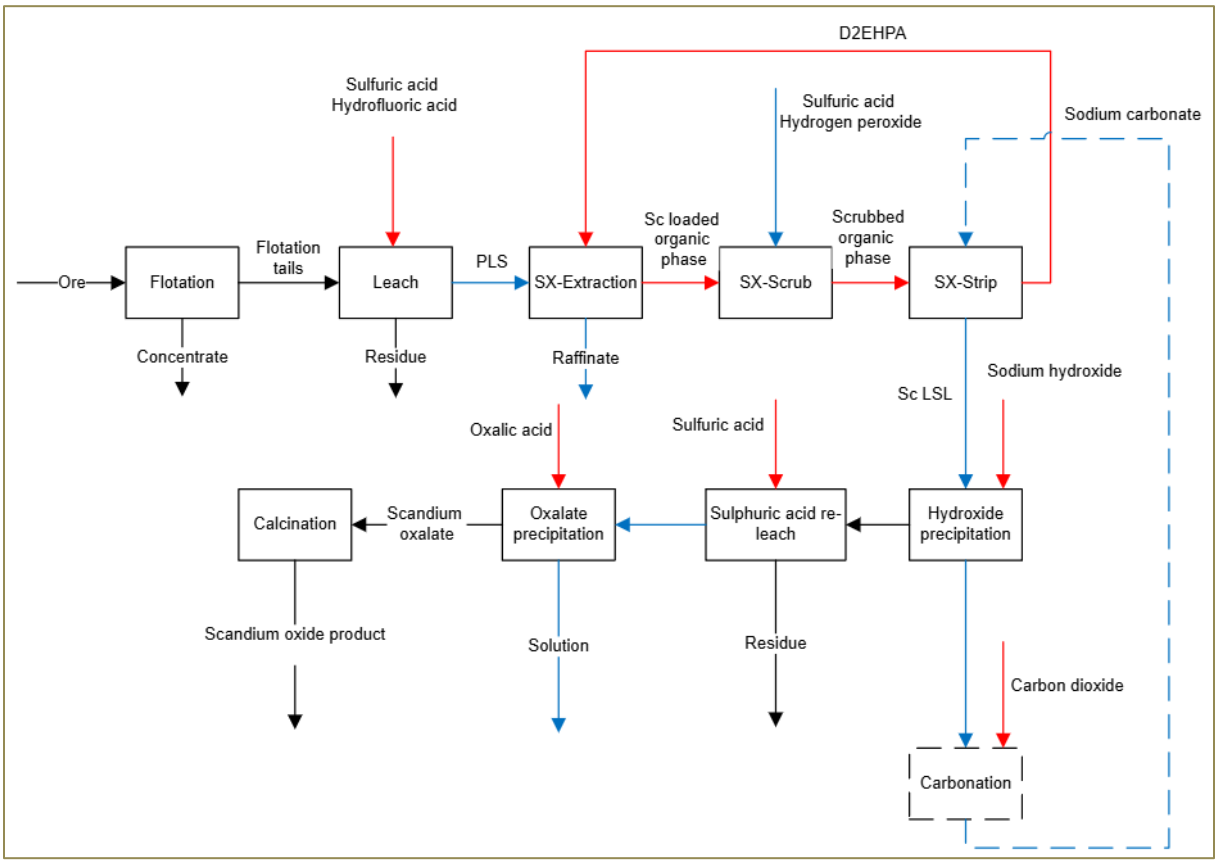
13.3.2 Large Scale Scandium Extraction Testwork

As a result of the successful demonstration of the extractability of scandium from flotation tailings it was decided to implement a scandium recovery testwork program in which the recovery of scandium to a final scandium trioxide product would be attempted. Owing to the low grade of scandium in ore and tailings (~30 g/t) a large-scale test would be required to produce a meaningful quantity of final product for assay and evaluation.

A large-scale testwork program (309 kg) was implemented at SGS Lakefield (SGS) in January 2024. This program comprised:

- Generating flotation tailings
  - Grind calibration
  - Bulk flotation
  - Filtration and dewatering of tailings
  - Repulping, washing, and drying of tailings material.
- Leaching testwork
  - Enhanced leaching tests
  - Acid bake tests
  - Bulk leaching of tailings
  - Selective precipitation.
- Liquor treatment
  - Acid neutralization tests
  - SX/IX shake-out tests
  - Bulk neutralization
  - SX/IX scouting tests
  - Bulk SX/IX testing
  - Batch scrubbing and stripping
  - Scandium precipitation
  - Scandium refining.

This testwork program was designed to confirm flotation performance, optimize leaching conditions, and develop downstream hydrometallurgical processes for scandium extraction, recovery, and refining. The overall testwork scheme is shown in Figure 13-4.



Source: SGS (2025).

Figure 13-4: Overall Laboratory Testwork Scheme

The following sections summarize the testwork. Full details are contained in SGS (2025, December 17), which is available upon request.

Flotation Testwork

Flotation testwork was conducted to generate sufficient flotation tailings for the scandium recovery testwork. Seventy-two samples totalling 309 kg were selected to provide a bulk composite for laboratory treatment, head composition is shown in Table 13-8. On testing, the bulk composite showed a significantly lower copper head grade compared to the earlier materials tested (518 vs. 2,100 g/t), while the scandium content was similar (48 vs. 43 g/t). The confirmatory rougher flotation tests under the previously established conditions achieved 82.1% Cu recovery (RF1), which was lower than the reference test (94.9%) and was likely owing to the lower copper head grade. Scandium distribution to rougher tailings was 97.5%, higher than the initial reference test (92.9%), confirming that the flotation conditions effectively reject scandium to tailings for subsequent recovery. Twenty-one bulk flotation tests were completed under RF1 conditions to generate rougher tailings for scandium recovery. Quality control checks on every fifth test showed consistent scandium distribution to tailings ranged from 95.1% to 97.8%, indicating stable flotation performance.

Table 13-8: Bulk Head-Analysis

Bulk Head-Composition						
SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO	Na_2O	K_2O
44.9%	12.2%	13.6%	9.44%	10.4%	1.7%	1.44%
TiO_2	P_2O_5	MnO	Cr_2O_3	V_2O_5	LOI	Sum
0.72%	0.16%	0.18%	0.06%	0.06%	4.16%	99.1%
Ag	As	Ba	Be	Bi	Cd	Co
<200 g/t	<1200 g/t	284 g/t	<3 g/t	<400 g/t	<40 g/t	<200 g/t
Cu	Li	Mo	Ni	Pb	Sb	Se
518 g/t	<800 g/t	<300 g/t	<300 g/t	<800 g/t	<400 g/t	<2,000 g/t
Si	Sn	Sr	Tl	U	Y	Zn
228,000 g/t	<800 g/t	204 g/t	<2,000 g/t	<400 g/t	15 g/t	<300 g/t
Sc	Au	Pt	Pd	Rh	Ru	Ir
48 g/t	0.1 g/t	<0.02 g/t	<0.02 g/t	<0.02 g/t	<0.05 g/t	<0.02 g/t

Leach Testwork

The flotation tailings used as feed material contained high levels of impurities (Fe, Si, Ca, Mg, Al), and a relatively low scandium grade of 48 g/t, which influenced leaching performance. Scoping-level hydrofluoric acid-enhanced atmospheric leaching achieved scandium extractions ranging from 62% to 82%, with higher acid dosages and extended leaching times improving the recovery of scandium. Leach tests No.5 (L5) demonstrated that the highest scandium extraction (82%) was achieved under conditions of 20% solids, initial feed acidity of 250 g/L sulphuric acid, and an 8-hour leach retention time. Under these conditions, the acid additions were 1,050 kg/t H_2SO_4 and 122 kg/t HF. Replacement of hydrogen fluoride with sodium fluoride (L3) resulted in a scandium extraction of 64%. Increasing solids content to 40% (L4) caused slurry gel formation and precipitation issues, which reduced scandium extraction to 62%.

Acid bake–water leach tests yielded poor scandium recovery (10%–18%), even with increased acid dosage and retention time, indicating this route was not viable for treating the flotation tailings.

Ten bulk-leaching tests were conducted under the optimum conditions identified in the bench-scale tests to produce a leach solution (~260 L) for subsequent scandium solvent extraction in bench-scale and pilot-plant tests.

Table 13-9: Bulk Leach Test Conditions

Test ID	Feed	Leachant	Sulphuric Acid Addition (kg/t)	Hydrofluoric Acid Addition (kg/t)	Target Feed Sulphuric Acid Concentration (g/L)	Target Feed Hydrofluoric Acid Concentration (g/L)	Solids (%)	Temp. (°C)	Retention Time (h)
BL1-BL10	Washed Dried Flotation Tailings	H_2SO_4 (96%), HF (50%)	1,053	124	250	15.00	20	95	5

Table 13-10: Bulk Extraction Test Results

Test ID	Leaching (%)										
	BL1	BL2	BL3	BL4	BL5	BL6	BL7	BL8	BL9	BL10	Average
Scandium	85	75	89	85	83	87	89	84	87	87	85
Titanium	83	80	86	84	75	82	86	76	80	81	81
Iron	92	92	94	92	89	93	95	90	91	91	92
Aluminum	64	64	73	64	44	65	78	47	54	62	62
Magnesium	87	88	91	87	82	88	93	82	85	87	87
Chromium	67	74	87	82	76	78	87	68	69	82	77
Magnesium	85	80	89	89	86	91	94	87	87	89	88
Iron	83	69	79	82	76	83	79	77	76	80	78



Figure 13-5: Bulk Leach Test Apparatus

Purification and Selective Precipitation

Attempts to remove impurities over a range of pH values resulted in significant scandium co-precipitation, indicating that single-stage pH adjustment was not sufficiently selective for iron and aluminum, and cannot achieve impurity removal without significant scandium losses. Operational challenges, including gelatinous precipitate formation and poor filtration at elevated pH, highlighted significant solid/liquid (S/L) separation and material handling difficulties making this option impractical.



Figure 13-6: Gelatinous Precipitate Formed During Selective Precipitation

Ion Exchange

Selective scandium ion exchange using Puromet MtS9500 was investigated but achieved only limited scandium extraction (maximum 12% at 60 g/L acidity), indicating that the high acid concentration in the feed solution negatively affected IX performance.

Solvent Extraction

Selective solvent extraction of scandium was considered on straight leach and partially neutralized leach liquors. Several extractants were investigated. The work focused on producing a concentrated scandium stream and only limited optimization testwork was carried out. Among three extractants tested (D2EHPA, Cyanex 572, Mextral 507P), D2EHPA demonstrated superior performance, achieving >94% scandium extraction across all acidities tested without prior neutralization. Cyanex 572 and Mextral 507P showed poor scandium extraction (≤14% and ~2%, respectively) and would require significantly lower acidity to be effective, making them less practical in this application. D2EHPA exhibited strong selectivity for scandium over iron (extraction ~1.5%–2.5% Fe) but only moderate selectivity over titanium (~40% Ti co-extraction). It was anticipated that scrubbing of the loaded organic would be able to reject co-extracted titanium. Maximum loading tests Scandium extraction test No. 4(ScE4) confirmed that scandium concentration in the organic phase increased from 29 to 246 mg/L Sc after 13 contacts, establishing a basis for circuit design. McCabe–Thiele modelling indicated that six extraction stages at an Organic phase to Aqueous phase (O/A) ratio of 0.08 were required to achieve less than 1.5 mg/L Sc in the raffinate.

A mini-solvent extraction pilot plant (employing only the extraction stages) was designed and constructed, and successfully operated for 63 hours, processing 234 L of pregnant leach solution and approximately 13 L of loaded organic. The circuit maintained stable operation at 48°C–55°C without pH control,

simplifying process requirements. The pilot plant configuration validated the design criteria and demonstrated effective scandium extraction (94% Sc) from pregnant leach solution containing high impurity levels (Fe, Ti, Al, Mg). The loaded organic phase achieved a scandium concentration of approximately 200 mg/L Sc, consistent with the McCabe–Thiele predictions for six stages. Increasing the O/A phase ratio improved scandium recovery but reduced scandium concentration in the loaded organic phase, indicating a likely trade-off between recovery and loading.

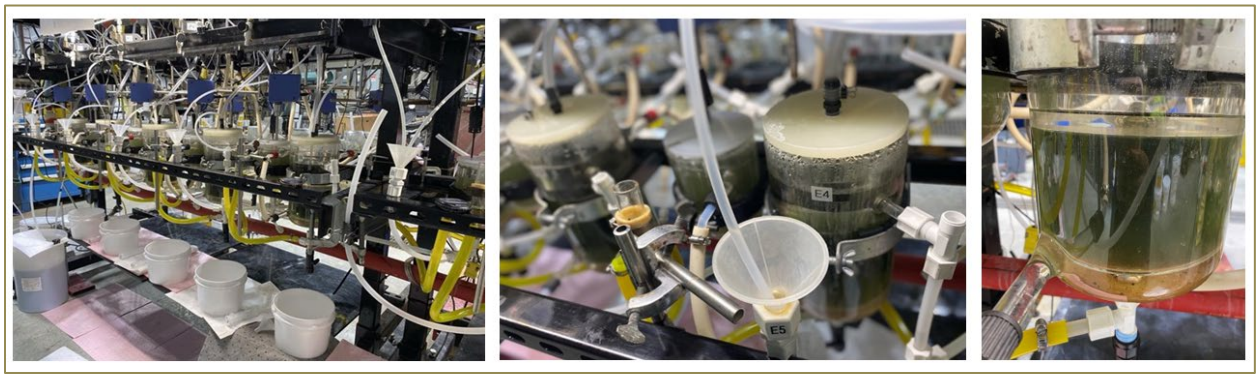


Figure 13-7: Mini Solvent Extraction Test Apparatus

Impurities such as titanium, molybdenum, and iron were co-extracted, highlighting the need for effective scrubbing steps. The data show that iron was crowded off at elevated scandium loading.

Metallurgical accounting showed accountabilities >94% for major metals (Sc, Ti, Fe, Al, Mg), confirming high data quality and reliable material tracking. Both oxidative acid (hydrogen peroxide + sulphuric acid) and complexing acid (oxalic acid + sulphuric acid) scrub systems exhibited excellent scandium selectivity with negligible losses (<0.1 mg/L Sc in spent scrub liquor) and effective iron removal. Oxidative acid scrubbing achieved approximately 67% Fe removal, whilst complexing acid scrubbing achieved ~98% Fe removal. Sequential scrubbing using the oxidative system (H_2SO_4 + H_2O_2) significantly reduced iron and titanium levels in the organic phase (from 244 mg/L Fe and 3,486 mg/L Ti to 114 mg/L Fe and 1,740 mg/L Ti, respectively) without affecting scandium, making it suitable for bulk scrubbing. Bulk scrubbing reduced iron from 217 to 14 mg/L, titanium from 677 to 7 mg/L, and molybdenum was below detection limits, confirming the robustness of the oxidative scrub system. Sodium carbonate (200 g/L) proved highly effective for scandium stripping from scrubbed organic across all tested conditions, with optimal performance at 80°C and an O/A ratio of 1:1, achieving 91%–100% efficiency. The process exhibited excellent phase behaviour, with no precipitate or crud formation, and the final strip liquor contained 224–225 mg/L Sc alongside impurities (Fe, Ca, Mg, P, Ti), indicating the need for a subsequent refining step.

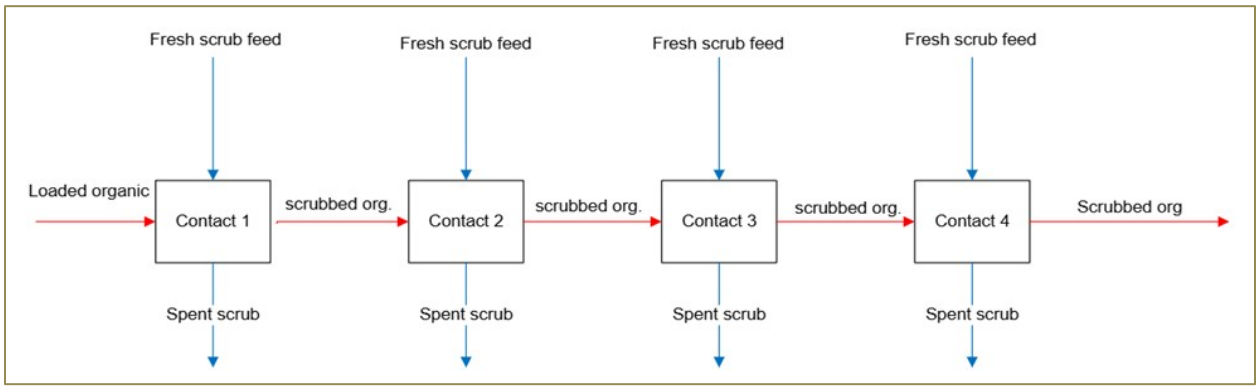


Figure 13-8: Sequential Scrubbing Circuit

Scandium Hydroxide Precipitation

Optimized conditions ($NaCO_3$ mole ratio of 2.6, 65°C, 3 hours) achieved ~99 % scandium precipitation from solution. Filtration challenges were observed owing to crystal formation, and filtration should be performed hot to prevent crystallization and accelerate processing. Iron and titanium largely co-precipitated (>98 % recovery to solids), indicating some impurity carry-over into the hydroxide stage.

Sulphuric Acid Re-dissolution

Redissolution in sulphuric acid was highly effective, achieving ~98 % mass recovery and producing a solution with 6.9 g/L Sc. Acid consumption was 1,290 kg/t of solids.

Oxalic Acid Precipitation

Oxalic acid precipitation achieved 96% scandium recovery and effective impurity rejection (95% Fe and 90% Ti removal).



Figure 13-9: Scandium Oxalate Precipitate

Calcination

The scandium concentration was ~63%, and the scandium oxide product was ~96% scandium oxide. A detailed composition of the scandium oxide product is found in Table 13-11. Zirconium (2.4%) was the primary impurity, which had not been observed until this point in the recovery procedure, suggesting a need for targeted zirconium removal before calcination.



Figure 13-10: Final Di-Scandium Trioxide Product

Table 13-11: Initial Di-Scandium Trioxide Composition

Element	Composition (%)
Sc	62.85
Al	<0.01
Fe	0.05
Mg	0.01
Ca	0.31
Na	0.03
K	<0.01
Ti	0.1
P	<0.002
Mn	<0.01
Cr	<0.01
V	<0.01
S	0.18
C(t)	0.05
Zr	2.4
Lost on Ignition (LOI)	0.46

14 MINERAL RESOURCE ESTIMATE

Tomasz Wawruch, Mineit's Qualified Person prepared the Mineral Resource estimate presented in this PEA. All figures and tables presented in this section were prepared by Mr. Wawruch, except where an alternative source is explicitly indicated. This estimate follows Micon's Mineral Resource estimate with an effective date of July 17, 2024 (Micon, 2024). This Technical Report incorporates significant new data from the 2024 and 2025 exploration campaigns, and this Mineral Resource estimate with an effective date of February 4, 2026, supersedes the 2024 Micon estimate.

The Mineral Resource estimate presented in this PEA was prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum's *CIM Definition Standards for Mineral Resources & Mineral Reserves* (CIM Definition Standards) (CIM, 2014), as incorporated by reference in NI 43-101. The results of the 2025 Mineral Resource estimate are shown in Table 14-1. Copper, gold, silver, cobalt, and scandium are reported within the optimized pit shell.

Table 14-1: Mineral Resources of the Hat Project Tabulated at 0.20% CuEq as of February 4, 2026

Mineral Resource Classification	Tonnage (Mt)	Average Grade					Metal Content				
		CuEq (%)	Cu (%)	Au (g/t)	Co (g/t)	Ag (g/t)	CuEq (Blb)	Cu (Blb)	Au (Moz)	Co (Mlb)	Ag (Moz)
Measured	272	0.44	0.22	0.18	76.26	0.37	2.61	1.11	1.41	35.6	2.17
Indicated	337	0.43	0.21	0.19	76.81	0.39	3.21	1.31	1.81	44.5	2.88
Total M+I	609	0.43	0.21	0.18	76.57	0.38	5.82	2.42	3.22	80.1	5.05
Inferred	503	0.41	0.18	0.19	76.62	0.38	4.57	1.72	2.77	66.2	4.19

Table 14-2: Scandium Oxide Mineral Resources of the Hat Project Tabulated at 0.20% CuEq as of February 4, 2026

Mineral Resource Classification	Tonnage (Mt)	Sc Tonnage1 (Mt)	Average Grade Sc (g/t)	Metal Content Sc2O3 2 (t)
Measured	272	34	28.79	1,081
Indicated	337	42	28.76	1,334
Total M+I	609	76	28.77	2,415
Inferred	503	63	28.69	1,996

Notes: [1]Scandium tonnages represent 12.5% of the mineralized material by category, reflecting the proportion of tailings expected to be processed through a dedicated scandium leach circuit under current metallurgical design constraints (Section 17.7.7).
[2] Scandium oxide metal content have been calculated using the metallurgical recovery of 72% and conversion factor from Sc to Sc_2O_3 of 1.534.

- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
- Inferred Mineral Resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that any part of the Inferred Mineral Resource will be converted to a higher category.
- The effective date of the Mineral Resource estimate is February 4, 2026.
- Metal contents for Cu, Au, Co, and Ag have been calculated using the following metallurgical recovery factors: Cu = 85%, Au = 89%, Co = 78%, and Ag = 68%

- Mineral Resources are reported within optimized open pit constraints and 0.2% CuEq cut-off grade, based on a $7.93/t milled processing cost and C$2.9/t milled general and administrative (G&A) cost, with a mining cost of C$3.01/t + incremental mining cost increasing by C$0.015/t for every bench below the reference level of 1,125 mRL.
- CuEq calculations do not include scandium. The formula used to calculate CuEq is:
  CuEq = [(((Ag × 46.0 × 0.68)/31.1035) + ((Au × 3200 × 0.89)/31.1035) + 0.0001 × (Co × 20.0 × 0.78 × 22.0462) + 0.0001 × (Cu × 4.8 × 22.0462 × 0.85))/(4.8 × 22.0462 × 0.85)] where all input variables are expressed in (ppm), and CuEq is expressed in percent (%).
- Rounding may result in minor variations between individual values and totals; such differences are not considered material to the Mineral Resource estimate.
- Mineral Resource classification reflects the level of geological confidence and satisfies the uncertainty criteria appropriate for exploration and resource development. Additional drilling will be required to reduce uncertainty to the level expected for production planning.
- The Mineral Resource estimate reflects the geological interpretation, drill-hole spacing, and estimation parameters available at the time of modelling. Any additional drilling is expected to influence the current outcome by improving confidence in the estimates and refining the geometry of the mineralized domains.

14.1 Drill-Hole Database

The cut off date for the database is November 9, 2025. Drilling completed at the Hat Project up to September 30, 2025, comprises 101 drill holes, totaling 49,886 m of diamond core. Holes 10, 44, 48, and 84 were never submitted for assay and were therefore excluded from the Mineral Resource estimation process.

The validated database therefore consists of 97 drill holes totaling 49,548 m. Although drilling was completed on September 30, 2025, additional time was required to receive the laboratory results. The drill hole database was finalized in November once all available assays had been incorporated. Figure 14-1 and Figure 14-2 show the drilling completed by September 2025, while the database cut-off reflects the later availability of assay data. The target variables in the database are gold, silver, copper, cobalt and scandium. Tonnage is addressed using values from the density (SG) database.

For geological modelling support, the database consists of descriptive data in which seven principal lithologies are recognized, grouped into a volcanic sequence represented by andesite, trachyte, and tuff, and an intrusive suite composed of diorite, gabbro, pyroxenite, and monzonite.

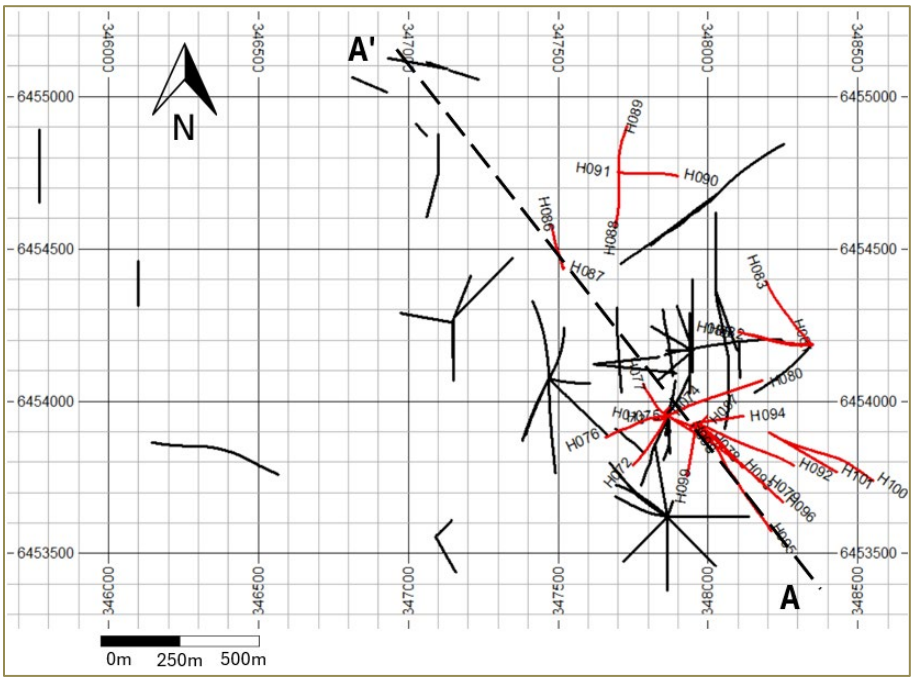


Figure 14-1: Plan View of Drill-Hole Locations Highlighting the Expansion of the Dataset, with Historical (Pre-July 2024) Drilling Shown in Black and More Recent Drilling in Red (UTM Zone 9N NAD 83)

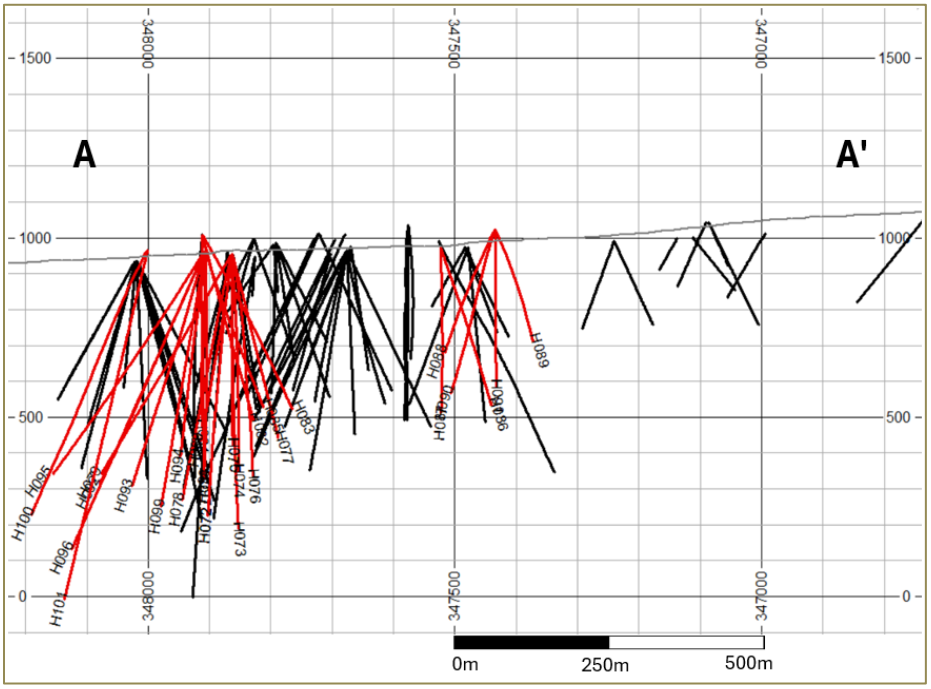


Figure 14-2: Cross-Section A–A´ (See Figure 14-1) Showing Historical Drilling (pre-July 2024) in Black and Drilling Completed Since July 2024 in Red (UTM Zone 9N, NAD 83)

14.2 Geological Interpretation and Modelling

All geological modelling and resource estimation was completed using Datamine Studio RM v3.21.8570.0. and GSLIB applications. The Mineral Resource estimate for this PEA incorporates an expanded drilling database and an improved understanding of the alkalic porphyry system. It follows a geological-genetic approach that integrates lithology, alteration, structure, and overburden to define the key domains controlling gold–copper mineralization. The model distinguishes the main mineralized intrusive stock—comprising dioritic to gabbroic phases—from the surrounding Upper Triassic volcanic and sedimentary host rocks (Sections 7.1 and 7.2), represented by tuffaceous meta-andesites.

Alteration and structural domains, including the potassic core, propylitic halo, and major fault zones, were identified as the principal geological features associated with mineralization continuity and grade distribution. Together, these domains provide the conceptual framework for understanding the mineralized system and for supporting the interpretation of the numerical models.

Although the spatial relationships governing the gold–copper system are well established at the Project's development stage, they cannot be used to constrain grade interpolation. Their qualitative nature, and the absence of a quantitative database, prevent their direct application as estimation domains. In addition, each variable exhibits its own spatial characteristics; imposing qualitative constraints on them would not improve—and could potentially degrade—the quality of the estimation.

The inherent relationship between the intrusive complex hosting the mineralized multi-element system and the volcanic rock units surrounding the intrusion is presented in Figure 14-3.

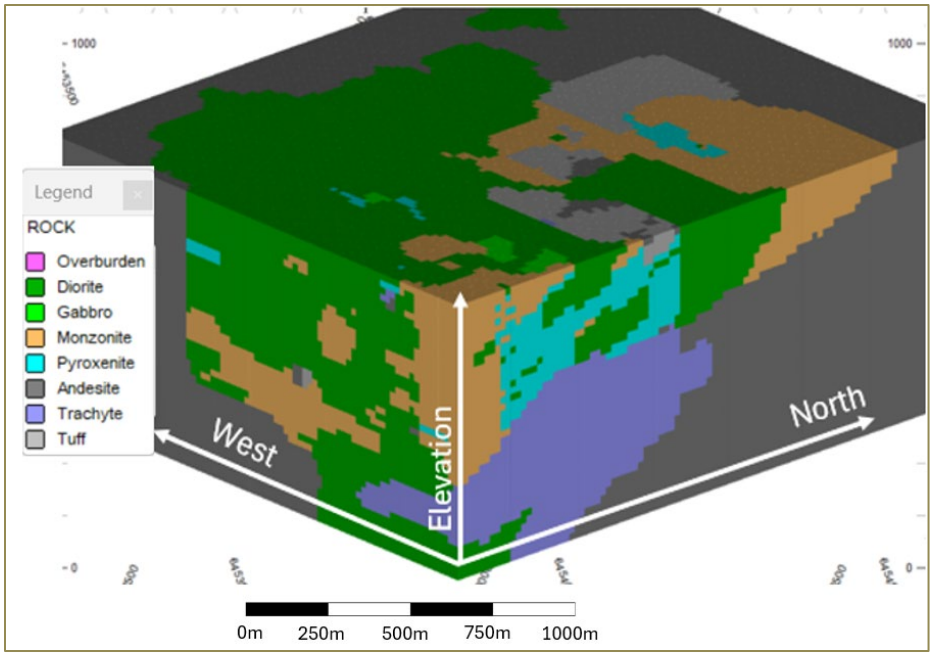


Figure 14-3: Isometric View of the Geological Block Model, Projected onto Three Orthogonal Planes, Showing Intrusive Units Within the Volcanic Host Sequence

14.2.1 Geological Probabilistic Modelling

The geological model incorporates seven lithological units in two suites—intrusive and volcanic (Table 14-3).

Table 14-3: Domain Grouping for Lithological Units

Intrusive	Volcanic
• Diorite • Gabbro • Monzonite • Pyroxenite	• Andesite • Trachyte • Tuffs

To construct a coherent 3-D lithological model, a probabilistic approach based on sequential indicator interpolation (SII) was applied. In this workflow, indicator variables are interpolated using ordinary kriging, producing spatial probability fields for each lithological category. This method was selected instead of traditional wireframe construction because it provides a quantitative, reproducible, and geostatistically defensible representation of geological uncertainty. The QP prefers this approach, as it allows decisions to be based on explicit probabilities rather than deterministic boundaries.

The lithology modelling follows a three-step sequential process (Figure 14-4 to Figure 14-6).

Step 1—Intrusive vs. Volcanic Probability

The first indicator model estimates the probability of each block belonging to either the intrusive or volcanic domain. Ordinary kriging of the binary indicator variable (intrusive = 1, volcanic = 0) produces a continuous probability field that honours drill-hole data and the spatial continuity defined by the indicator variogram.

Step 2—Volcanic Unit Probabilities

Within the volcanic domain, a second set of indicator interpolations is performed for andesite, trachyte, and tuffs. Each volcanic unit is coded as an indicator variable and interpolated using ordinary kriging. Lithology with the highest estimated probability is assigned to each block within the volcanic domain.

Step 3—Intrusive Unit Probabilities

A parallel set of indicator interpolations is carried out within the intrusive domain for diorite, gabbro, monzonite, and pyroxenite. Again, ordinary kriging is used to estimate the probability of each intrusive unit, and the lithology with the highest probability is assigned to each block.

This sequential probabilistic workflow results in a fully 3-D lithological model that honors conditional data, preserves realistic spatial relationships, and avoids the geometric biases inherent in hand-built wireframes. The resulting lithological distribution is consistent with geological interpretation and provides a robust foundation for subsequent grade estimation.

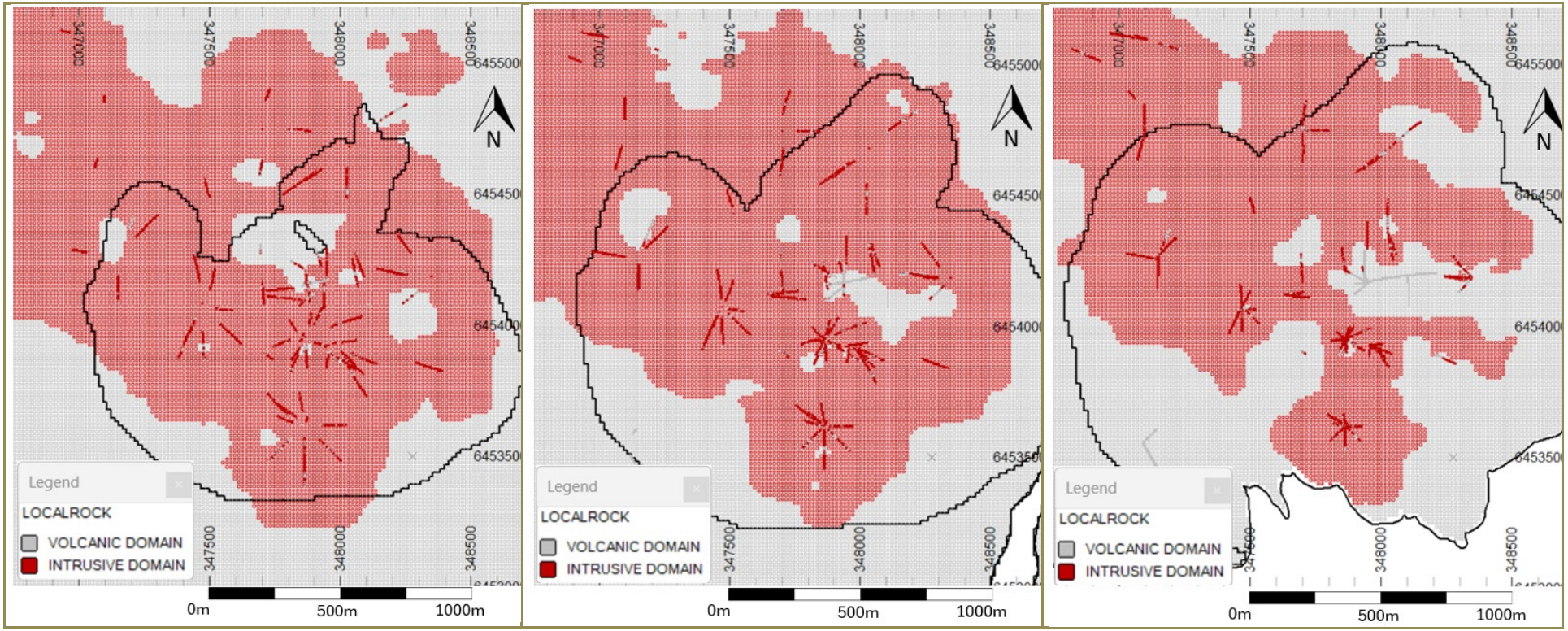
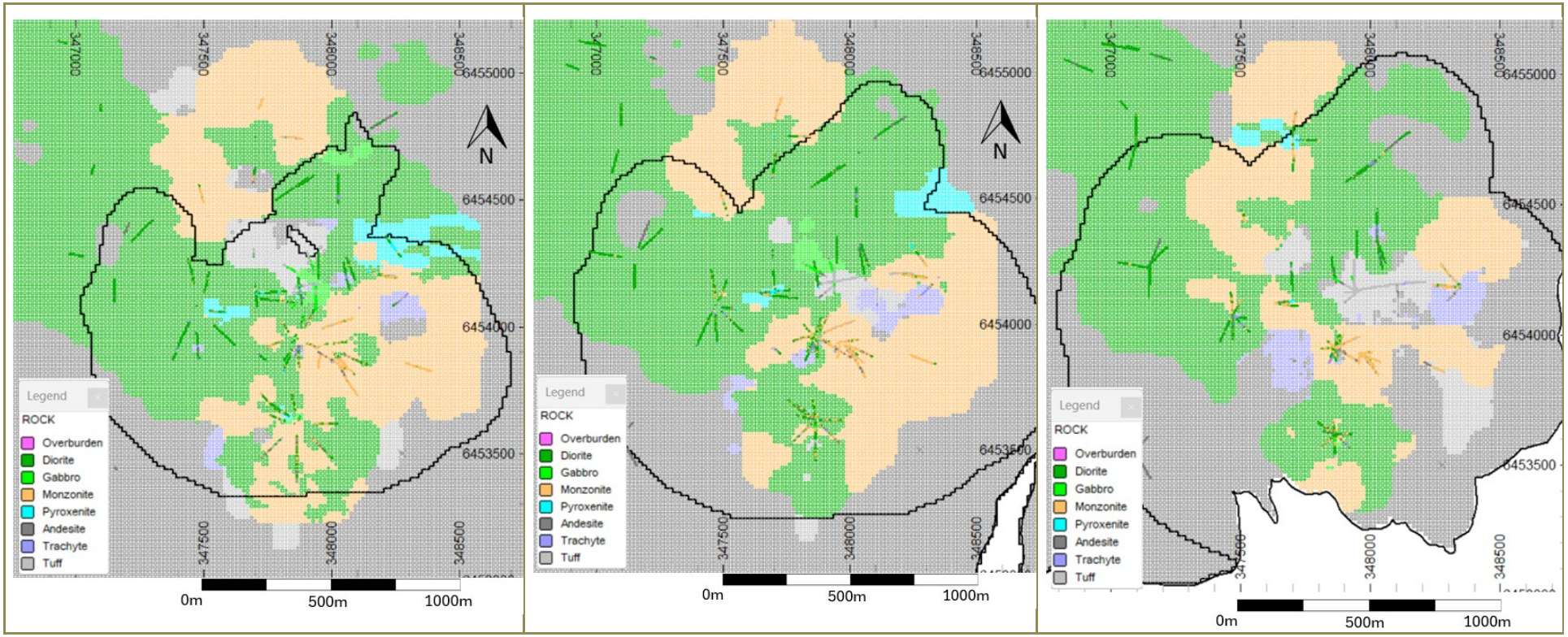
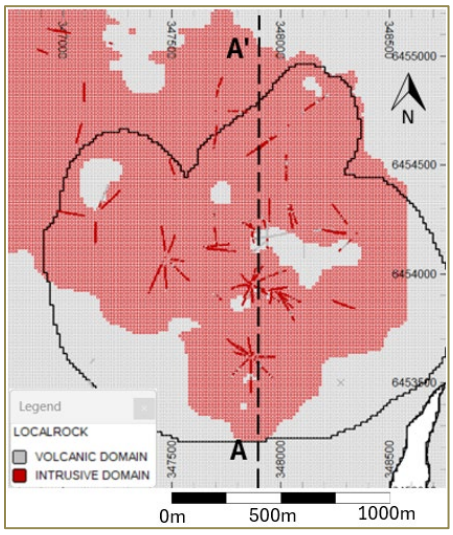


Figure 14-4: Probabilistic Indicator-Based Modelling of Intrusive Formations Embedded within the Volcanic Unit (ASL = Above Sea Level). The Pit Shell is Shown for Orientation Purposes Only and Is Not Part of the Currently Planned Hat Project

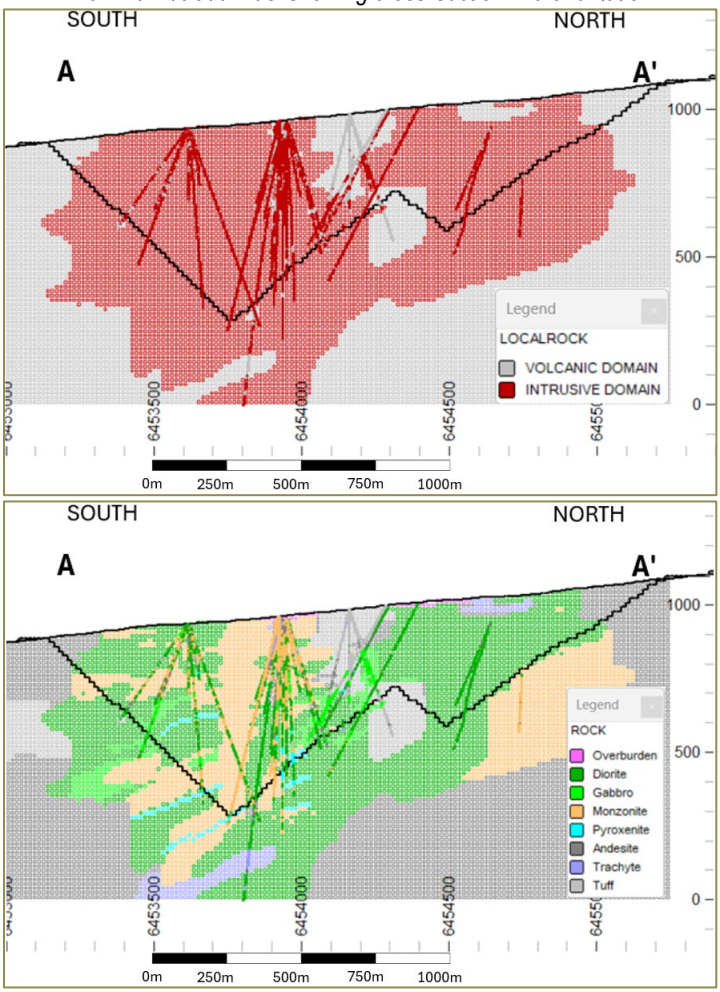


Plan View 700 masl *Plan View 800 masl* *Plan View 900 masl*

Figure 14-5: Indicator-Based Probabilistic Outcomes for Individual Lithological Units Constrained by the First-Step Intrusive–Volcanic Model (ASL = Above Sea Level). The Pit Shell is Shown for Orientation Purposes Only and Is Not Part of the Currently Planned Hat Project



Plan view at 800 masl showing cross-section line orientation



Stepwise probabilistic modelling: (1) volcanic vs. intrusive differentiation → (2) three volcanic units → (3) four intrusive units

Figure 14-6: East 347900 Looking West, Cross-Section of Three Steps Probabilistic Modelling (Grid in Metres). The Pit Shell is Shown for Orientation Purposes Only and Is Not Part of the Currently Planned Hat Project

14.3 Sample Lengths and Compositing

Assay data have an average analytical sample length of 1.9 m. Analytical length refers to the interval of core samples cut and submitted to the laboratory for mechanical preparation and assay analysis.

Regularization was applied to ensure that each sample has equal representation in both statistical and spatial analyses. An additional objective of compositing was to reduce variability within the populations. It is important to note that the grade variables exhibit high variability at both global and local scales. For the Hat Project, recognized as a porphyry-style deposit, the global coefficient of variation (CV) was calculated above 3.0 for gold, silver, and copper, which is unusually high for this deposit type (see Table 14-4).

In porphyry copper systems, the CV is commonly used as an early indicator of grade variability and estimation complexity. Typical porphyry deposits exhibit relatively low CVs values (<1), which provide greater flexibility in the choice of estimation techniques (Bargawa et al., 2023). In contrast, the Hat Project shows an unusually high CV (>3), indicating significantly higher-grade variability than is generally observed in porphyry copper–gold systems.

Table 14-4: Descriptive Statistics for the Variable-Length Sample Grades of the Five Variables Modelled in the PEA: Au, Ag, Cu, Co, and Sc; Each Row Summarizes the Distribution of Sample Values for One Element

Element	Record	Minimum (ppm)	Maximum (ppm)	Average (ppm)	Variance (ppm^2)	SD (ppm)	Skewness	Kurtosis	CV
Au	25534	0.00	18.95	0.10	0.13	0.36	19.70	633.31	3.5
Ag	25534	0.01	45.90	0.28	0.87	0.93	20.86	701.27	3.3
Cu	25534	1.00	168,000	1,162.94	12,463,832	3,530.42	17.69	524.42	3.0
Co	25534	1.10	4,470.00	62.16	8,345.58	91.35	13.59	344.56	1.5
Sc	25139	0.40	173.40	28.89	84.39	9.19	0.55	3.84	0.3

Future mining scale was also considered. Based on the Project size and the generally low grades, an economically viable operation would likely require production between 45 and 70 kt/d. This extrapolation suggests possible bench heights of 15 to 20 m. Accordingly, block dimensions can be supported by composite lengths ranging from 5 to 15 m without excessively smoothing the grade distribution.

Several compositing exercises were carried out using lengths of 5, 10, and 15 m:

- 10 m composites provided superior selectivity and estimation quality compared to 5 m composites, relative to block size
- 15 m composites yielded higher estimation quality overall, but the degree of smoothing shortened the high-grade tailings and compromised representation of higher grades.

Estimation quality was assessed using the regression line coefficient (slope of regression), a standard diagnostic parameter derived from the kriging system and commonly applied in resource estimation and classification. This coefficient measures the linear relationship between true and estimated grades and is

directly linked to the kriging weights and estimation variance, as Emery (2005) described. For the Hat Project, the regression coefficient was used to evaluate block-grade distribution by swapping databases only. Based on the observed behaviour of the regression slope, the decision was made to regularize the sample support to a length of 10 m to improve estimation efficiency and reduce conditional bias (See Table 14-5).

Table 14-5: Descriptive Statistics for the 10 m Composite Grades of the Five Variables Modelled in the PEA: Au, Ag, Cu, Co, and Sc; Each Row Summarizes the Distribution of Composite Values for One Element

Element	Record	Minimum (ppm)	Maximum (ppm)	Average (ppm)	Variance (ppm^2)	SD (ppm)	Skewness	Kurtosis	CV
Au	4882	0.00	3.40	0.10	0.04	0.20	7.37	84.19	1.9
Ag	4882	0.01	5.75	0.28	0.22	0.47	5.63	45.26	1.7
Cu	4882	3.16	28,910.00	1,150.34	3,999,859	1,999.96	5.97	54.57	1.7
Co	4882	5.00	475.00	61.50	2,409.46	49.09	3.69	20.08	0.8
Sc	4797	3.48	61.00	28.92	50.34	7.09	0.27	0.60	0.2

For lithological modelling, an additional objective was to increase sample density within the model space to better capture local variability, particularly in areas where dyke-like characteristics are present. To achieve this, a composite length of 3 m was applied to the lithology data.

14.4 Identification of the Natural Cut-Off Grade

The current PEA resource model for the Hat Project includes five grade variables (Au, Ag, Cu, Co, and Sc). Across the complete dataset of 97 drill holes, preliminary stationarity analysis of grade and variance by lithology shows no meaningful trends or dependencies. Although certain alteration styles show some association with elevated grades—particularly potassic, sodic, and chlorite–epidote assemblages, these relationships are not sufficiently consistent or robust to justify geological sub-domaining at this stage.

In all, 97 drill holes were used for the 2026 Mineral Resource estimation. Qualitative review of lithology and alteration logging does not reveal any clear geological feature capable of defining a coherent mineralized domain. In the absence of such evidence, subdividing the variable universes on geological grounds is not warranted. Should stronger geological controls become evident in future work, sub-domaining may be reconsidered.

Given this context, the purpose of the present analytical exercise is to identify an inflection point that separates low-grade background values from relatively higher grades. Establishing this threshold is necessary to approximate the volume of potentially economic mineralization and to determine whether any underlying structure exists that could support a sustainable and defensible domain definition.

14.4.1 Global Grade Distribution

Figure 14-7 shows that most values in the global grade distribution are low-grade. However, for silver, gold, and copper, long but narrow high-grade tails appear in the distribution, starting around the 70th percentile of the cumulative distribution function.

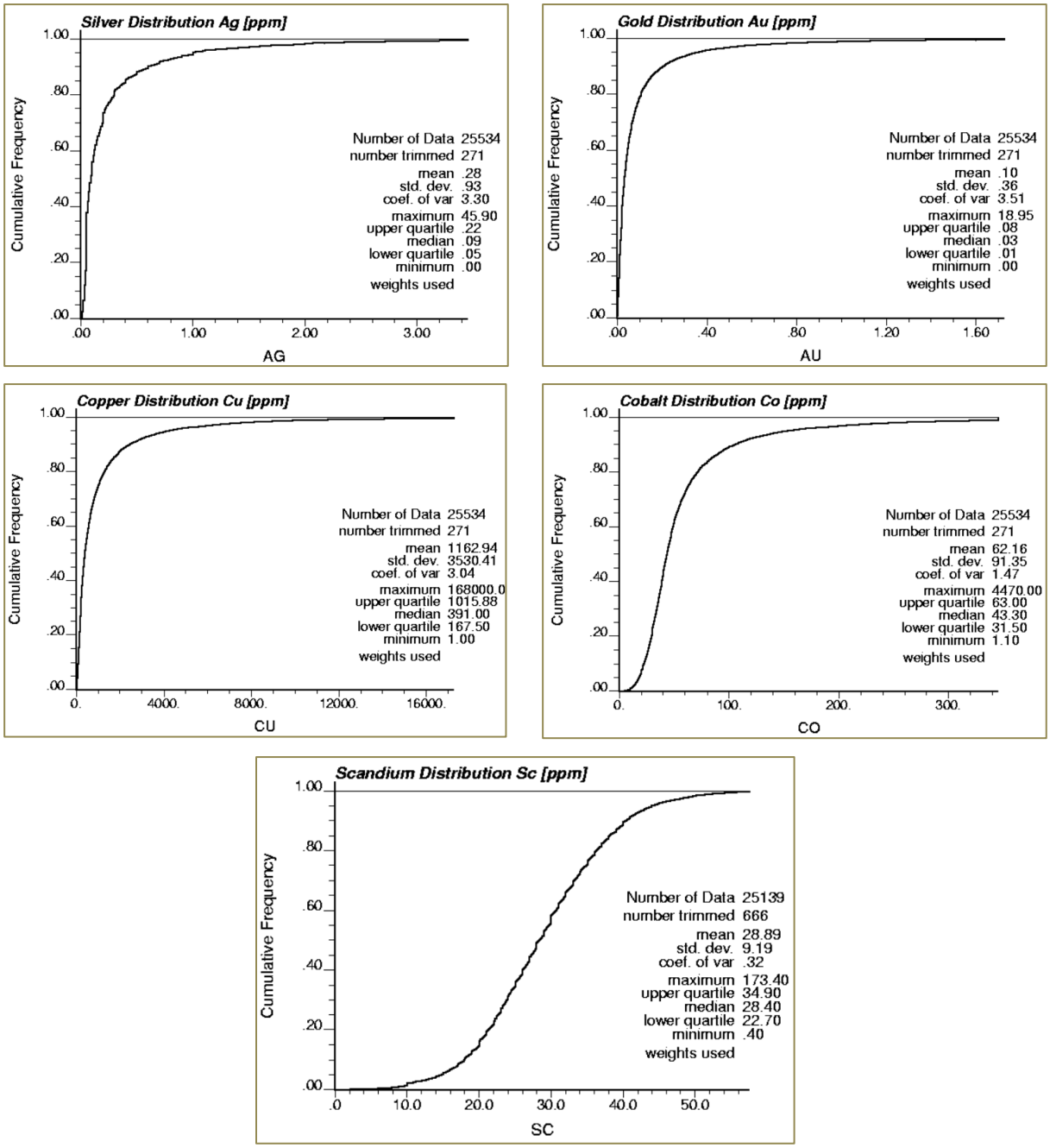


Figure 14-7: Global Grade Distribution for Each Grade Variable

It is important to note that these variables exhibit extremely high CVs, which distinguishes them from the grade distributions typically observed in porphyry copper deposits (Section 14.3).

The cobalt population presents a well-developed distribution, with a low-grade tail and a high-grade tail beginning to develop around the 4th decile.

The scandium population follows a near-normal distribution, with a skewness coefficient close to 1 and kurtosis close to 3. No evidence of bimodal distribution was observed, suggesting that no superimposition of distinct populations is present. The high proportion of low-grade values prompted the development of an analytical, volumetric approach to assess whether the estimation universe could be meaningfully subdivided into domains. Within the dataset, a proportional effect is inherent, with variance increasing as grade increases—a behaviour characteristic of metal-grade populations (Journel & Huijbregts, 1978; Sinclair, 1974). The method applied, and further detailed in Section 14.4.2, provides a structured means of detecting internal divergence within what is otherwise treated as a single, undifferentiated population.

14.4.2 Mobile Panel

The objective of the mobile panel exercise is to determine an inflection point at which low-grade values can be separated from relatively higher grades. Such separation is required to define the probable volume of economic grades and to exclude the low-grade background.

For the Hat Project, the exercise was performed on rough data for five variables. A randomly moving panel of 100 × 100 × 30 m was applied, with a minimum cut-off of seven samples required to compute the statistics.

The resulting data dispersion cloud was processed to calculate conditional average variance within each grade class. Figure 14-8 illustrates the results of the exercise. At the lower end of the grade distribution, where the inflection point is expected, relative differences between successive data classes were calculated against the constant grade step. This inflection can be observed in Figure 14-8, when the trend of variance begins to increase more rapidly.

Four of the five variables behaved as expected, exhibiting increasing variance with increasing grade. This relationship allowed identifying a natural cut-off for the four analyzed variables in Table 14-6. Scandium did not exhibit any significant increase in variance as grade increased. Scandium is therefore considered a spatially stationary variable, exhibiting relatively stable mean and variance. This behaviour provides an a priori indication of the expected Mineral Resource classification outcome for scandium.

Scandium is a compatible element in many rock-forming minerals such as pyroxenes, amphiboles, and clays. It is therefore rarely concentrated except in specific geological settings (Natural History Museum, n.d.).

Similar behaviour has been documented in other scandium-bearing systems, including lateritic deposits (e.g., Syerston–Flemington), bauxite-hosted scandium layers (e.g., Karaçam), and mafic–ultramafic cumulate intrusions (e.g., Crater Lake), where scandium exhibits low spatial variability due to its residence in major rock-forming minerals.

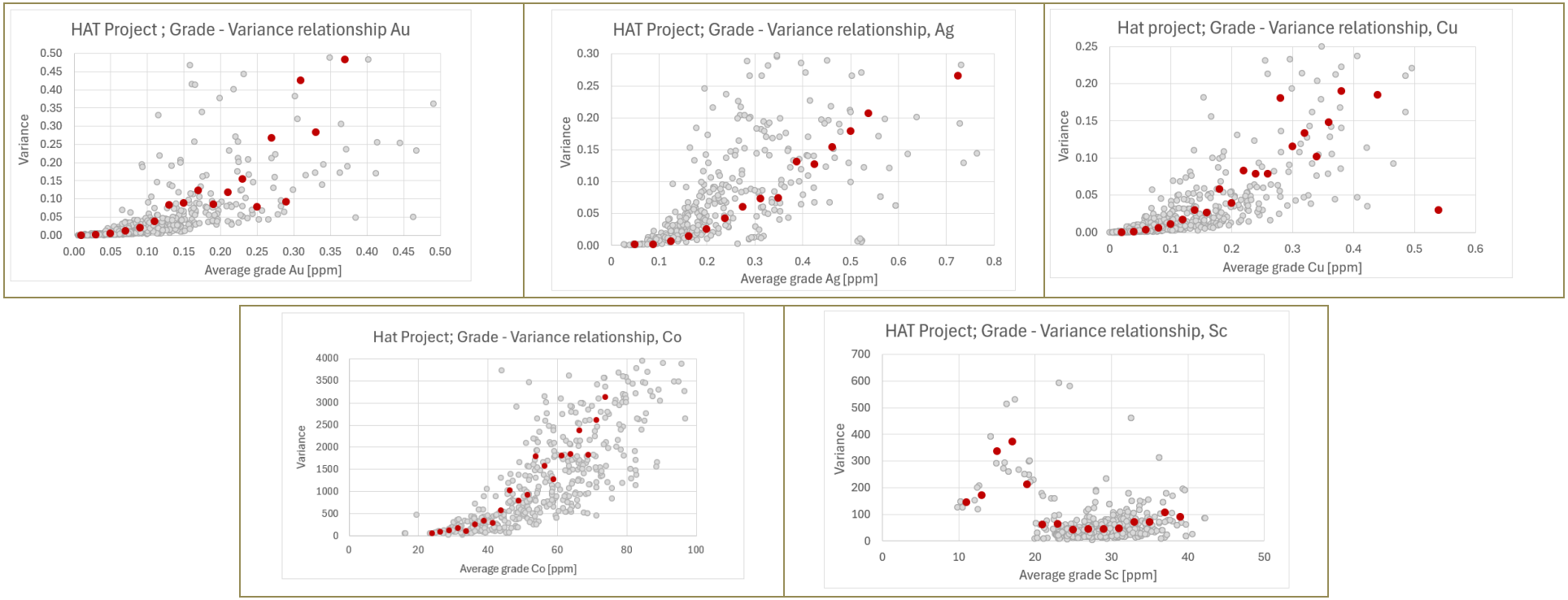


Figure 14-8: Grade–Variance Dispersion Distribution to Assess Natural Cut-Off Within Population

Table 14-6: Selected Cut-Off Grades

Element	Cut-Off (ppm)
Au	0.07
Ag	0.1
Cu	900
Co	35
Sc	n/a

The exercise supports the conclusion that scandium will be treated as a single domain of mineralization in the estimation process. Other variables will be analyzed according to defined cut-offs, which will be decisive in guiding future data treatment throughout estimation and classification exercises.

Grades below the cut-off define the background mineralization, while grades above the cut-off represent the portion of the population with potential economic significance

14.4.3 Creation of Indicator Database

The indicator databases were constructed using 10 m composites for four variables—copper, gold, silver, cobalt—based on the selected cut-offs (Table 14-6). The modelling objective is to develop a probabilistic model of the mineralized domain above the defined cut-offs.

A probabilistic modelling approach was selected over deterministic mesh-bound methods to better capture geological uncertainty, quantify confidence levels, and provide a transparent basis for resource classification. This method allows each block to be assigned a probability of exceeding the selected cut-off, supporting risk-based evaluation and ensuring compliance with NI 43-101 standards.

14.5 Reasons for Choosing Probabilistic Modelling

- Captures geological uncertainty
  - Deterministic mesh models impose hard boundaries (inside/outside), which can oversimplify complex mineralization.
  - Probabilistic modelling assigns a likelihood (0–1) of exceeding cut-off, better reflecting uncertainty in grade continuity and geology.
- Improves Mineral Resource classification
  - Probabilistic outputs provide a quantitative measure of confidence for each block.
  - Supports NI 43-101 requirements for distinguishing Measured, Indicated, and Inferred categories based on confidence levels.

- Handles sparse or variable data
  - In areas with limited drilling or irregular sampling, deterministic boundaries can misrepresent mineralization.
  - Probabilistic models integrate variography and spatial correlation, allowing interpolation of probabilities even in data-poor zones.
- Supports risk-based decision making
  - Investors and engineers benefit from understanding not just "where ore is" but how likely it is to be economic.
  - Probabilistic models allow scenario testing (different cut-offs, economic assumptions) without redrawing rigid meshes.
- Transparency and defensibility
  - Probabilistic modelling is based on indicator kriging interpolation, as well-established geostatistical method.
  - It provides clear, reproducible metrics that can be audited, unlike subjective deterministic boundary drawing.

At short local scales, indicator distribution can be highly variable. In some cases, intervals from nearby drill holes may yield contrasting values that appear to oppose one another. The most reliable way to resolve these discrepancies is through spatial weighting, with kriging interpolation. Ordinary kriging was selected because it is a best linear unbiased estimator (BLUE), providing the most objective and unbiased interpolation of spatially correlated variables (Ashtiani, Z., & Deutsch, C. V. (2024); Emery & Lantuéjoul, 2006; Markvoort & Deutsch, 2024). Consequently, a probabilistic approach is preferred in complex situations, particularly where drill holes are arranged in a fan pattern, with spacing that increases with depth, and very narrow data spacing near the drill pad.

The method follows the indicator convention, where a value of 1 is assigned to the target population (samples equal to or above the cut-off), and a value of 0 is assigned otherwise (Table 14-6).

Statistical analysis of the new database reveals the proportion of data above cut-off for each variable:

- Copper—33.3%
- Gold—37.5%
- Silver—58.5%
- Cobalt—75.5%.

14.5.1 Indicator Variography

Spatial continuity of the indicator variables was evaluated using the correlogram, defined as the complement of the variogram: $\rho(h) = 1 - \gamma(h)$

The commonly used expression is obtained directly from the correlogram definition by substituting the covariance–variogram relationship and assuming second-order stationarity.

$$\rho(h) = Cov[Z(u), Z(u + h)] / [\sigma(u) \times \sigma(u + h)]$$

Where:

Z(u) = value at the head location

Z(u + h) = value at the tail location

Cov[...] = covariance between head and tail samples

σ(u) = standard deviation of head values

σ(u + h) = standard deviation of tail values.

For this reason, the simplified expression ρ(h) = 1 – γ(h) is used in the correlogram figures, as it provides a clearer representation of the formula used to define spatial continuity.

This formulation follows the geostatistical convention described by Deutsch and Journel (1998) and Ashtiani and Deutsch (2024). The correlogram provides an increasing trend with distance, which is more intuitive for interpreting spatial correlation in indicator variables. Empirical correlogram points and fitted 3-D models are presented in Figure 14-9 to Figure 14-12 and were used to define search radii and anisotropy parameters for interpolation. The empirical correlogram points, adjusted spatial correlogram models, and numerical model parameters for each variable.

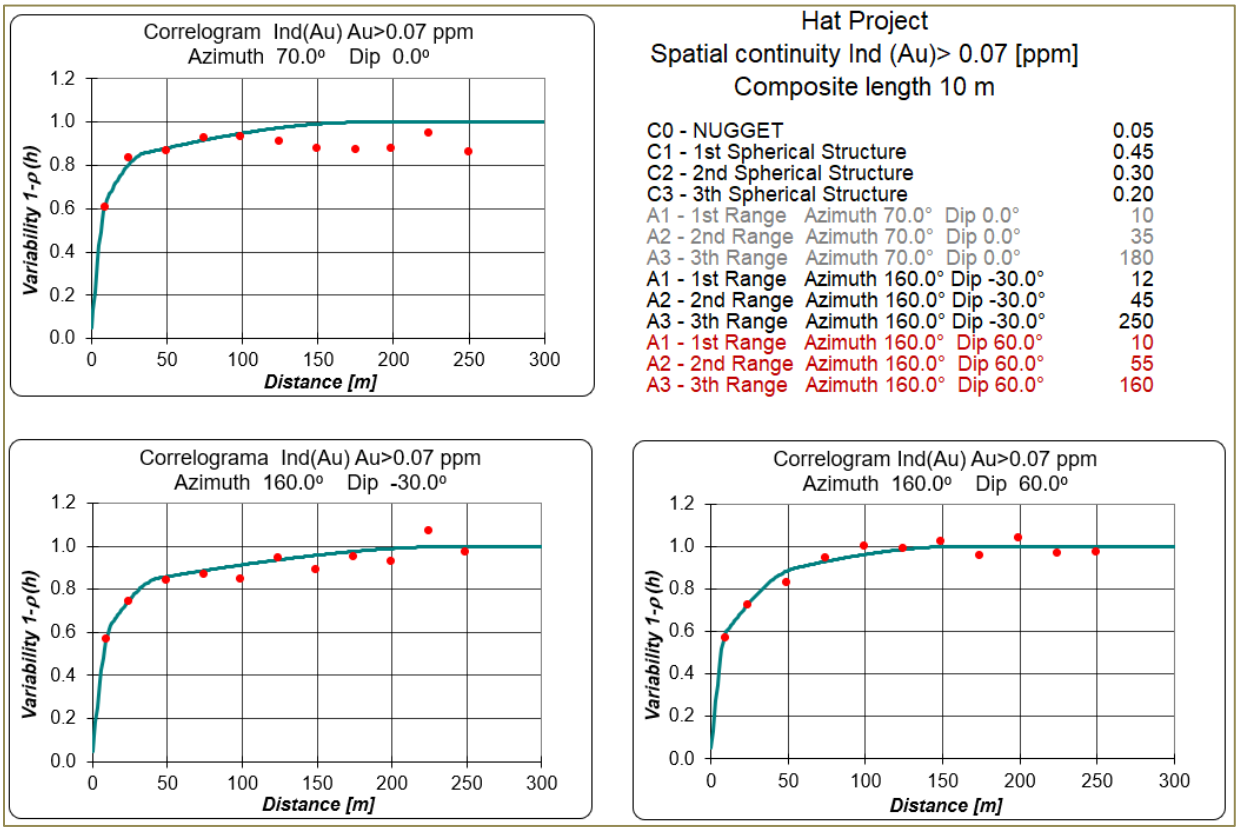


Figure 14-9: Gold—Correlogram Models and Experimental Points Conceived for Indicator Modelling

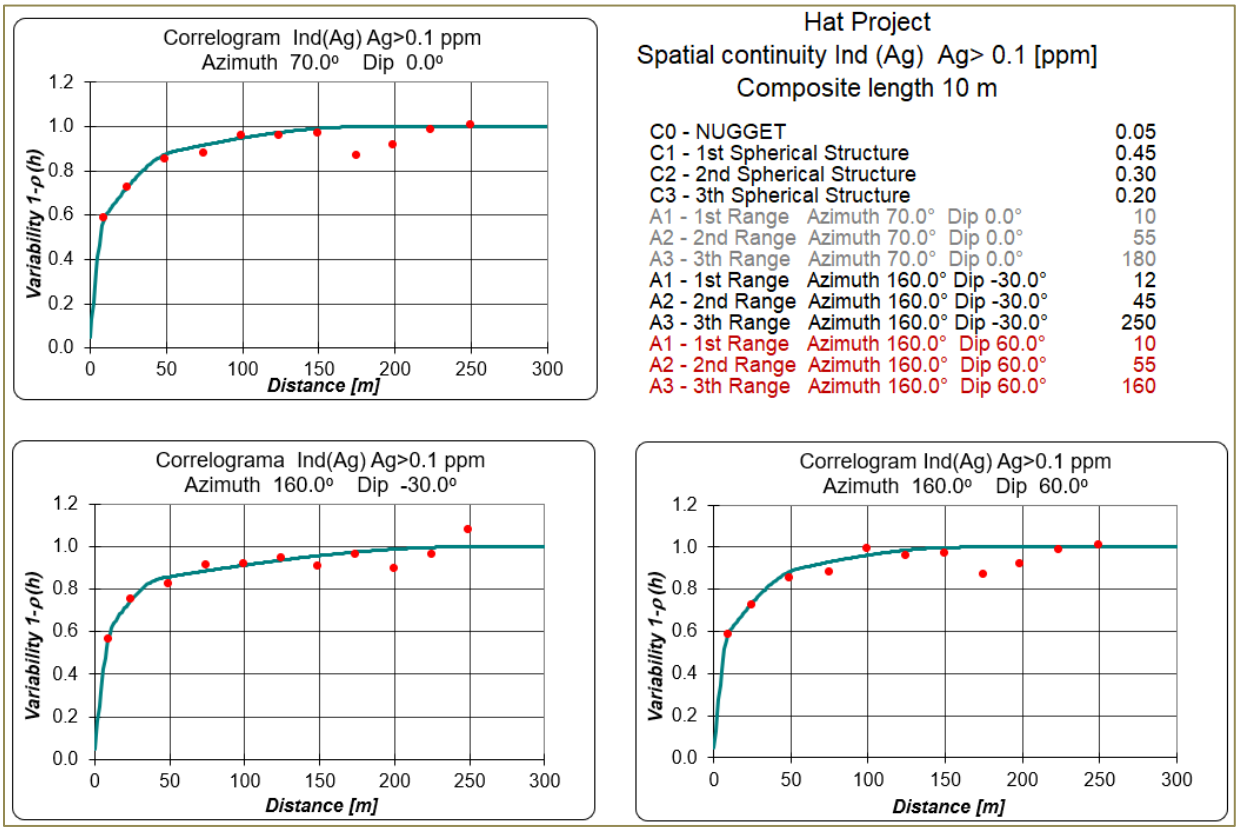


Figure 14-10: Silver—Correlogram Models and Experimental Points Conceived for Indicator Modelling

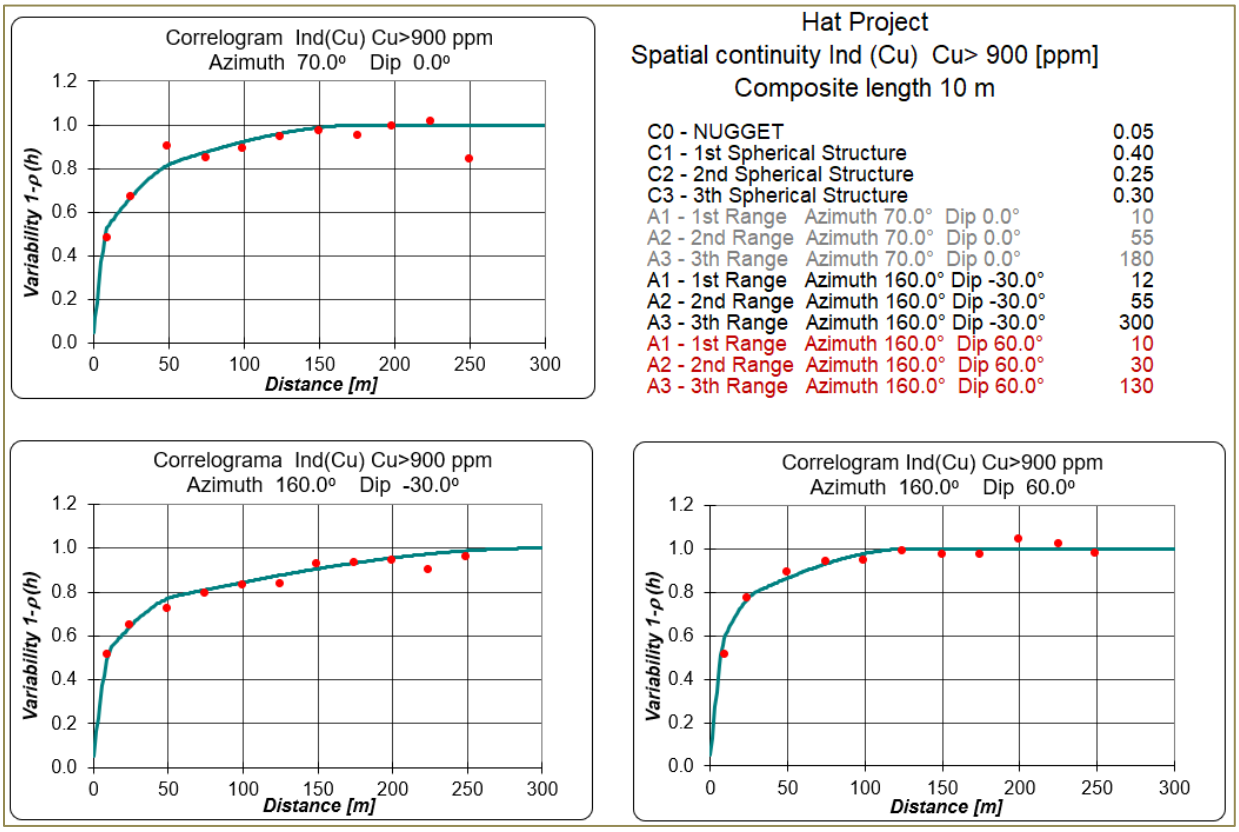


Figure 14-11: Copper—Correlogram Models and Experimental Points Conceived for Indicator Modelling

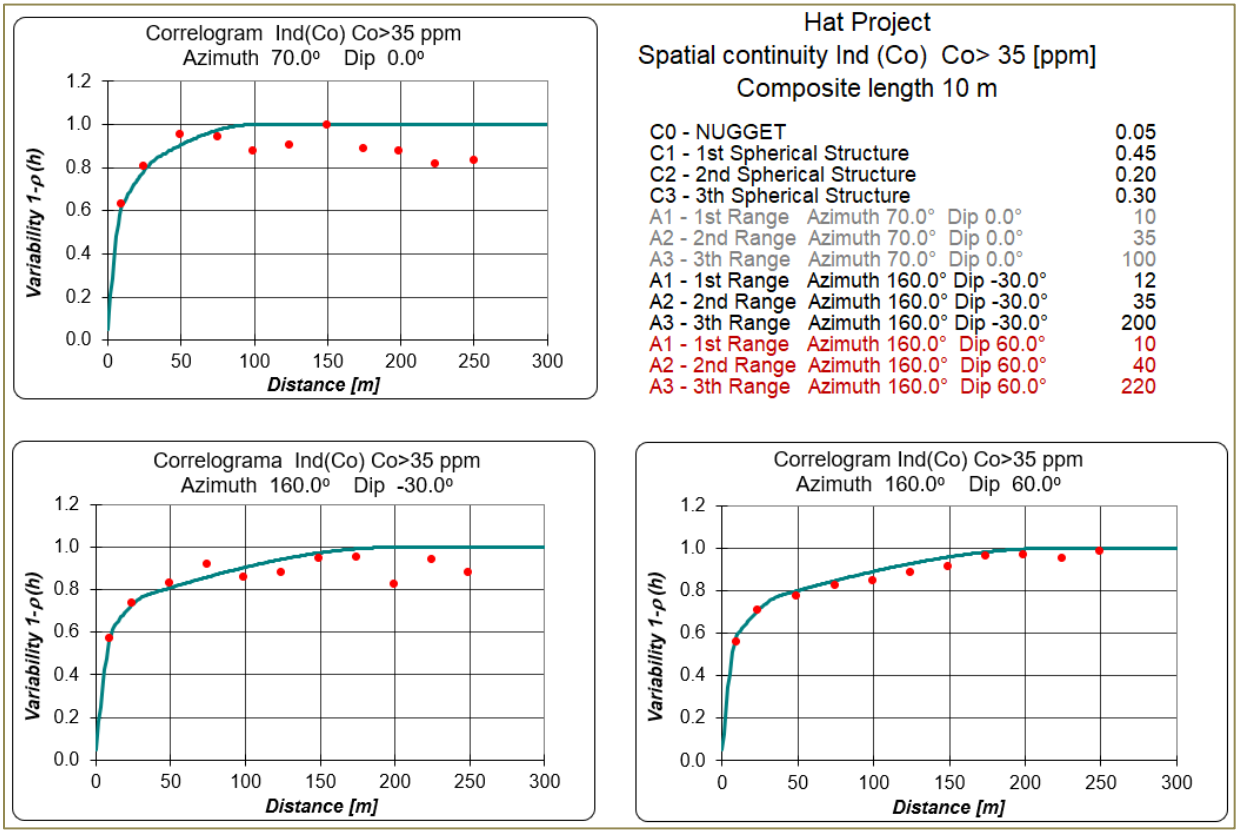


Figure 14-12: Cobalt—Correlogram Models and Experimental Points Conceived for Indicator Modelling

14.5.2 Probabilistic Model Construction

In the probabilistic model, each block is assigned a probability ranging from 0 to 1. A probability of 1 indicates that the block has a 100% likelihood of being above the cut-off defined in Table 14-6, which was used to construct the indicator database. Separate cut-offs were established for each variable, as described in Section 14.4.2.

Individual probabilistic models were created for each variable using ordinary kriging interpolation. The selected probabilistic domains are treated as objective volumes for grade estimation. An elementary probabilistic approach was applied by converting continuous grade values into binary indicators using a defined threshold. In this framework, the kriged indicator represents the probability of exceeding the threshold, with a value of 0.5 serving as the natural probabilistic boundary between volumes below and above the cut-off. This approach follows standard indicator-based probabilistic modelling practices, where the 50% probability level is used as the default decision threshold (Esri, 2024).

Estimation was performed using the following kriging parameters:

- Minimum samples—1
- Maximum samples—16
- Octant selection—not applied

- Maximum samples per drill hole—not restricted
- Search radius—full-radius search applied.

For block spatial consolidation, the probabilistic models were subjected to a smoothing process. This process involved selecting the closest neighborhood of each block, defined as the group of blocks in direct contact with the block under re-evaluation. Weighted probabilities were then assigned to these blocks, ensuring that isolated blocks were merged into the predominant local environment. In this way, the domains were consolidated and prepared for subsequent estimation steps.

Figure 14-13 to Figure 14-16 present the consolidated probabilistic domains for gold, silver, copper, and cobalt. Cross-sections were selected at random to illustrate the quantified variability in domain shapes.

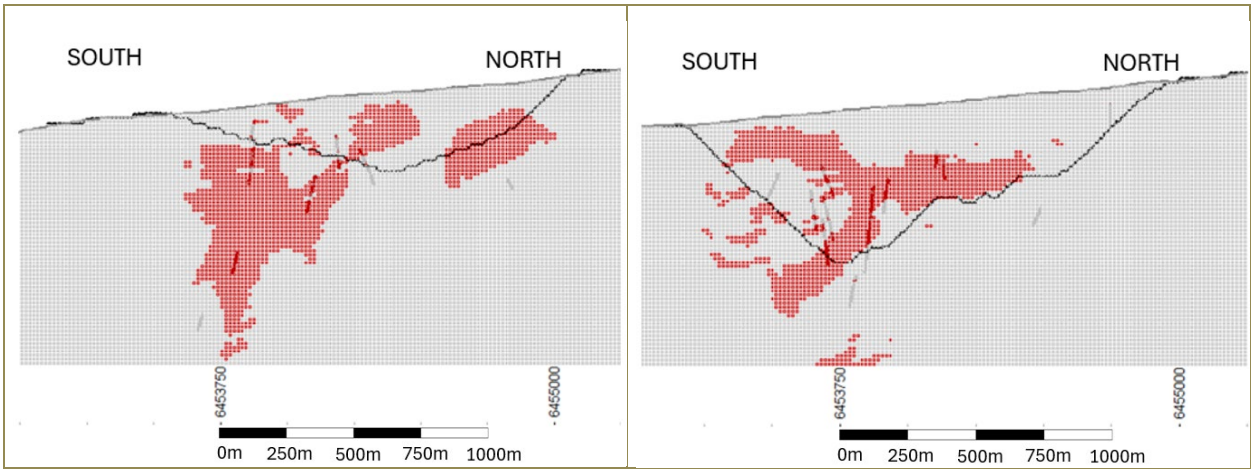


Figure 14-13: View West, Vertical Sections of Probabilistic Gold Domain. The Pit Shell is Shown for Orientation Purposes Only and Is Not Part of the Currently Planned Hat Project.

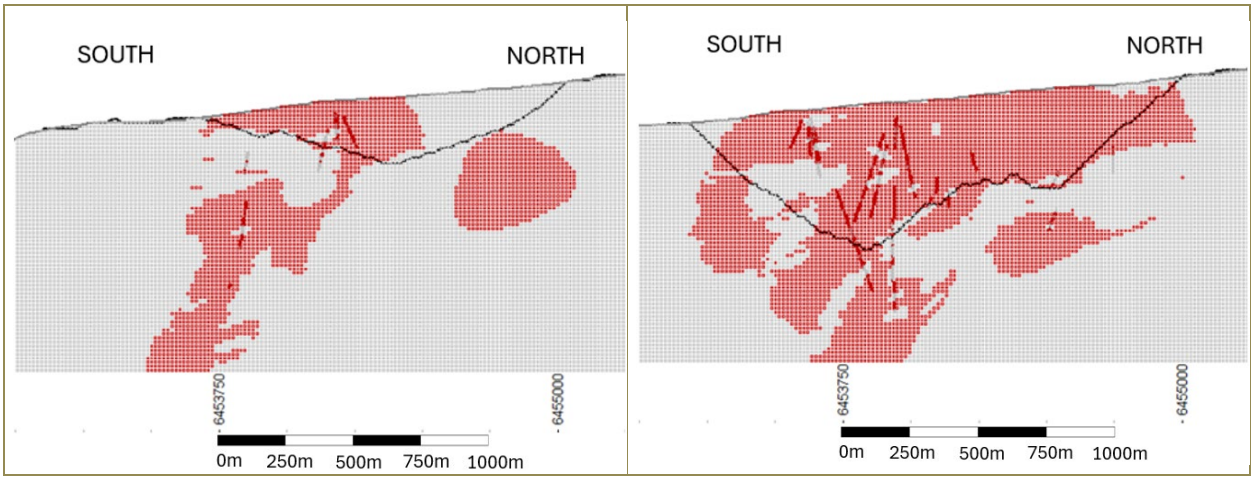


Figure 14-14: View West, Vertical Sections of Probabilistic Silver Domain. The Pit Shell is Shown for Orientation Purposes Only and Is Not Part of the Currently Planned Hat Project

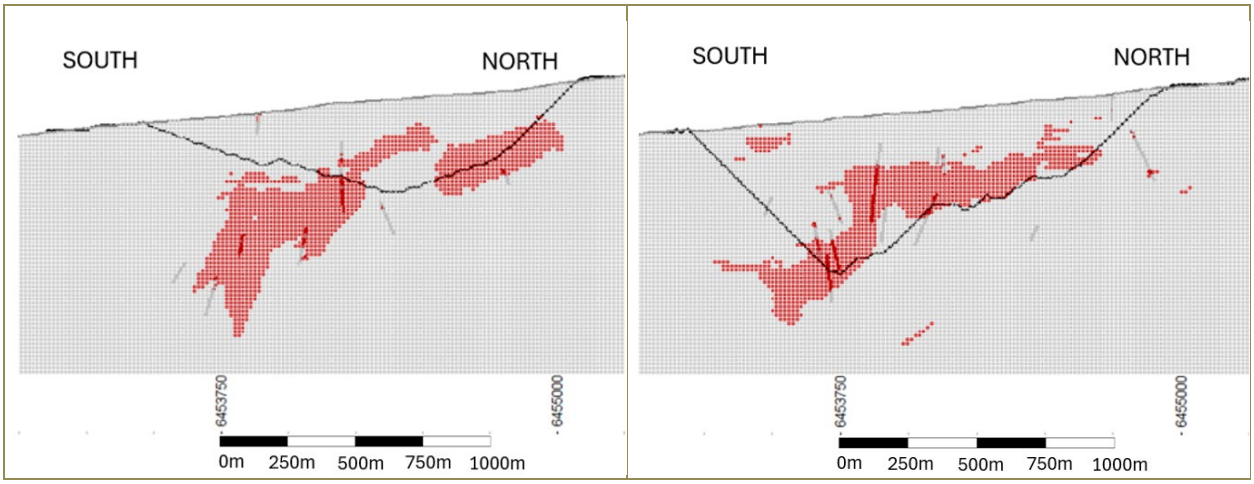


Figure 14-15: View West, Vertical Sections of Probabilistic Copper Domain. The Pit Shell is Shown for Orientation Purposes Only and Is Not Part of the Currently Planned Hat Project

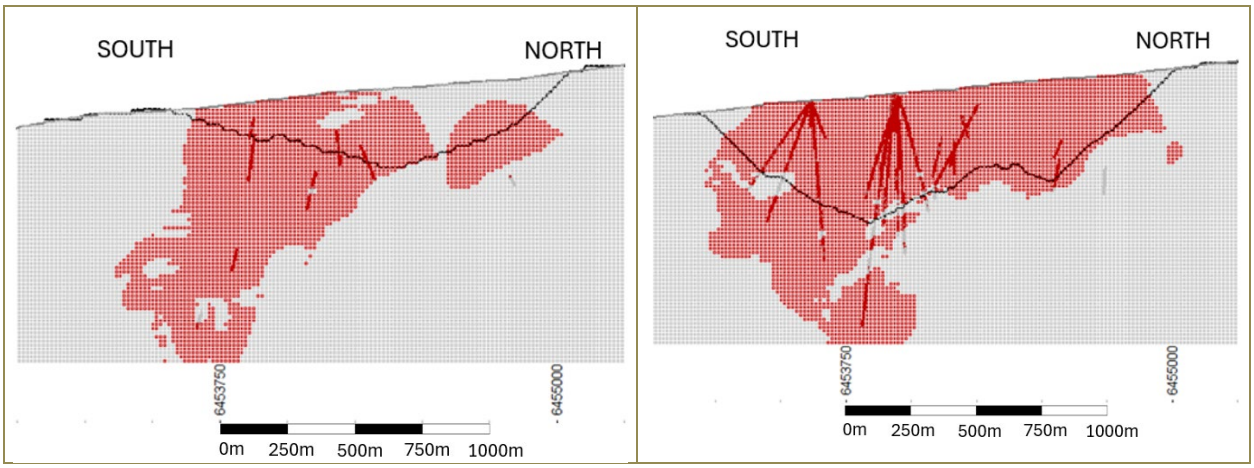


Figure 14-16: View West, Vertical Sections of Probabilistic Cobalt. The Pit Shell is Shown for Orientation Purposes Only and Is Not Part of the Currently Planned Hat Project

The spatial diversity of domains represents probable economic Mineral Resources volume for the four variables. Scandium is not included in the presentation due to its spatially pervasive and global character. Instead, scandium represents an underlying grade distribution and will be estimated without geometric constraints.

The block pattern distribution in indicator modelling reflects both the spatial correlogram model and the spatial distribution of the data. In some cases, high probabilities represent a consolidated orebody, while in other cases the influence of local data may generate separate clusters of blocks or even sparsely distributed blocks. This behaviour is normal and expected (Maleki et al., 2017). Geological shapes and probabilistic domains are not necessarily smooth or regular, and the modelling process captures this natural variability. Understanding this characteristic is essential, as those unfamiliar with the indicator

technique may otherwise misinterpret these patterns as anomalies, when in fact they represent the inherent complexity of geological systems.

14.6 Compilation of Lithology, Geophysics, and Mineralized Domains

Important inputs for the Project have already been discussed and presented. These include the historical geological knowledge of dynamic regional events, the massive intrusive body emplaced into volcanic host rocks, epithermal processes that established a complex mineralization system within the intrusive rocks, and geophysical examinations conducted over both intrusive and volcanic units.

It is both necessary and informative to compile these elements into an integrated geological–mineralization framework, effectively creating a “patchwork” tied to geophysical observations.

14.6.1 Lithological Environment

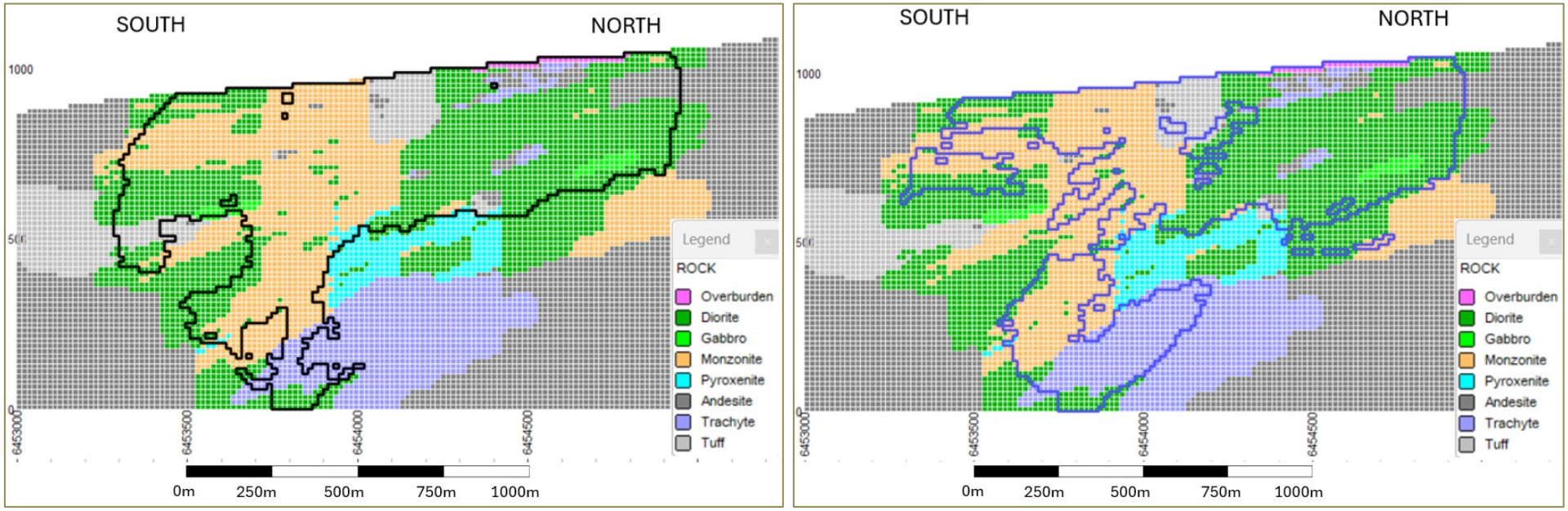
The Project lies within an intrusive rock formation. Epithermal solutions undersaturated in silica migrated through brecciated channels and fracture zones, impregnating the intrusive system of rocks shown in Figure 14-16. Cobalt is the most pervasive mineralization, showing the most consolidated presence. Silver exhibits a large volumetric distribution similar to cobalt, but with a more variable pattern of isolated blocks. Silver is unique among the four variables in extending beyond the non-mineralized footprint at the footwall of the cobalt, gold, and copper zones. Copper and gold display more controlled sectors of distribution, related to the lower portions of the cobalt and silver domains. Both extend deeper in a highly synchronized manner, forming a sub-vertical mineralization column.

14.6.2 Spatial Correlation of Variables

The internal correlation among the four variables (Au, Ag, Cu, Co) is notable. Although the probabilistic models were built independently, with databases treated using analytically defined cut-offs, the spatial coincidence of the domains demonstrates that they represent a single mineralization event with multiple pulses; this is described in Section 7.3. These pulses established an interconnected system among the four variables, as illustrated in Figure 14-17 to Figure 14-20. Copper and gold are spatially correlated, with domain boundaries often extraordinarily aligned across multiple sections. Gold frequently extends southward and below surface where copper is absent, yet both variables respect the hard non-mineralized contact at the cobalt footwall. Cobalt encapsulates the mineralization where gold and copper are developed. Silver generally follows cobalt and hosts copper and gold mineralization, but occasionally appears beyond the no-pass limits defined for cobalt, silver, and gold.

14.6.3 Scandium

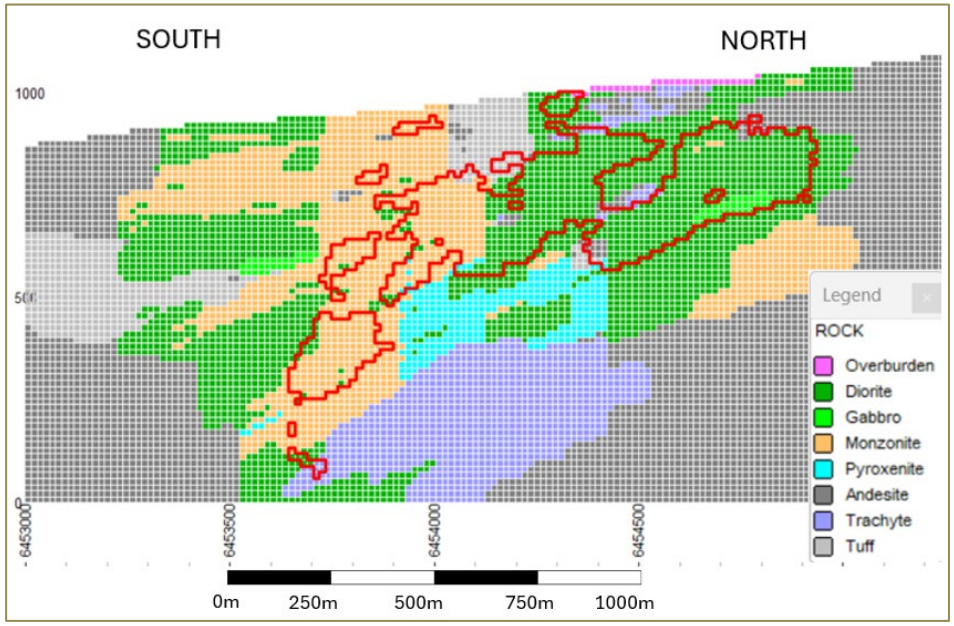
Scandium is not discussed in this section because it exhibits a homogeneous distribution with local highs, behaving as an underlying commodity rather than a structurally or spatially constrained mineralization domain.



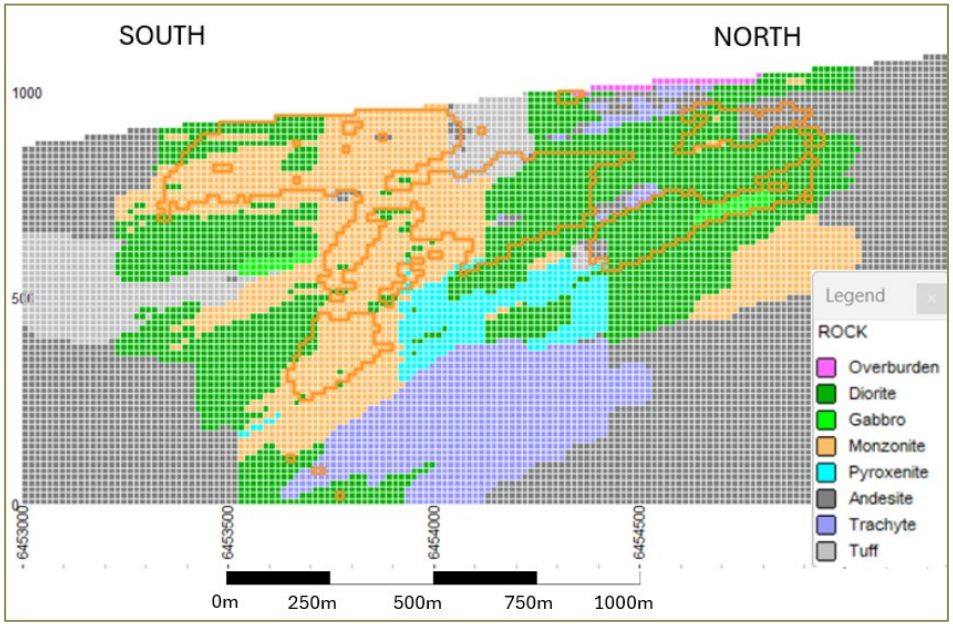
East 348000 Looking West, Cobalt Domain Outlined in Black

East 348000 Looking West, Silver Domain Outlined in Blue

Figure 14-17: East 348000 Looking West, Spatial Relationship Between the Cobalt and Silver Domain Outline and the Intrusive Rock System

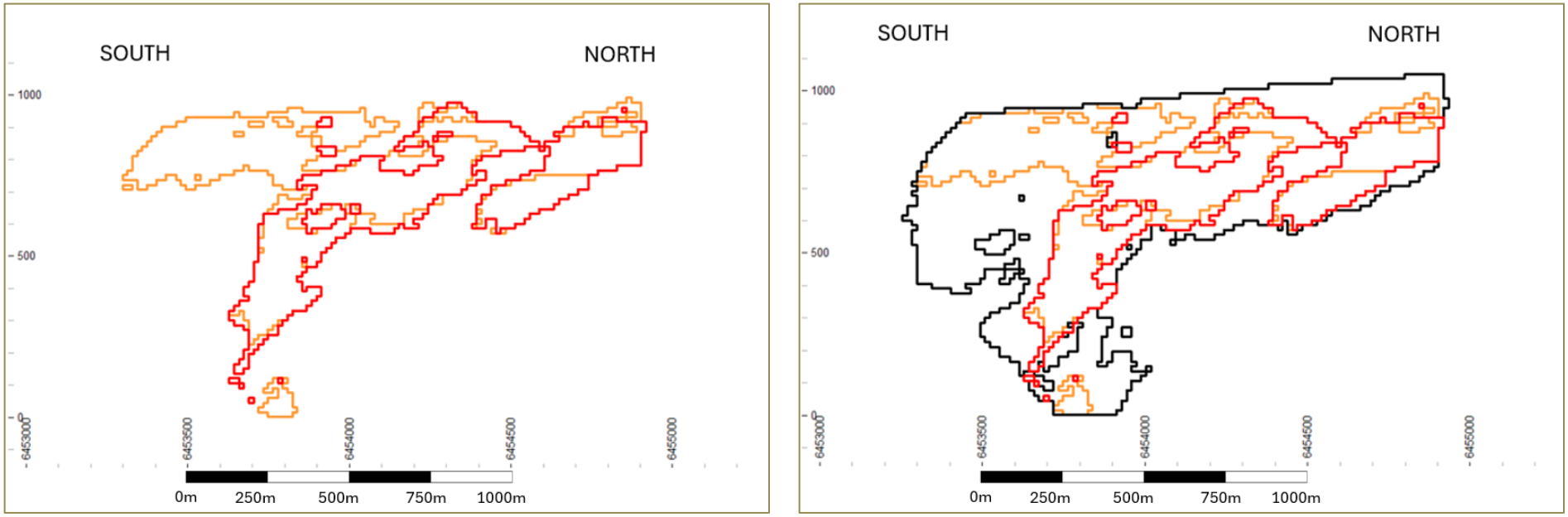


East 348000 Looking West, Copper Domain Outlined in Red



East 348000 Looking West, Gold Domain Outlined in Gold

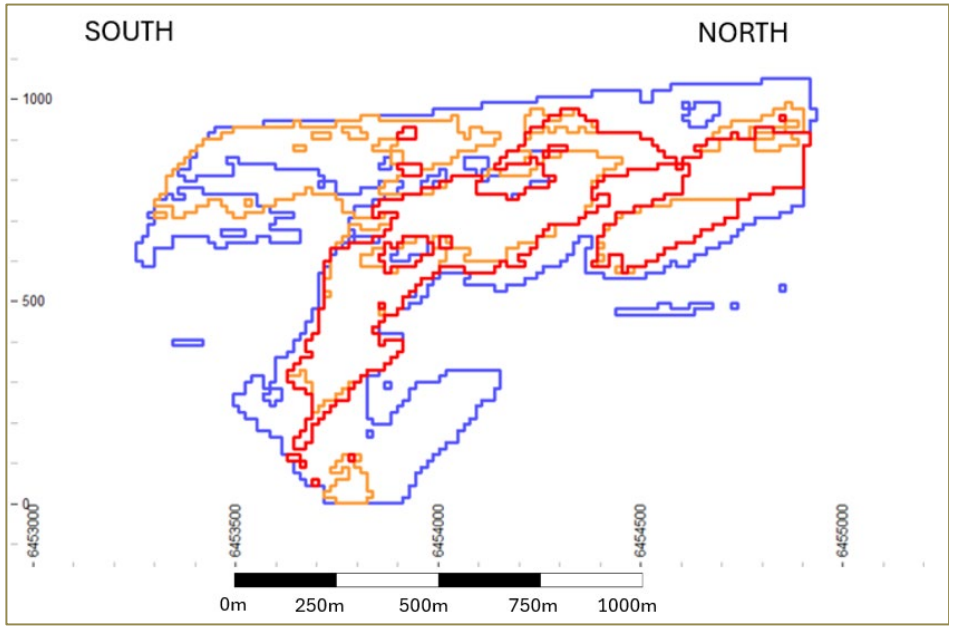
Figure 14-18: East 348000 Looking West, Spatial Relationship Between the Copper and Gold Domain Outline and the Intrusive Rock System



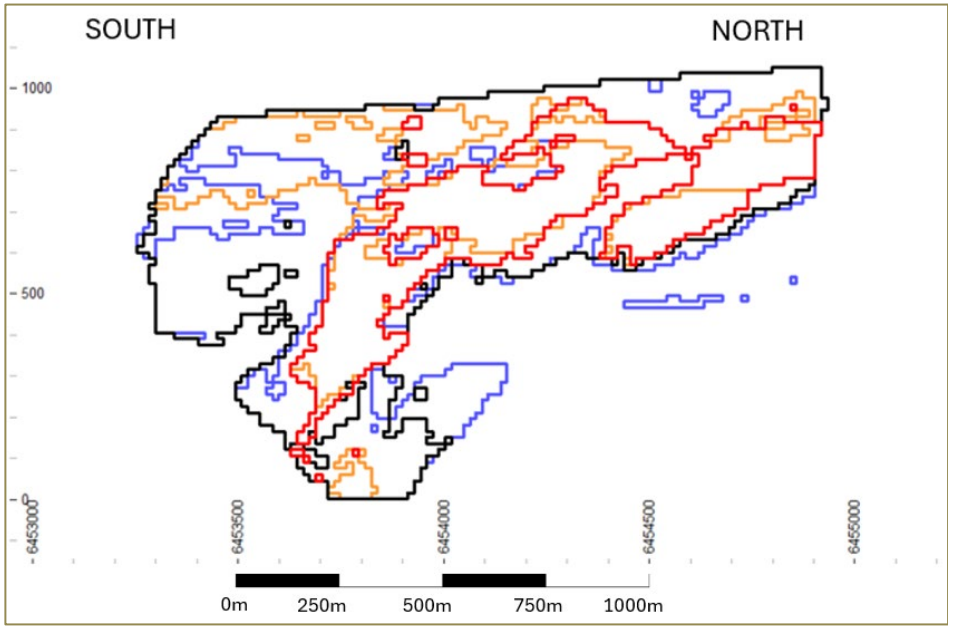
(A) Spatial Correlation between Copper (Red) and Gold (Orange)

(B) Spatial Correlation between Copper (Red), Gold (Orange), and Cobalt (Black)

Figure 14-19: East 348000 Looking West, Cross-Section of Mineralized Domain Spatial Correlation—(A) Spatial Correlation Between Gold and Copper; (B) Spatial Correlation Among Gold, Copper, and Cobalt



(C) Spatial Correlation between Copper (Red), Gold (Orange), and Silver (Violet)



(D) Spatial Correlation of Copper (Red), Gold (Orange), Silver (Violet), and Cobalt (Black)

Figure 14-20: East 348000 Looking West. Cross-Section of Domain Spatial Correlation—(C) Spatial Correlation Among Copper, Gold and Silver; (D) Spatial Correlation Among Copper, Gold, Silver and Cobalt

14.7 Integration of Geophysics and Probabilistic Domain Modelling

As summarized in Section 9.2, high chargeability values in an induced polarization (IP) survey commonly indicate the presence of sulphide minerals capable of storing and releasing electrical charges. Chargeability is therefore a key parameter for identifying, characterizing, and mapping sulphide-bearing mineralization zones.

Figure 14-21 to Figure 14-25 illustrate the spatial correlation between the geophysical prospecting results and the probabilistic geostatistical domain model. The red zones represent bulk chargeability values of approximately 10 mV/V and higher, whereas the dark-blue to light-blue zones mark a sharp transition to low chargeability on the footwall side. This contrast defines an impermeable boundary separating the mineralized system from barren rocks.

The presence of this impermeable boundary is further supported by drill-hole sampling, particularly in holes H090, H091, H089, H039, H040, and H023, among others, which show the abrupt termination of gold, copper, and cobalt mineralization against the low-chargeability footwall. The strongest expression of this boundary is observed in the cross-sections for gold, silver, copper, and cobalt shown in Figure 14-22to Figure 14-25. Silver, while generally following the same spatial trend, locally penetrates slightly deeper into the low-chargeability zone than gold, copper, and cobalt, yet silver still respects the overall boundary geometry. Across these same cross-sections, the high-chargeability zones align closely with all probabilistic domains, with particularly strong correspondence along the steeper and deeper segments of the mineralized system.

The comparison between the geophysical prospecting data and the probabilistic mineralized-domain model demonstrates a clear and coherent spatial correlation in cross-section. The principal chargeability features are well captured within the probabilistic domain envelopes, reinforcing the validity of the modelling approach. This correspondence provides strong support for the probabilistic domain-building method and represents a robust outcome for Mineral Resource estimation.

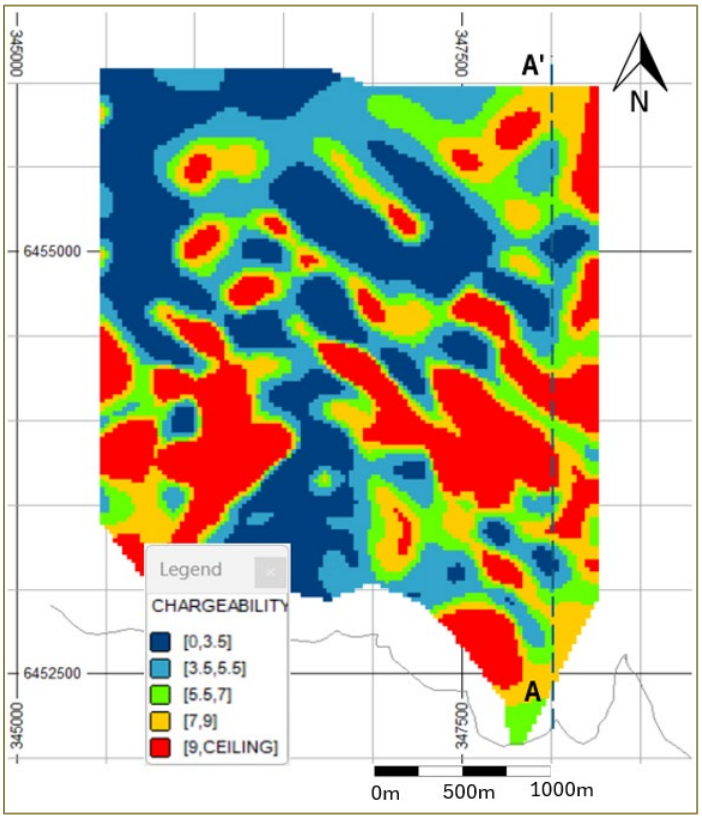


Figure 14-21: DCIP Map Plan View 700 masl IP High-Chargeability (mV/V)

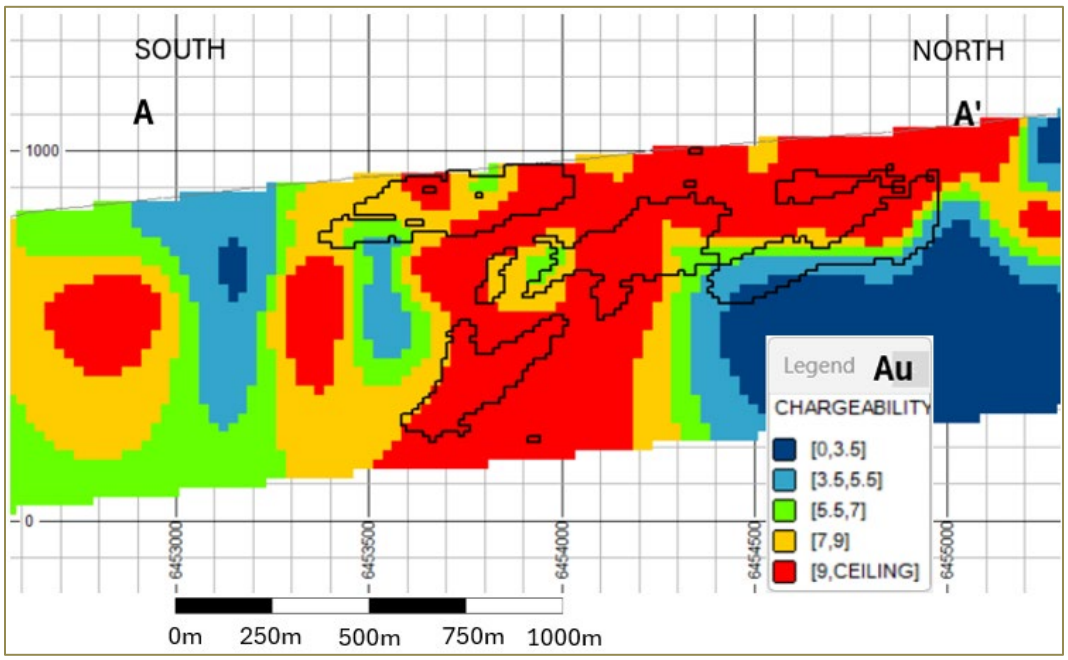


Figure 14-22: Section East 348000 looking West, Gold Domain Outline Overlapping IP High-Chargeability (mV/V)

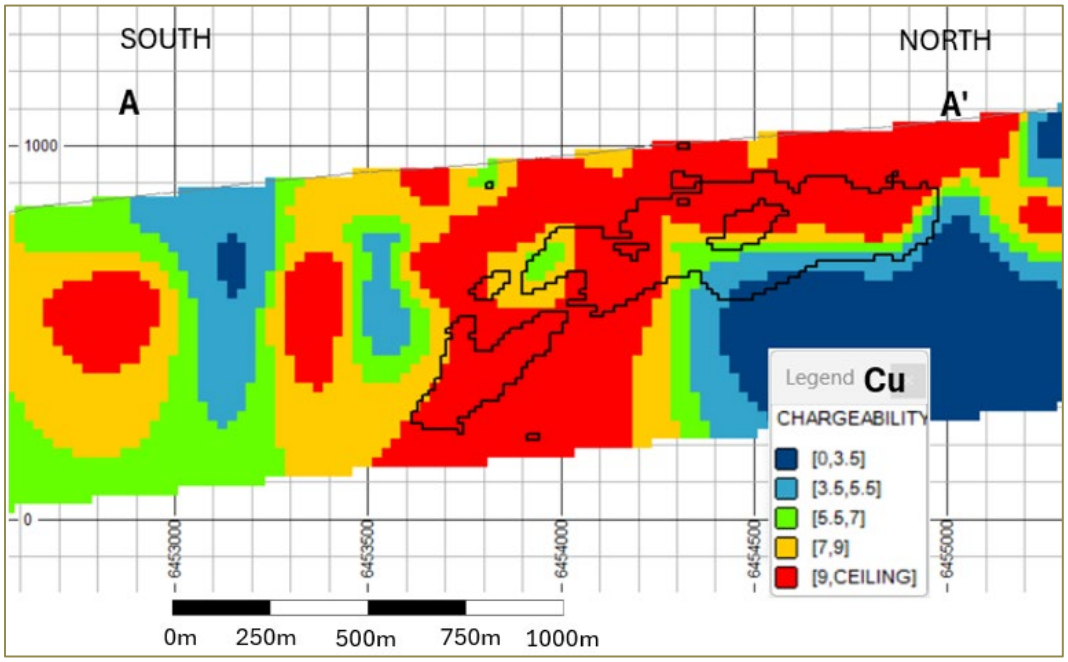


Figure 14-23: Section East 348000 looking West, Copper Domain Outline Overlapping IP High-Chargeability (mV/V)

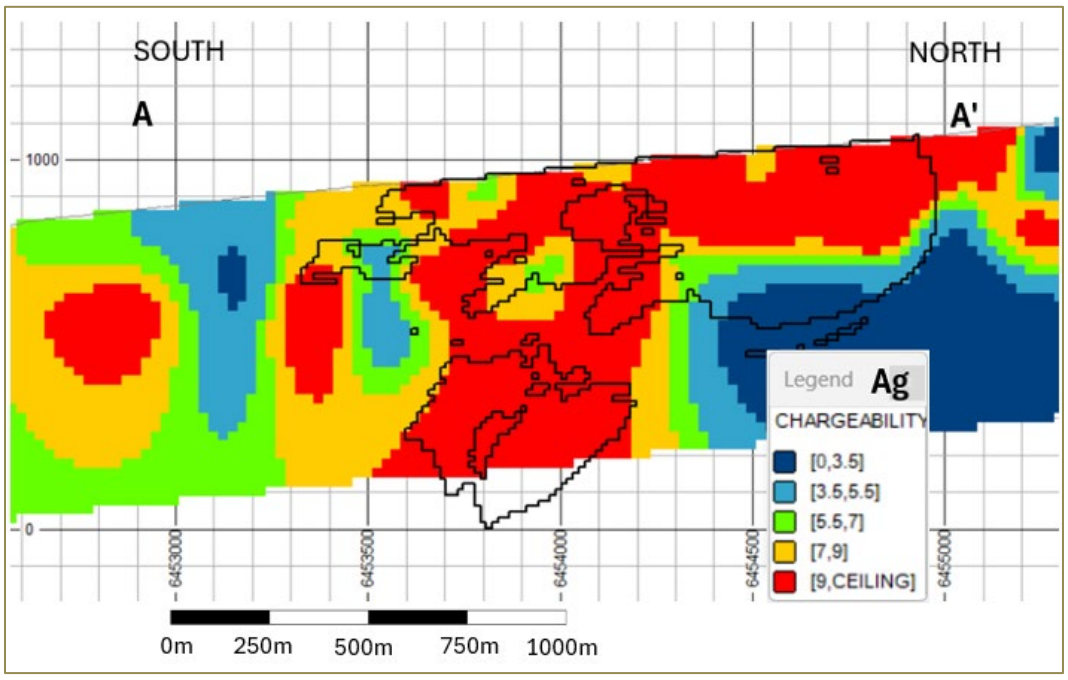


Figure 14-24: Section East 348000 looking West, Silver Domain Outline Overlapping IP High-Chargeability (mV/V)

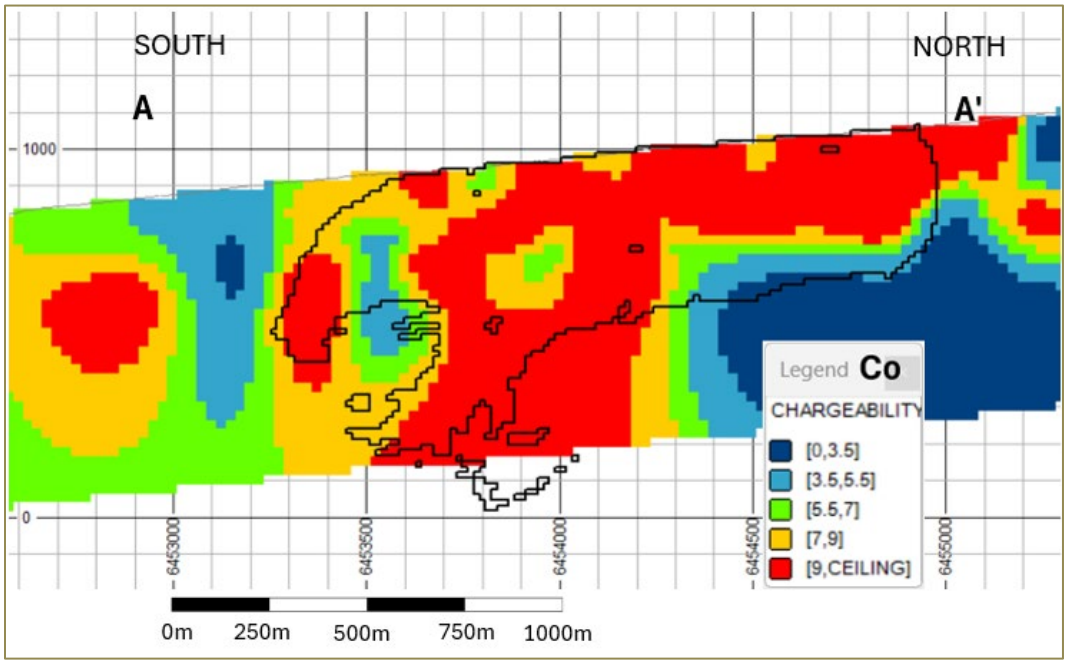


Figure 14-25: Section East 348000 Looking West, Cobalt Domain Outline Overlapping IP High-Chargeability (mV/V)

14.8 Volumetric Selection

14.8.1 Probabilistic Modelling Outcomes

The probabilistic modelling described in Section 14.5.2 produces two principal outcomes. First, it defines a three-dimensional domain representing the target for grade estimation. Second, it provides a corresponding domain for data extraction—based on 10 m composites—that supports grade estimation and classification. For each of the four variables (Au, Ag, Cu, and Co), the probabilistic model is applied to volumetric selection of data. The purpose of volumetric selection, rather than a strict numerical cut-off, is to select all data within the defined domain. Because short-range variability between samples near the cut-off threshold can make selective inclusion challenging, the probabilistic approach was adopted.

It is expected that some data below the cut-off threshold will be present within the target domain, where the probability of a block being above cut-off predominates. These data, selected volumetrically within each domain, are treated as an integrated population representing internal dilution. The analytical method used to quantify the effect of volumetric selection is an exercise known as “back-flagging”, which provides traceability of the data included in the model. This approach aligns with established geostatistical validation practices, including contact analysis and model-to-data comparison methods described in Maleki & Emery (2020).

The purpose of Table 14-7 is demonstrates the block model’s selectivity relative to 10 m composite support when the probabilistic domain volumes are applied to select the composites located inside each domain. Each column tracks the same population of composites as they progress through the selection steps:

- Column 3—How many composites are above the cut-off? This is the starting point. It counts all 10 m composites in the database that exceed the cut-off grade for each metal.
- Column 4—What percentage of the total database does that represent? This simply expresses Column 3 as a percentage of all available composites.
- Column 5—How many of those composites fall inside the domain volume? This shows how many of the above-cut-off composites (Column 3) are captured by the volumetric domain. If Column 5 is smaller than Column 3, the difference is targeting loss.
- Column 6—How many total composites (above and below cut-off) fall inside the domain? This is the full population selected by the domain volume. If Column 6 is larger than Column 5, the extra composites are dilution.
- Column 7—Target loss. These are composites that should be inside the domain (because they are above cut-off) but were not captured by the domain volume.
- Column 8—Dilution. These are composites that should not be inside the domain (because they are below cut-off) but were included by the domain volume.
- Columns 9 and 10—Target loss (%) and dilution (%). These express Columns 7 and 8 as percentages of the total composites above cut-off (Column 3). Copper and gold show the highest percentage of composites excluded, whereas the larger volumes of the cobalt and silver domains mitigate the effects of selectivity.

Table 14-7: Back-Flagging Results (Excluding Scandium)

1	2	3	4	5	6	7	8	9	10
Variable	Cut-Off (ppm)	Composite 10 m Above Cut-Off	% of Total Available Composites	Volumetric Selection of Target Composites	Volumetric Selection of All Composites	Target Loss	Dilution	Target Loss (%)	Dilution (%)
Au	0.07	1,833	37.5	1,552	1,893	281	341	15.33	18.01
Ag	0.10	2,858	58.5	2,643	3,058	215	415	7.52	13.57
Cu	900.0	1,628	33.3	1,324	1,569	304	245	18.67	15.62
Co	35.0	3,687	75.5	3,507	3,875	180	368	4.88	9.50

14.8.2 Outlier Treatment

From a global distribution perspective, capping extreme values was not strictly necessary to avoid overestimation. The high-value composites were not spatially isolated in a way that would expose the estimation outcome to significant risk of bias.

However, the technique of sequential conditional Gaussian simulation can sometimes produce undesirable effects during grade reproduction, particularly when dealing with values that do not consolidate at the distribution tailings. For this reason, selective capping was applied to ensure stability and defensibility of the simulation results.

The database, consisting of 10 m composites, contains isolated outliers that were capped as shown in Table 14-8.

Table 14-8: Data Were Composited to 10 m, with Capping Applied to Isolated Outlier Values

Variable	Capped Composite Count	Capped Value (ppm)
Au	0	n/a
Ag	4	5.75
Cu	1	28,910
Co	7	475
Sc	1	61

14.8.3 Grade Distribution Across Domain Boundaries

In resource modelling, the treatment of domain boundaries plays a critical role in determining how grade estimation is conducted. Specifically, the decision to treat boundaries as permeable or impermeable affects whether samples located just outside the domain are allowed to influence grade interpolation within it. This consideration is particularly important when domains are constructed using probabilistic volumetric selection, as discussed earlier.

To evaluate boundary behaviour, a targeted analysis was performed to assess grade continuity across domain contacts. The objective was to determine whether external composites—those located immediately outside the domain—exhibit grade characteristics that justify their inclusion in the estimation process. If such composites consistently show elevated grades relative to the domain cut-off, the boundary may be considered permeable. Conversely, if external grades are low or inconsistent, the boundary should be treated as impermeable to prevent dilution or bias.

The procedure involved tagging composites according to their relative position with respect to each domain boundary. For each variable (Au, Ag, Cu, and Co), four 10 m composites were selected on either side of the boundary, four internal and four external. This symmetrical sampling approach ensures that the analysis captures short-range grade behaviour without introducing spatial bias.

The evaluation criterion was straightforward: outer composites were considered influential only if their average grade exceeded the cut-off in Table 14-6used to construct the probabilistic domain. This follows standard practice in grade-shell validation, where proximity and grade magnitude jointly determine the permeability of the boundary.

Results indicate that domain boundaries are only subtly influenced by relatively higher grades in adjacent external composites. This influence is minor and does not materially affect the global grade distribution. For all variables analyzed, the average grade of external composites remained below the cut-off thresholds used to define the probabilistic domains.

Accordingly, the boundaries are treated as impermeable for estimation purposes, meaning that external grades are not allowed to influence the interpolation within the domains. This approach ensures that grade interpolation remains consistent with the domain logic and avoids introducing bias from low-grade external samples. It also reinforces the defensibility of the estimation strategy by demonstrating that boundary behaviour was explicitly tested and validated.

Conclusion

The external grade distribution across the probabilistic domain boundaries does not materially influence the estimation strategy. Treating the boundaries as impermeable ensures that grade estimation within the domains remains robust and consistent with the probabilistic modelling approach. The decision to treat domain boundaries as impermeable must also consider that a significant proportion of low-grade composites—those already below the cut-off—are included within each domain. As shown in Section 14.8.1 (Figure 14-26), the diluting composites represent approximately 18% for silver, 13.5% for gold, 15.6% for copper, and 9.5% for cobalt. This quantified grade distribution supports the decision to maintain impermeable boundaries, ensuring that additional low-grade material from outside the domains is not introduced and preventing excessive dilution across domain contacts.

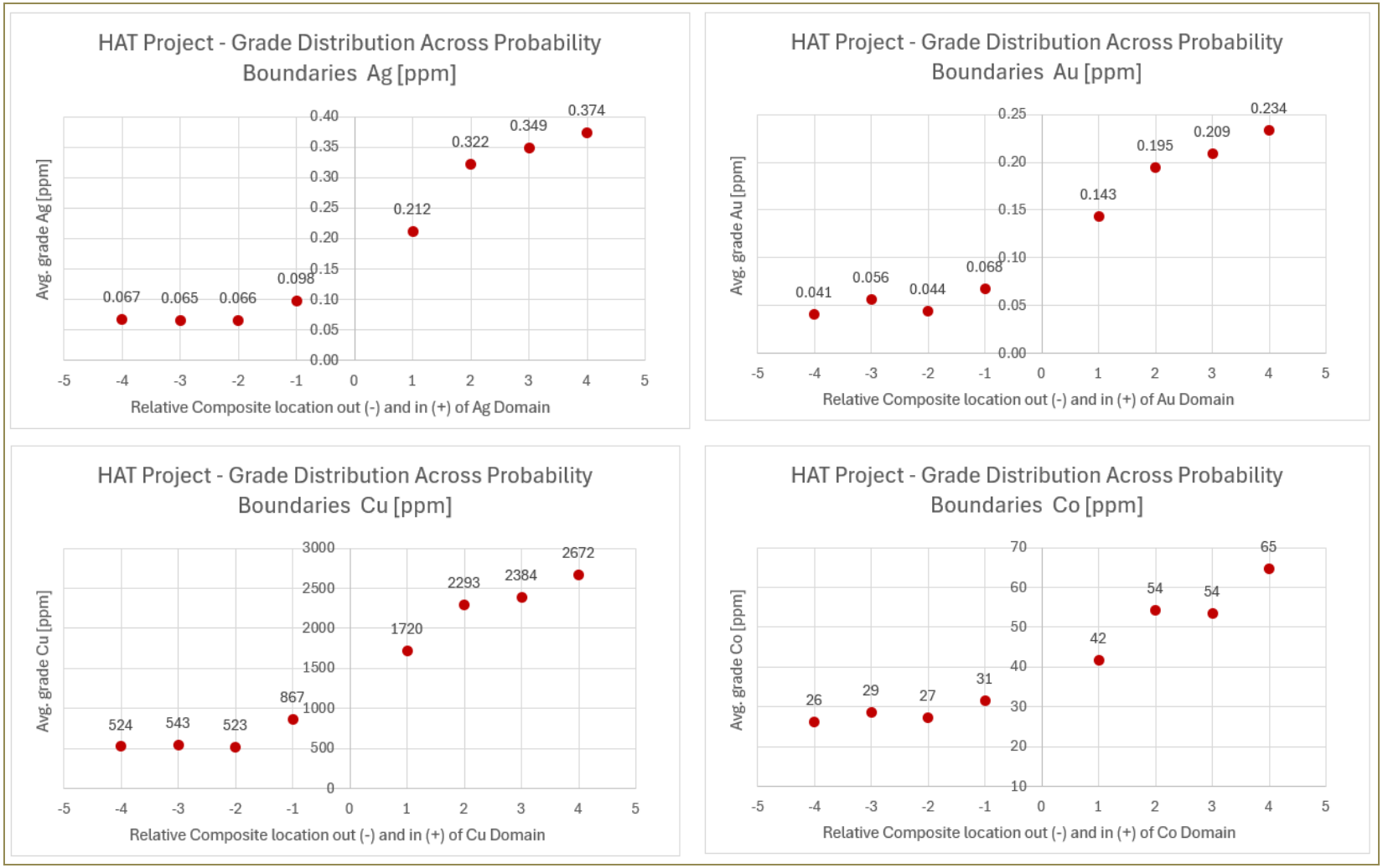


Figure 14-26: Grade Distribution at Probability Boundaries

14.9 Declustered Data for Each Domain

The spatial grade estimation in this PEA is based on conditional Sequential Conditional Gaussian Simulation (SCGS). To ensure that the transformation from real-grade space to Gaussian space is unbiased, the composite database must first be declustered. Without declustering, clustered drilling patterns can distort the underlying grade distribution and lead to misleading estimates of both global and local expected grades. For this reason, the database used for SCGS—and for all supporting statistical analyses—is treated as declustered.

Declustering was performed using ordinary kriging. The kriging weights assigned to each composite during the interpolation process were accumulated for every variable, producing a set of declustered weights associated with the original composite coordinates. These weights form the basis for unbiased statistical characterization and for the subsequent transformation required by the SCGS workflow.

The declustering process is summarized by the statistics in Table 14-9.

Table 14-9: Statistical Summary of Declustered Composites for Each Variable Used in the Grade Estimation Process

Variable	Sample No.	Min.	Max.	Average	Variance	SD	Skewness	Kurtosis
Au	1893	0.00	3.16	0.20	0.07	0.26	5.23	37.99
Ag	3058	0.01	5.75	0.43	0.31	0.56	4.00	23.23
Cu	1569	64.30	28,910	2,398	6,586,699.30	2,566.46	4.12	26.25
Co	3875	8.49	475.00	65.82	2,323.48	48.20	3.75	20.21
Sc	4664	3.48	61.00	29.43	53.93	7.34	0.17	0.12

14.10 Variography

Spatial variance and correlation were evaluated using both traditional variogram and correlogram formulas. The data were transformed into Gaussian normal space prior to analysis, and multiple directions and dipping angles were considered.

For gold, silver, copper, and cobalt the correlogram experimental points provided better representation of spatially structured ranges (Figure 14-27 to Figure 14-31) than the equivalent variogram results. All variables analyzed, apart from scandium, exhibit their longest ranges of spatial correlation between 150° and 160° SSE with a dipping angle of 30°. A second important axis of continuity is observed in the direction perpendicular to the main strike, trending upward at 60°. The direction perpendicular to these two axes represents the shortest continuity, trending approximately 70° ENE.

Scandium follows a different spatial continuity pattern compared to the other variables. The principal axis of continuity is oriented at 110° E with a steeper dipping angle of 50°. A secondary axis of continuity, consistent with the features observed in the other variables, is subvertical with the same azimuth of 110°, trending upward at 40°. The shortest continuity for scandium is observed to be 20° N.

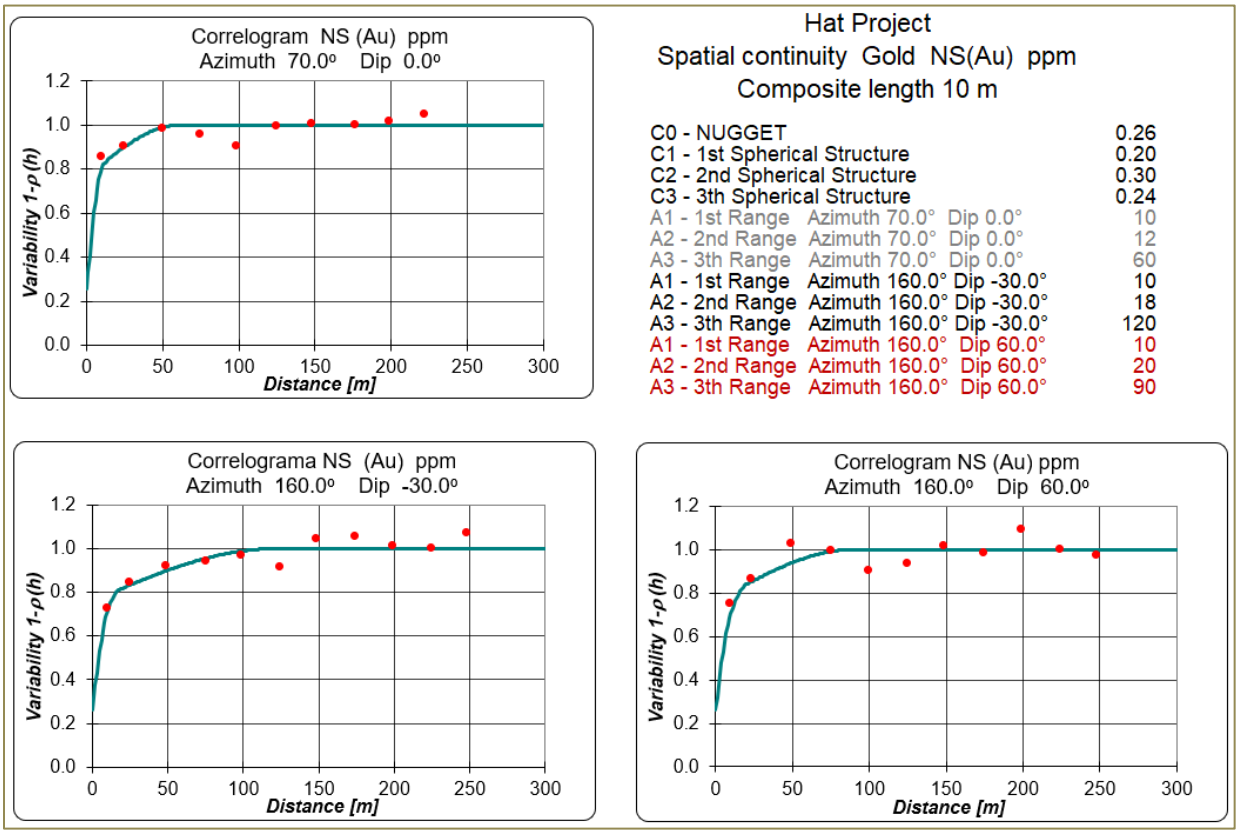


Figure 14-27: Correlogram Models and Experimental Points—Gold

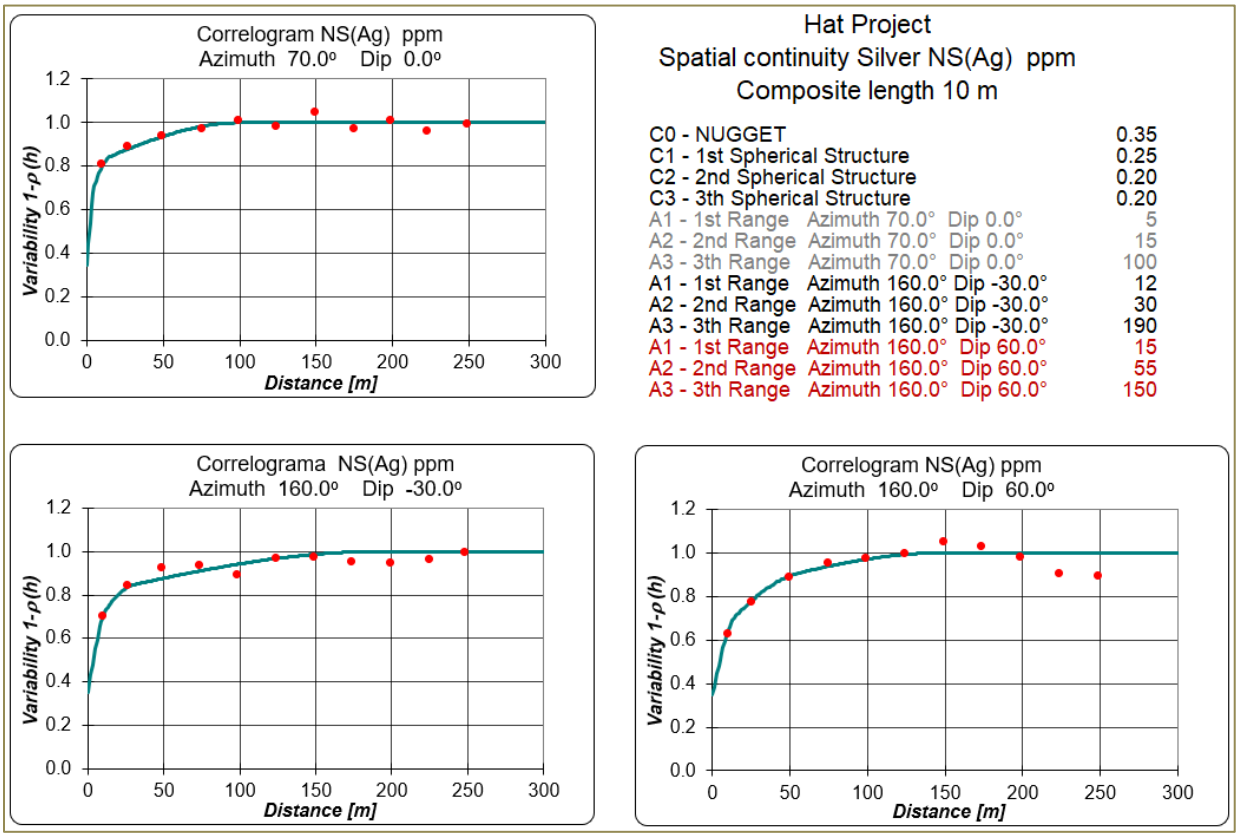


Figure 14-28: Correlogram Models and Experimental Points—Silver

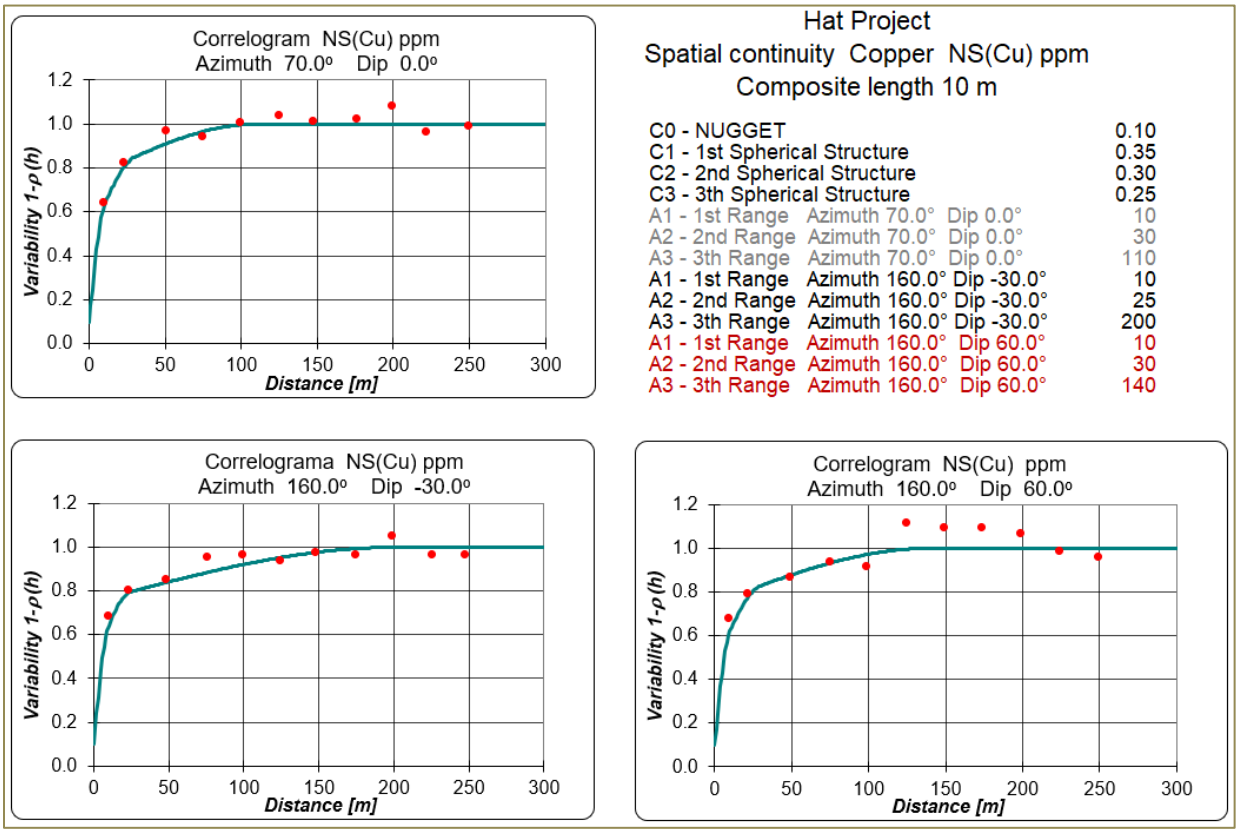


Figure 14-29: Correlogram Models and Experimental Points—Copper

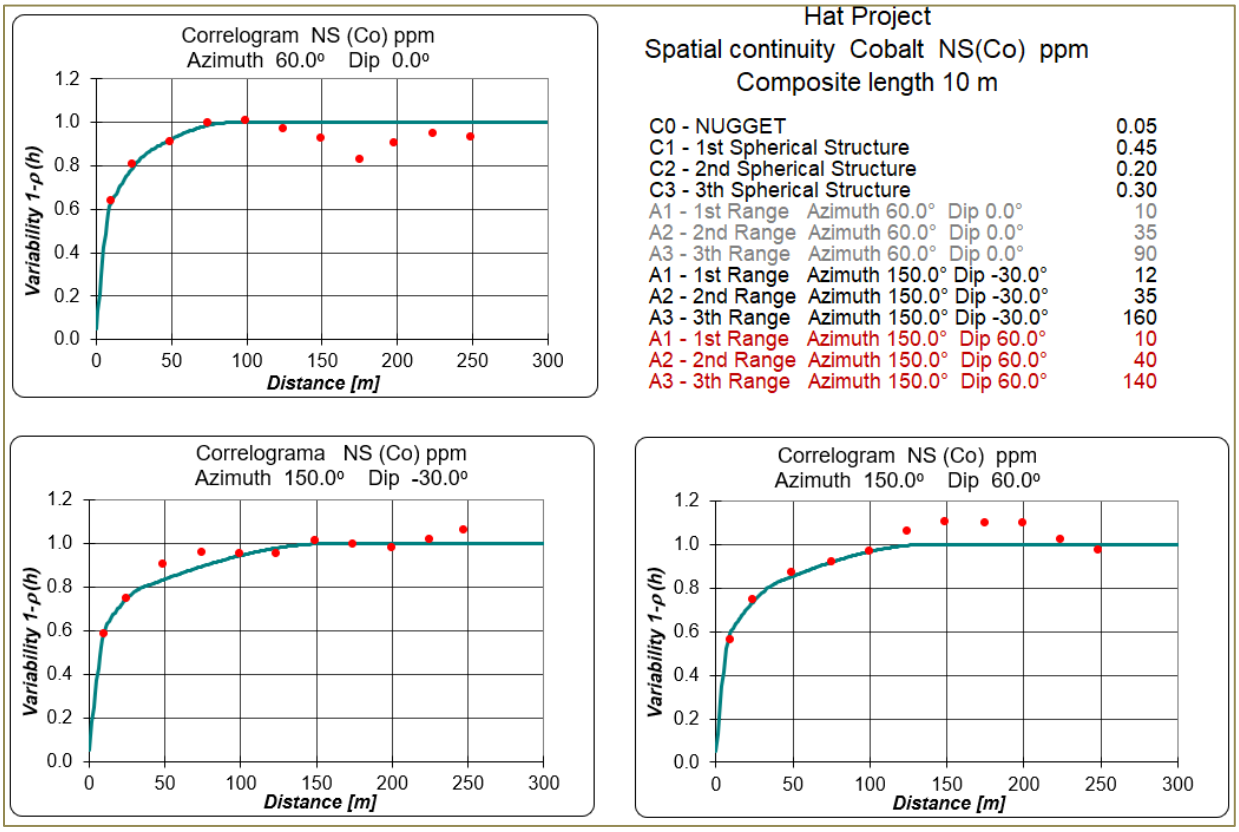


Figure 14-30: Correlogram Models and Experimental Points—Cobalt

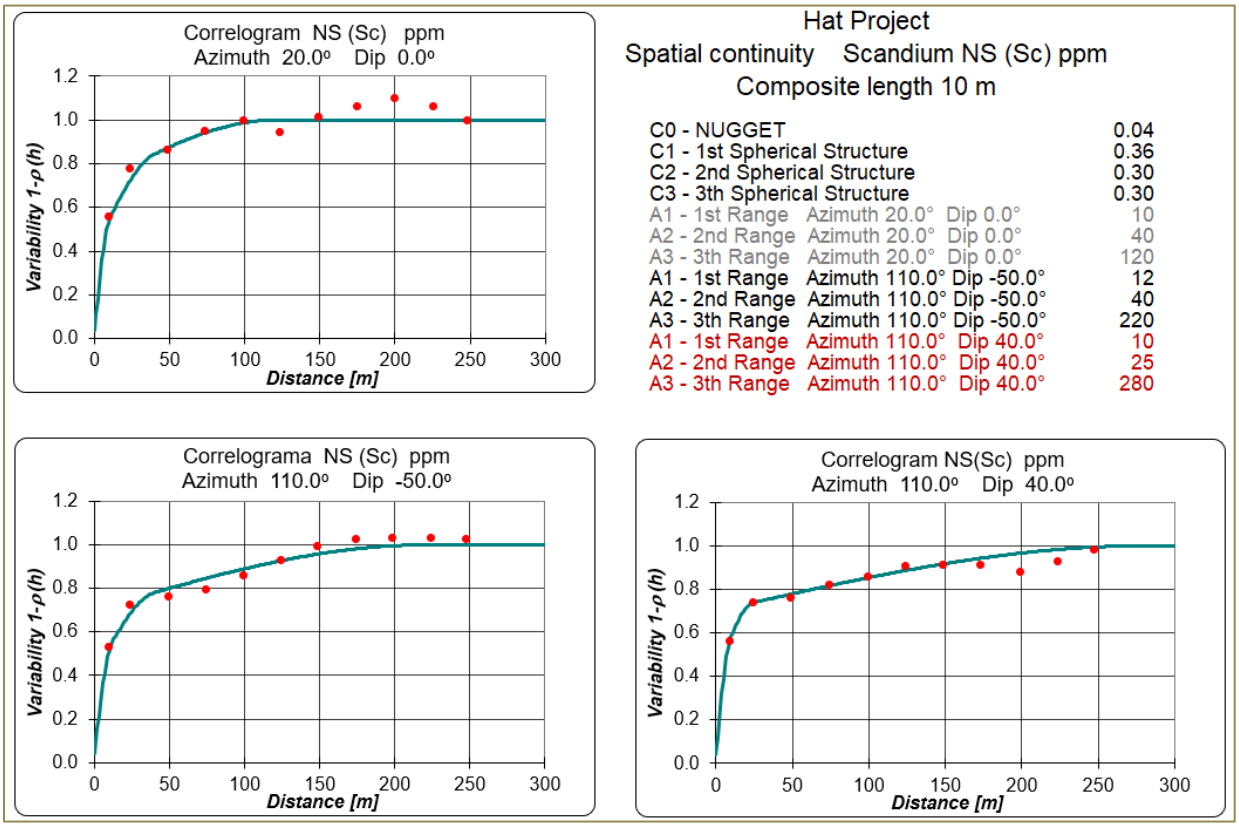


Figure 14-31: Correlogram Models and Experimental Points—Scandium

The nugget effect for each variable was assessed through the QA/QC program. For variance calculations, the results of field duplicates were utilized. A cut-off grade was applied to the duplicates to ensure correct representation of point variability within each domain.

The nugget effect serves as an introductory descriptor for each variable, indicating the inherent variability characteristics of the analyzed data (Table 14-10). These values reflect the influence of spatial correlation and are directly relevant to subsequent classification exercises. All spatial parameters used in correlogram models are normalized to 1.

Table 14-10: Nugget Effect by Variable

Variable	Nugget
Au	0.26
Ag	0.35
Cu	0.10
Co	0.05
Sc	0.04

14.11 Mineral Resource Estimation

This subsection outlines the geostatistical workflow used to generate the Mineral Resource estimates, including Gaussian transformation and Sequential Gaussian Simulation for both mineralized and background domains. The objective is to ensure consistent treatment of all domains and to capture realistic spatial variability in the final block model.

Table 14-11 presents the block model configuration.

Table 14-11: Block Model Prototype

	X	Y	Z
Model Origin (UTM Z9)	346750	6453000	0
Block Size (m)	15	15	15
Number of Blocks	136	150	75
Model Extents (m)	2,040	2,250	1,125
Rotation	0°	0°	0°

14.11.1 Gaussian Transformation and Sequential Gaussian Simulation for Mineralized and Background Domains

The databases for the five variables (Au, Ag, Cu, Co and Sc) were transformed into Gaussian distributions prior to simulation. Expected grades were then generated through 51 sequential conditional simulations, ensuring that both local variability and global grade distributions were honoured. The simulated nodes were subsequently assigned to the blocks corresponding to each mineralized domain, producing a set of realizations consistent with the probabilistic domain framework.

The same simulation workflow was applied to the domains representing background mineralization. These background domains encompass low-grade material and occupy the volume outside the mineralized shells. To maintain methodological consistency and avoid introducing bias between mineralized and non-mineralized volumes, the background grades were treated identically: the composite data were transformed into Gaussian space, and SCGS was performed using the same parameters and number of realizations.

This approach ensures that both mineralized and background domains are modelled using a unified geostatistical framework, allowing the final block model to reflect realistic spatial variability across the entire deposit volume.

The following figures summarize the parameters used in the SCGS process:

- Minimum number of composites—1
- Maximum number of composites—16
- Previously simulated nodes—12
- Search radius—full-radius search applied.

Table 14-12: Summary of Numerical Parameters for Grade Simulation

	Spherical	Axis Y (m)	Axis X (m)	Axis Z (m)
Gold				
C0	0.26	1st Rot. 160° by Z	2nd Rot. −30° by X	
C1	0.20	10	10	10
C2	0.30	18	12	20
C3	0.24	120	60	90
Search		180	110	140
Silver				
C0	0.35	1st Rot. 160° by Z	2nd Rot. −30° by X	
C1	0.25	12	5	15
C2	0.20	30	15	55
C3	0.20	190	100	150
Search		240	150	200
Copper				
C0	0.10	1st Rot. 160° by Z	2nd Rot. −30° by X	
C1	0.35	10	10	10
C2	0.30	25	30	30
C3	0.25	200	110	140
Search		250	160	190
Cobalt				
C0	0.05	1st Rotation 150° by Z	2nd Rotation −30° by X	
C1	0.45	12	10	10
C2	0.20	35	35	40
C3	0.30	160	90	140
Search		210	140	190
Scandium				
C0	0.04	1st Rotation 110° by Z	2nd Rotation −50° by X	
C1	0.36	12	10	10
C2	0.30	40	40	25
C3	0.30	220	120	280
Search		250	150	300

Notes: Rotation about the Z-axis is applied clockwise when viewed from above.
Rotation about the X-axis is applied clockwise when viewed from the positive X direction, resulting in a downward rotation.

14.12 Validation Estimate

14.12.1 Validation of Conditional Simulations

The global validation of the 51-realization simulation set consists of comparing the conditionally simulated results against the input data using the cumulative distribution function, as shown in Figure 14-32. This approach confirms that the simulations reproduce the statistical characteristics of the original dataset.

By validating the simulations against the input data, the exercise demonstrates that the simulation model is statistically consistent and appropriate for deriving expected grades (E-type) and quantifying local grade uncertainty. The conditional simulations honour the observed distributions and provide a defensible representation of the plausible range of grade outcomes, strengthening the robustness of the Mineral Resource statement.

This validation exercise demonstrates that:

- The simulations reproduce the input data distribution without introducing bias.
- The variability and uncertainty of grades are properly captured and support the transition from point support to block support.

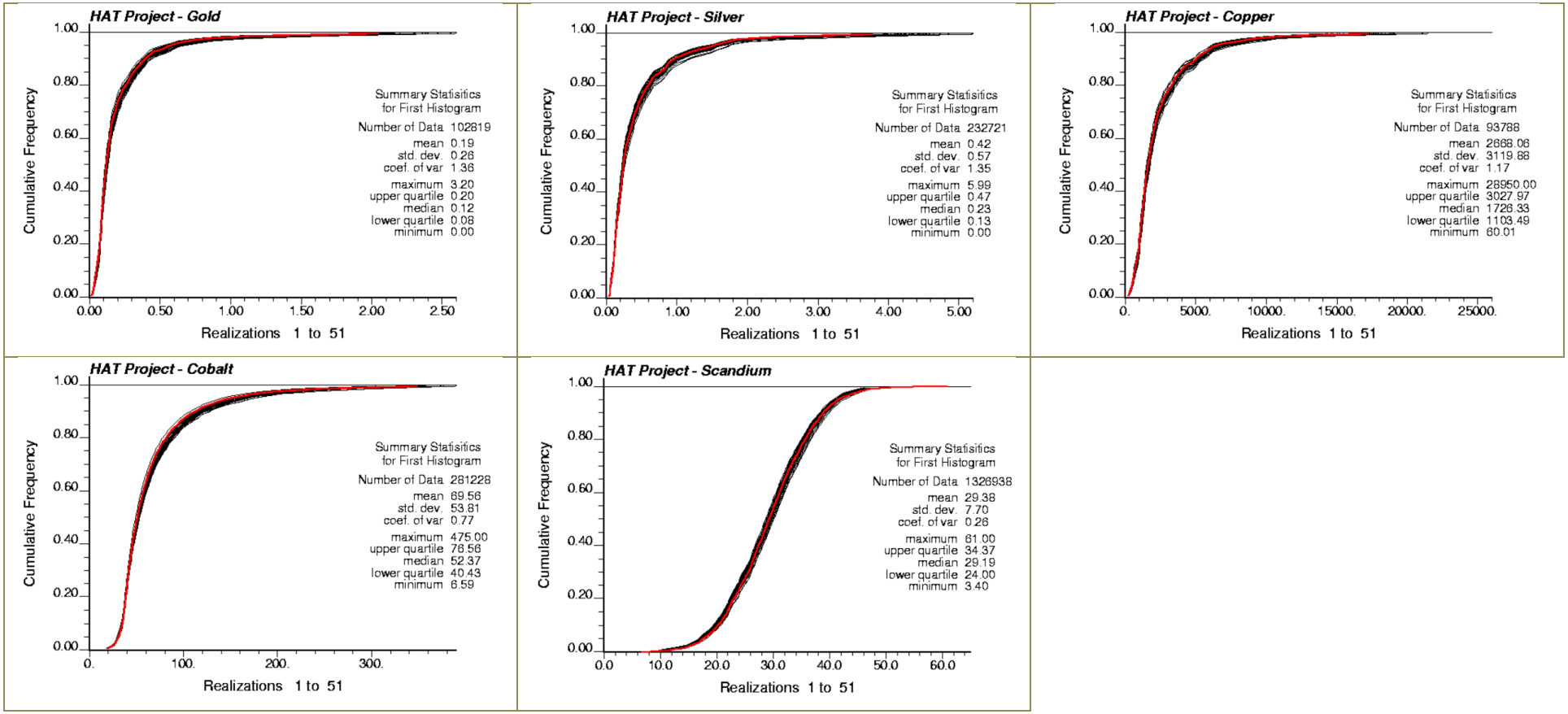


Figure 14-32: Validation of Successful Reproduction of 51 Simulations of Input Data Red Line

14.12.2 E-Type Grade Derivation and Change of Support

The expected grade (E-type) model was derived as the arithmetic average of 51 conditional simulations generated at 15 × 15 × 15 m block support. This averaging process inherently applies a change of support from point-scale composite data to block-scale estimates (Rossi & Deutsch, 2013).

As the support increases, the grade distribution undergoes the expected smoothing effect: the high-grade and low-grade tails are shortened, and the overall variability is reduced. This behaviour reflects the transition from point variability to block variability, where local extremes are moderated by the volumetric averaging imposed by the block size.

The resulting E-type model therefore represents the most probable grade outcome at block support and provides a statistically coherent basis for subsequent uncertainty analysis and Mineral Resource evaluation.

The change of support from point-scale composites to block-scale grades introduces a predictable smoothing effect that reduces the variance of the block grade distribution. This behaviour is quantified through the variance retention factor and variance removed factor, defined as

Variance retained = $F_{VR} = \sigma^2_block / \sigma^2_sample$

Where:

σ^2_block = variance at block support

σ^2_sample = variance at point (composite) support

F_{VR} = proportion of variance preserved after change of support.

Variance removed = $F_{VRM} = 1 - (\sigma^2_block / \sigma^2_sample)$

Where:

F_{VRM} = proportion of variance lost due to smoothing

σ^2_block = block variance

σ^2_sample = sample variance.

Values of variance retention close to 100% indicate minimal smoothing, whereas lower values reflect progressively stronger variance reduction. In the present case (Table 14-13), the block variance represents only a small fraction of the point-scale variance, meaning that a major proportion of the original variability is removed during regularization. The variance removed values show that the block estimates for gold, silver, copper, cobalt, and scandium have reduced variances by at least 85%. This reduction in variance moderates high grades and elevates low grades, resulting in a narrower and less-skewed block-scale distribution. The effect of smoothing is shown in the dispersion plots: the point-scale comparison in Figure 14-33, Figure 14-35, Figure 14-37, Figure 14-39, and Figure 14-41 exhibit a symmetric cloud with a regression slope near unity, whereas the block-scale comparison in Figure 14-34, Figure 14-36, Figure 14-38, Figure 14-40, and Figure 14-42 show a horizontally compressed cloud and a regression slope

greater than one. This steepening of the regression line is a geometric consequence of the reduced block variance and confirms the expected change-of-support behaviour.

Table 14-13: Comparison of Sample-Scale and Block-Scale Variance Metrics and the Resulting Steepening of the Dispersion-Cloud Regression Line for Au, Ag, Cu, Co, and Sc

Parameter	Au	Ag	Cu	Co	Sc
Block variance (σ²_block)	0.006	0.046	635700	350.3	6.7
Sample variance (σ²_sample)	0.070	0.312	6587000	2323.0	53.9
Variance retention factor F_{VR} (%)	7.93	14.74	10	15.1	12.4
Variance removed F_{VRM} (%)	92.07	85.26	90	84.9	87.6
Regression slope (point scale)	1.00	0.98	0.93	0.97	1.08
Regression slope (block scale)	1.52	1.40	1.37	1.23	1.21
Increase in slope (%)	52	43	47	27	12

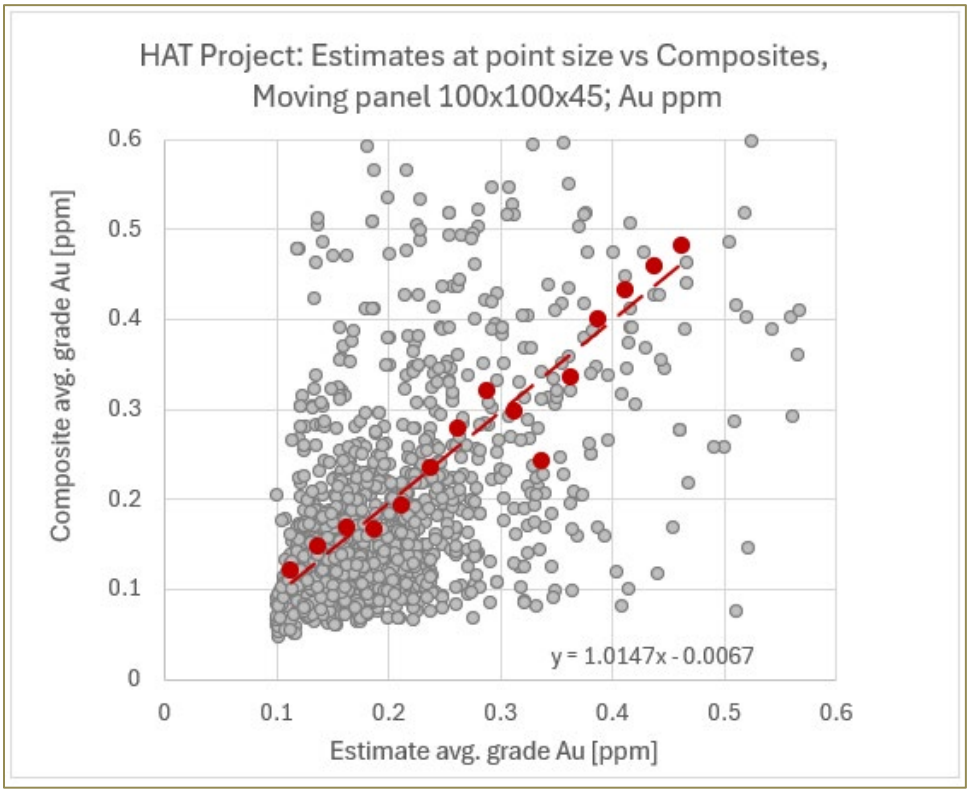


Figure 14-33: Gold Point Dispersion Distribution with Regression Line Showing Unbiased Estimates

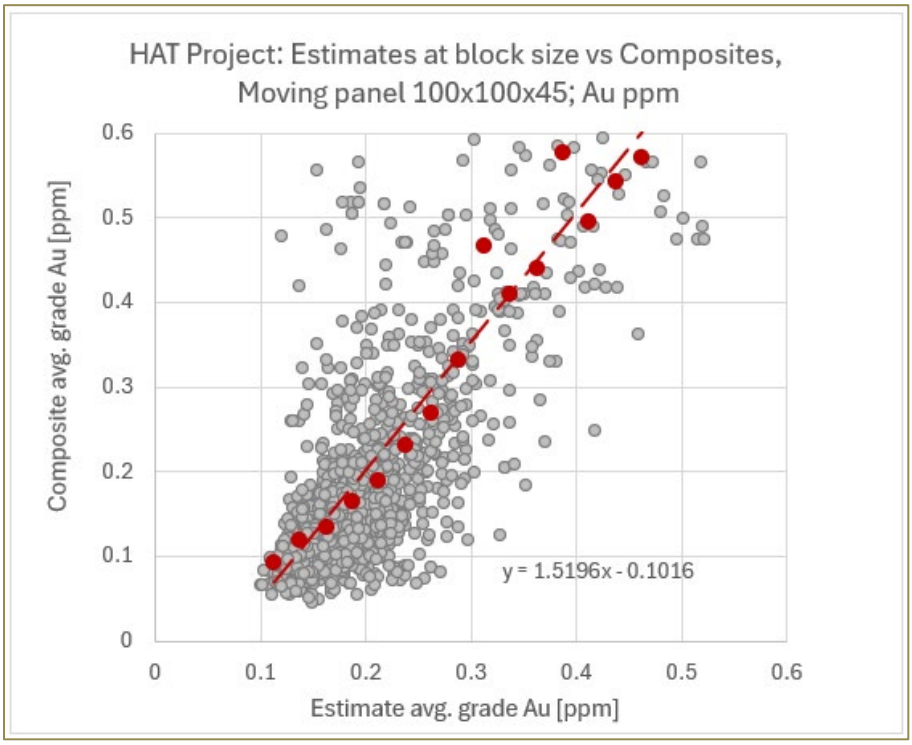


Figure 14-34: Gold Block Dispersion Distribution with Regression Line Showing Unbiased Estimates

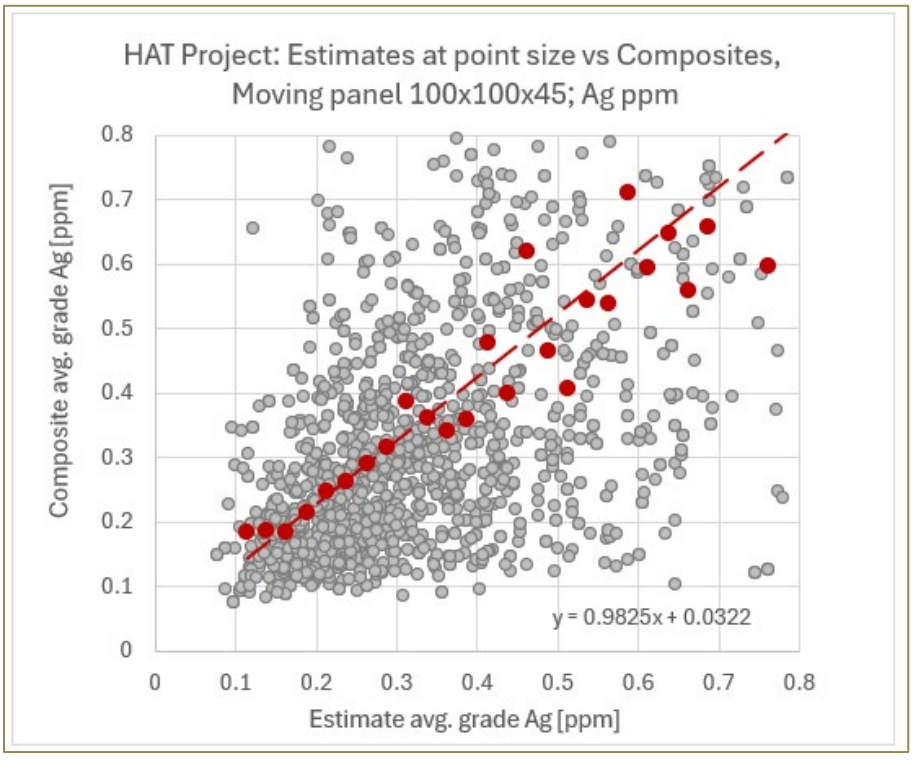


Figure 14-35: Silver Point Dispersion Distribution with Regression Line Showing Unbiased Estimates

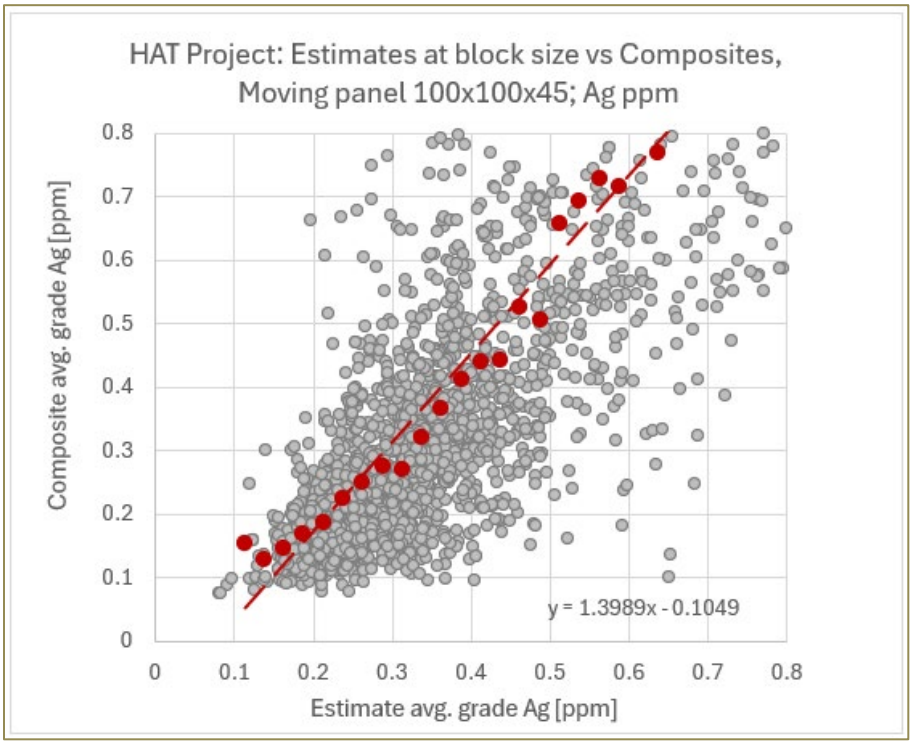


Figure 14-36: Silver Block Dispersion Distribution with Regression Line Showing Unbiased Estimates

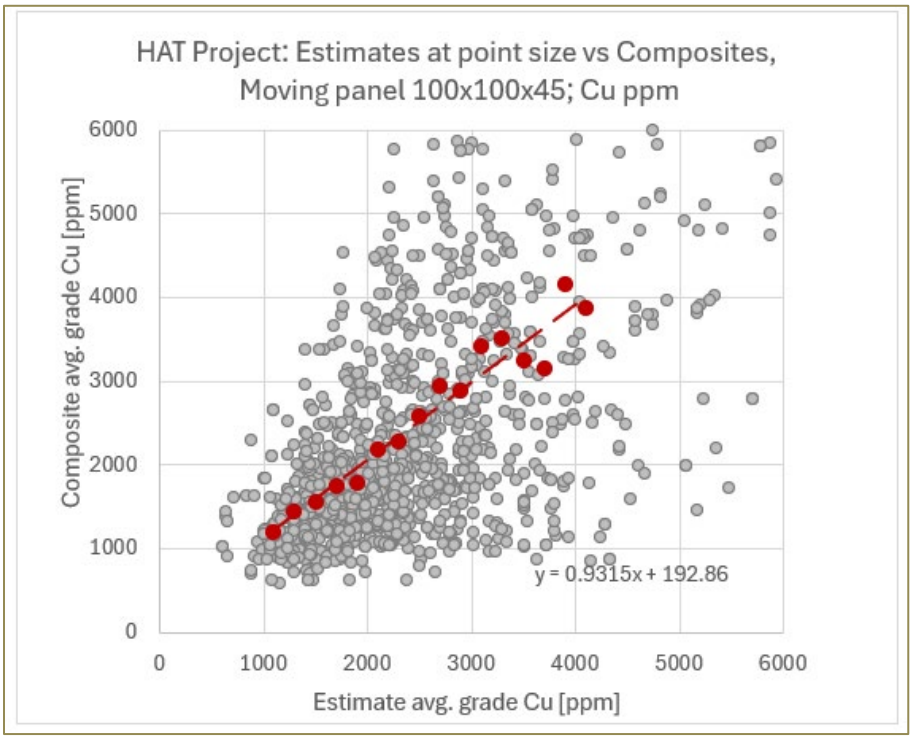


Figure 14-37: Copper Point Dispersion Distribution with Regression Line Showing Unbiased Estimates

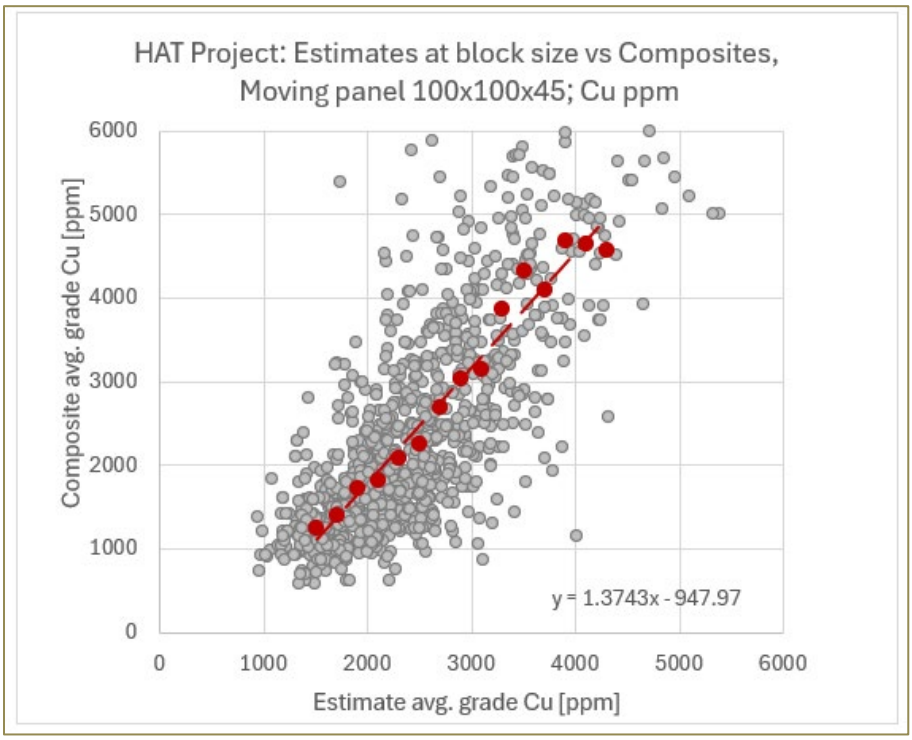


Figure 14-38: Copper Block Dispersion Distribution with Regression Line Showing Unbiased Estimates

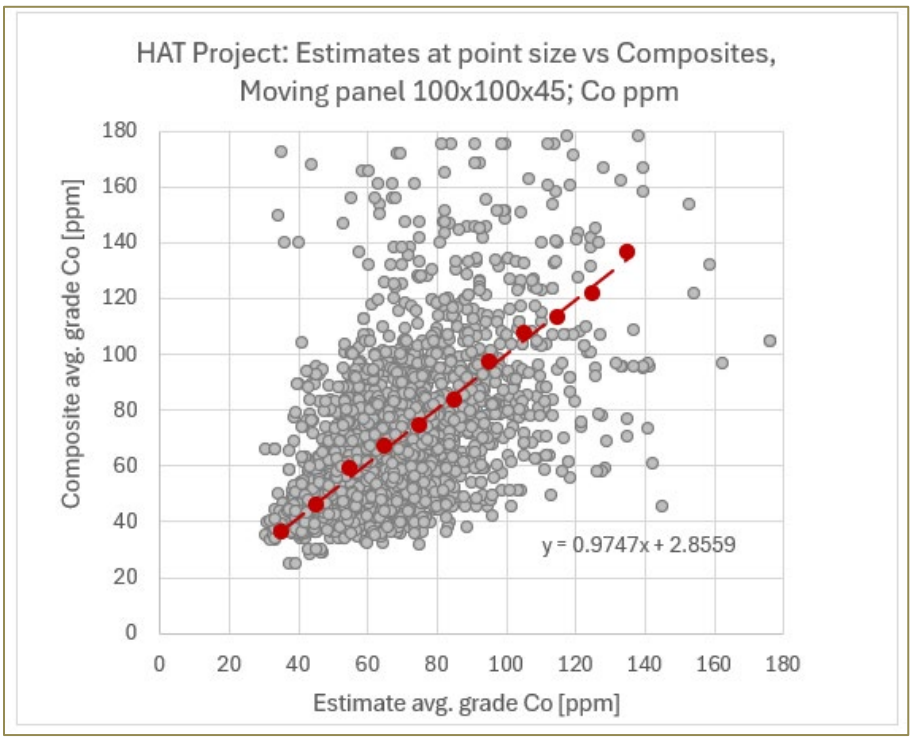


Figure 14-39: Cobalt Point Dispersion Distribution with Regression Line Showing Unbiased Estimates

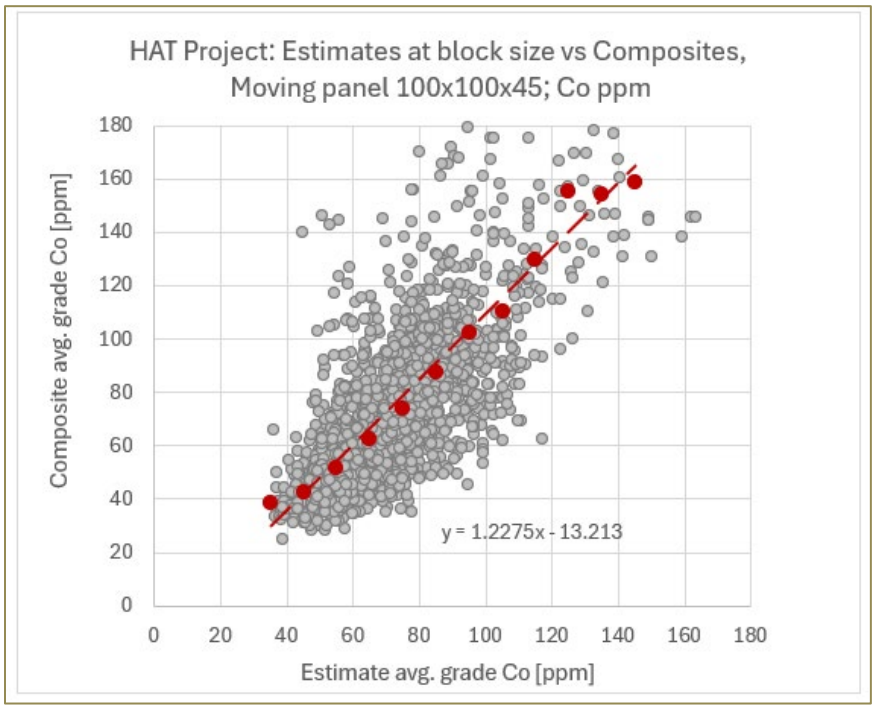


Figure 14-40: Cobalt Block Dispersion Distribution with Regression Line Showing Unbiased Estimates

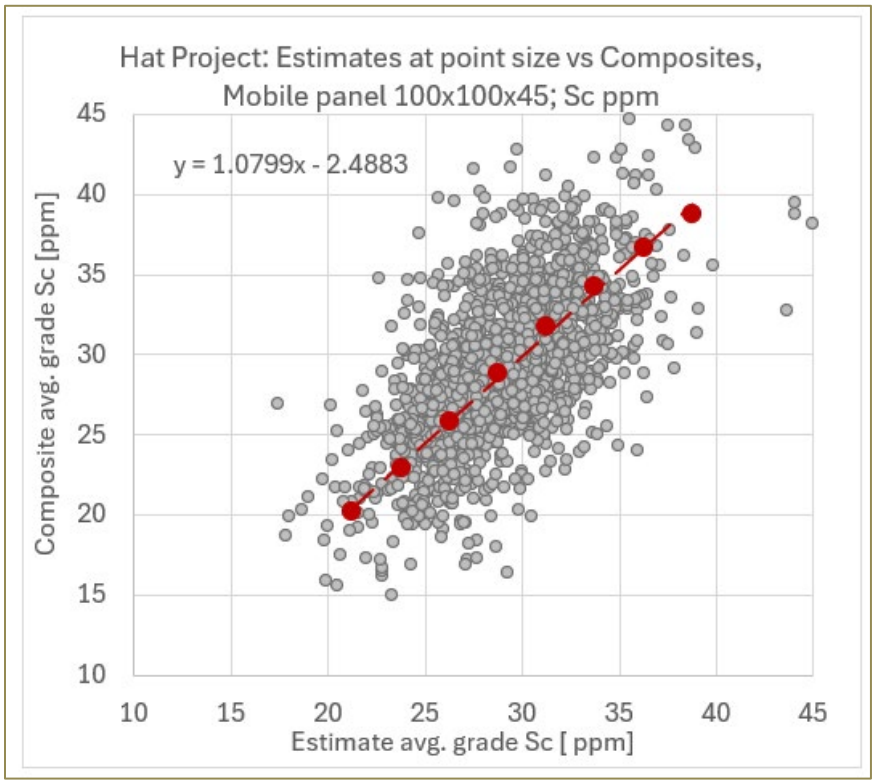


Figure 14-41: Scandium Point Dispersion Distribution with Regression Line Showing Unbiased Estimates

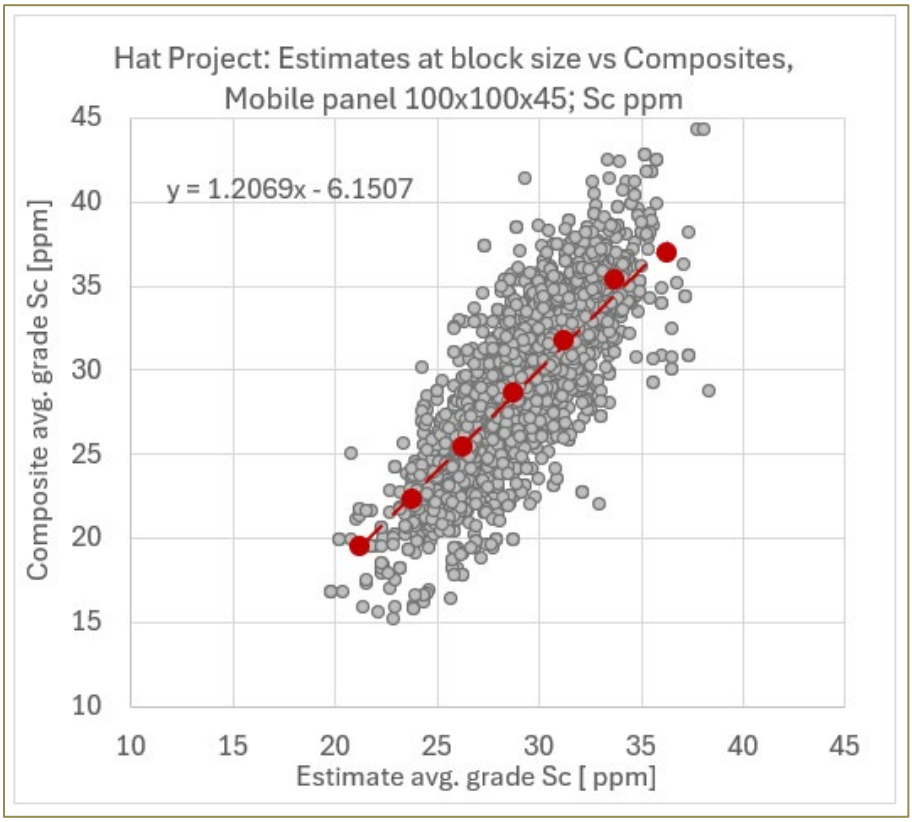


Figure 14-42: Scandium Block Dispersion Distribution with Regression Line Showing Unbiased Estimates

14.12.3 Average Grade Distribution Along, Across, and With Depth

Swath plot validation—the comparison of sample and block averages computed within successive slices—is a widely used and efficient technique under certain conditions. However, when data density is limited and the estimated volume spans large distances, long slices (sometimes up to 1,500 m) may incorporate drill holes and block estimates that are not always locally influenced by the data captured within a given swath. As Murphy (2019) noted, swath plots are often interpreted without considering the spatial distribution of samples and blocks, meaning that block estimates within a slice may be informed by samples located outside the slice boundaries. This behaviour is inherent to geostatistical interpolation: as Ortiz (2024) explains, block estimates are influenced by all samples within the search neighbourhood, which typically extends well beyond the thickness of the swath. Consequently, swath plots must be interpreted with an understanding of these spatial limitations. Nevertheless, swath comparisons remain necessary to support and validate the estimates. In this section, standardized validation figures (Figure 14-43 to Figure 14-47) are presented to illustrate the average grade distribution of gold, silver, copper, cobalt, and scandium along strike, across strike, and with depth. These figures provide confidence that the block model reproduces the input data trends at appropriate scales.

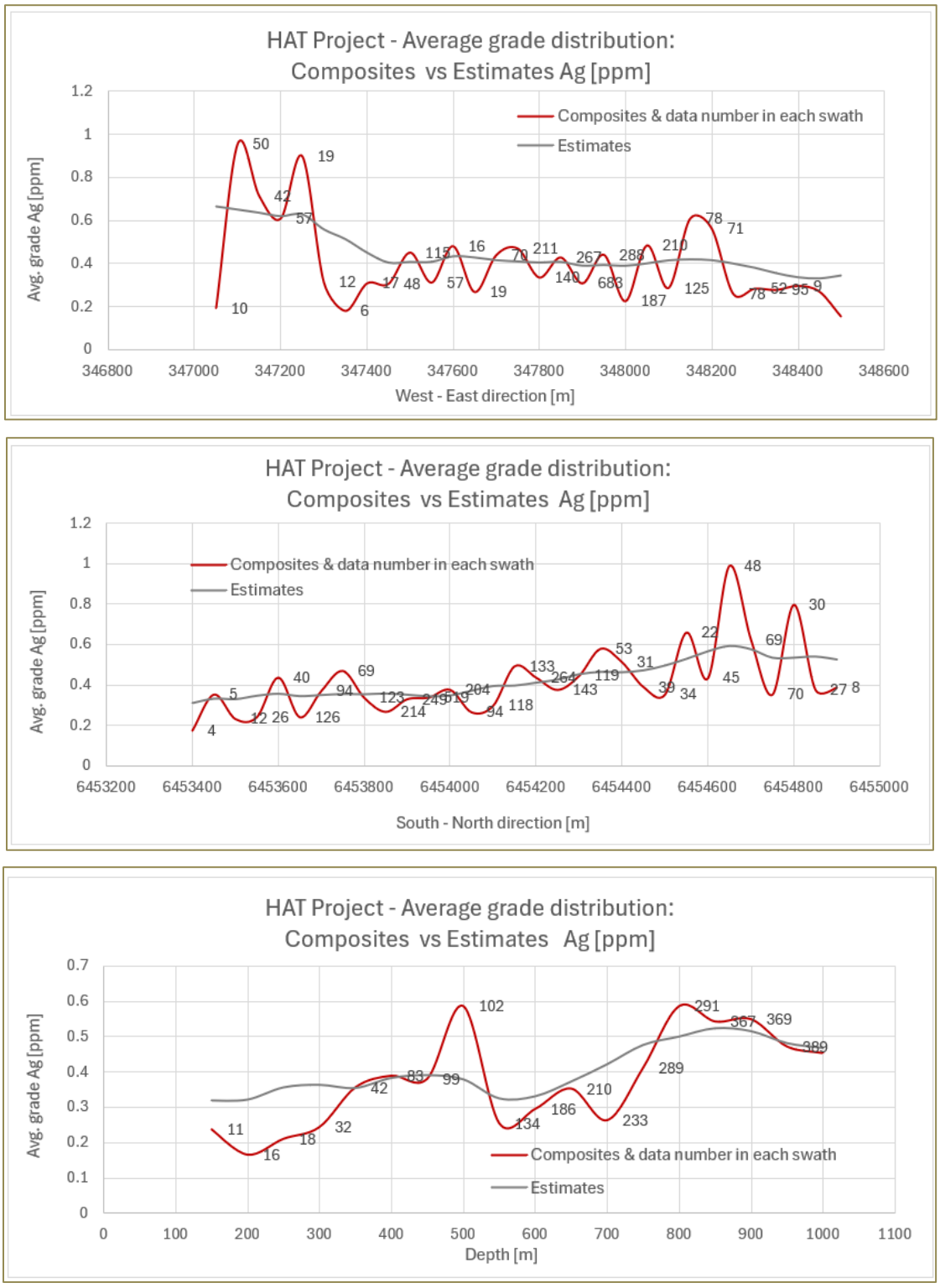
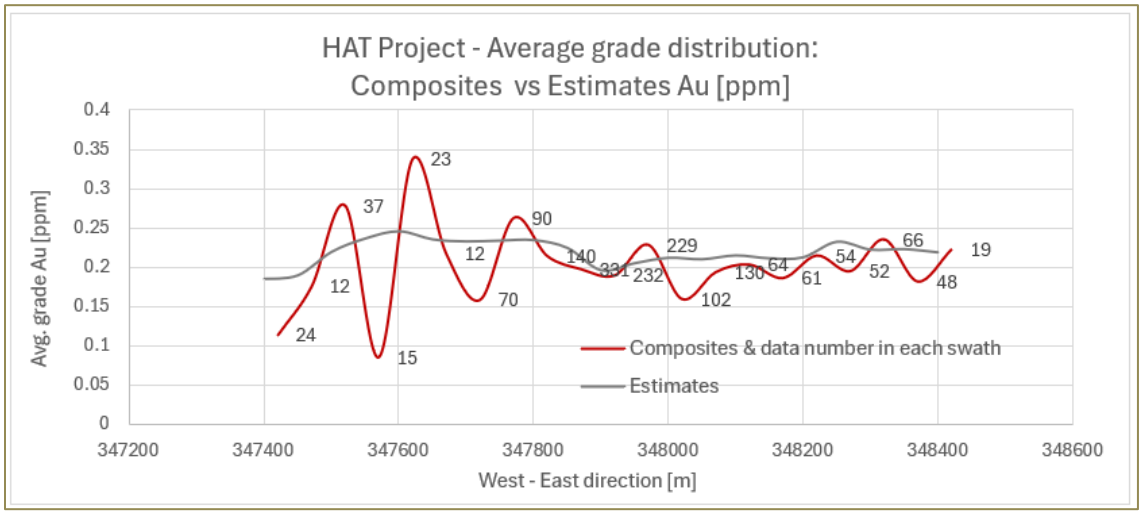
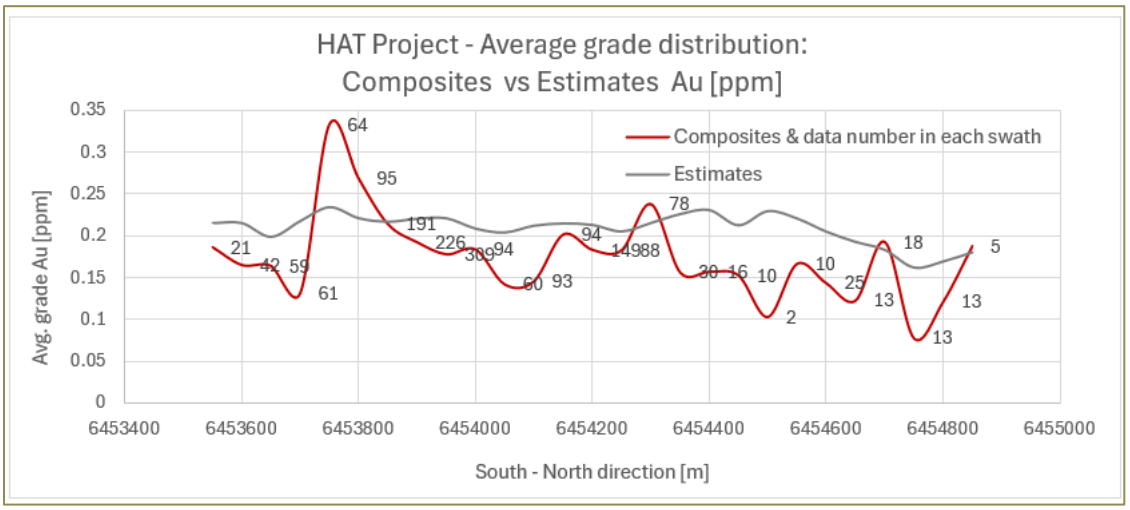


Figure 14-43: Silver Grade Validation—Swath Plot





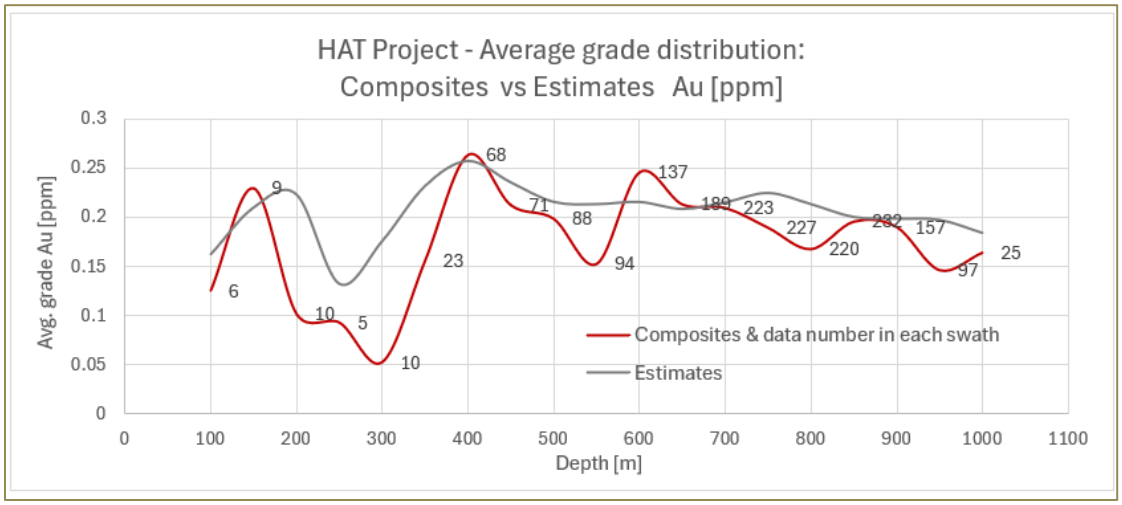
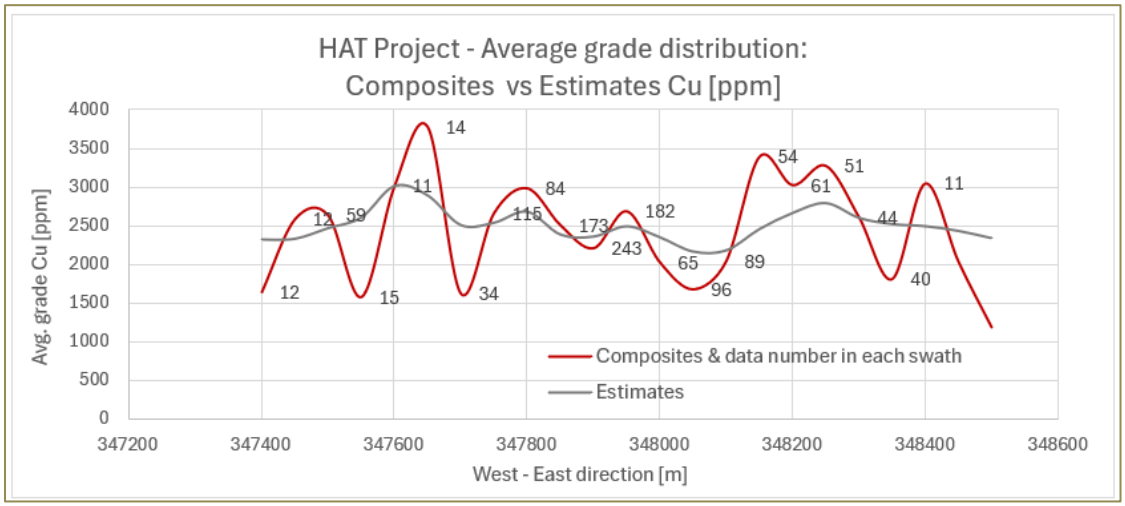
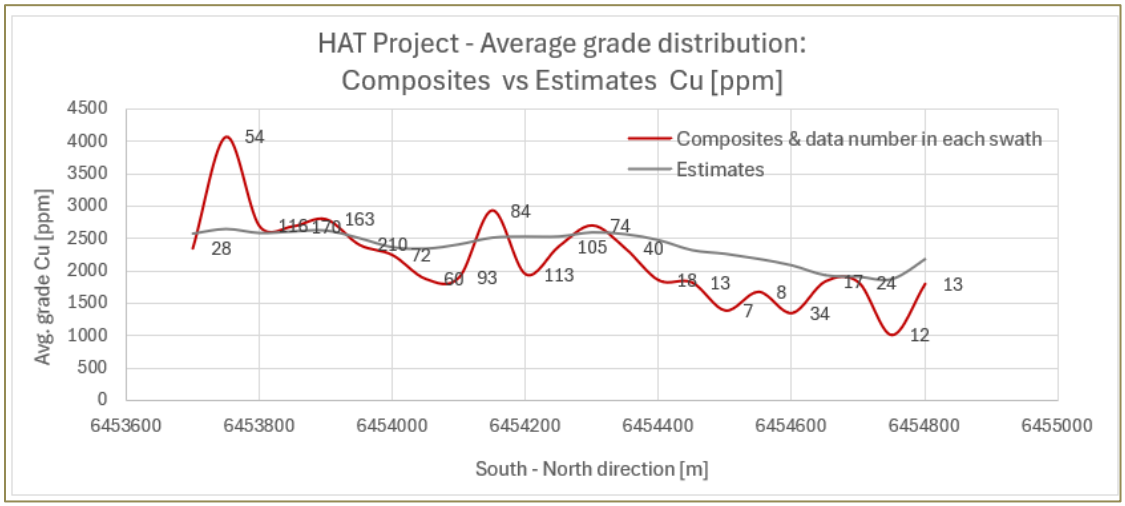


Figure 14-44: Gold Grade Validation—Swath Plot





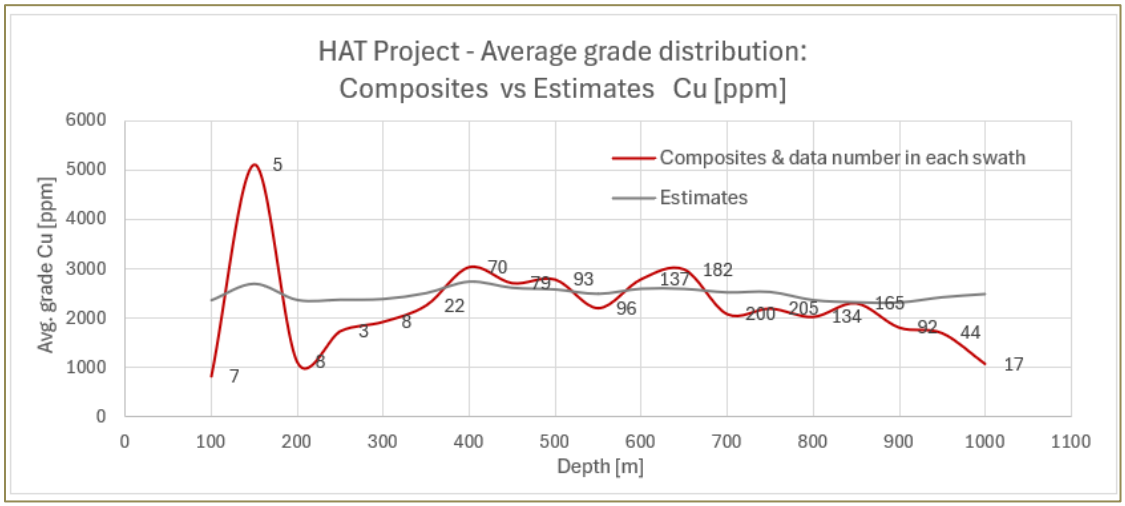
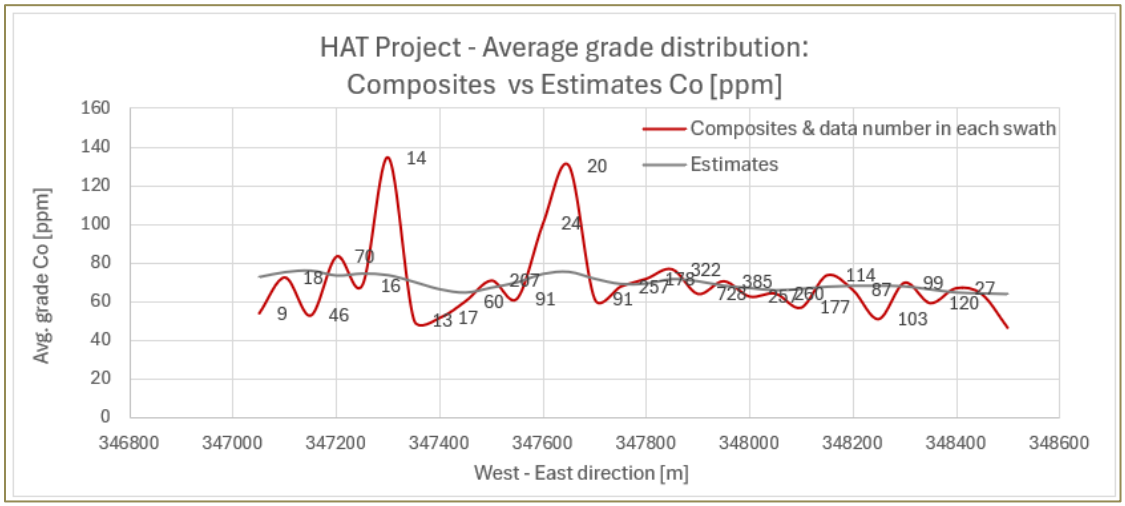
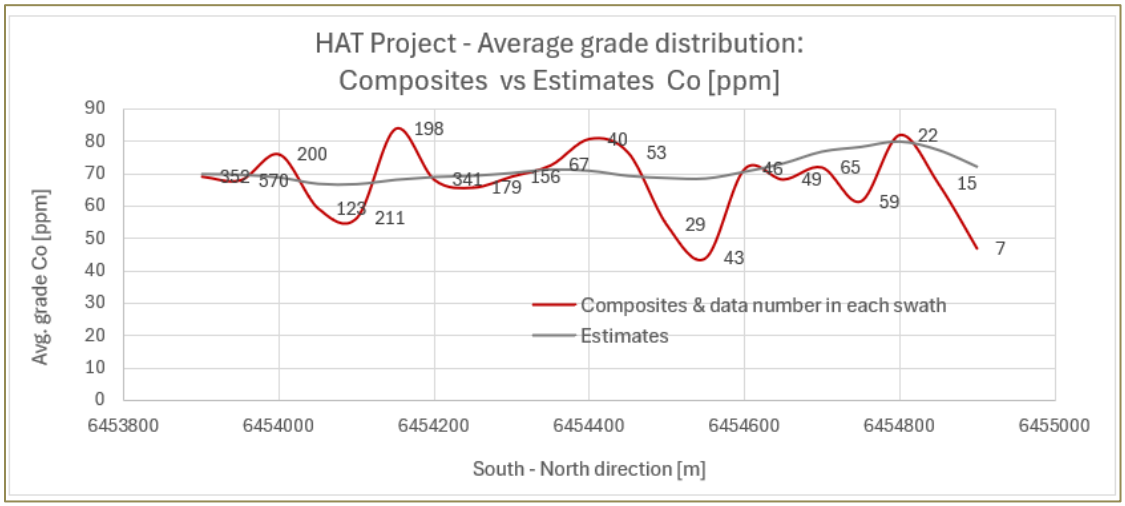


Figure 14-45: Copper Grade Validation—Swath Plot





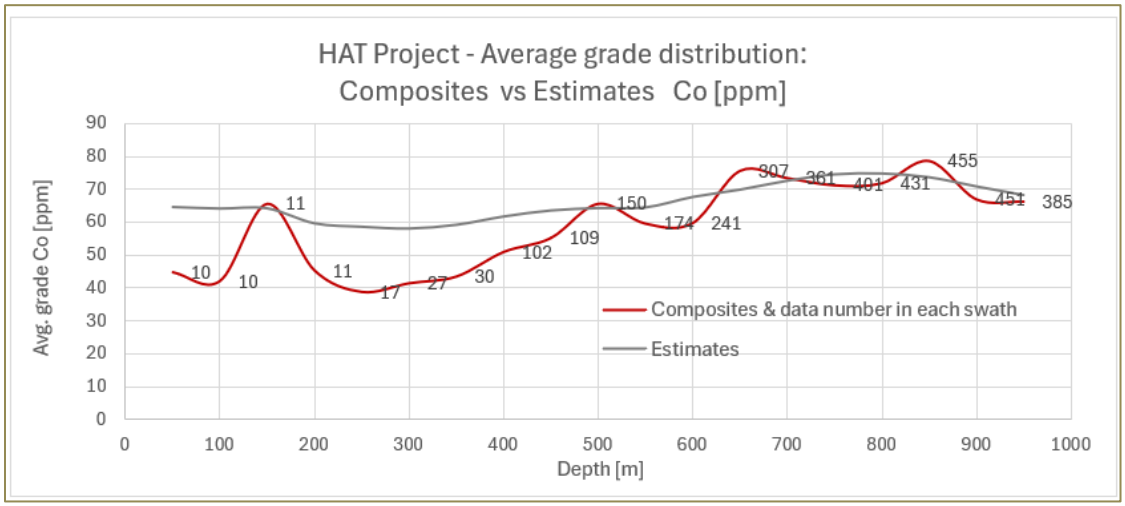
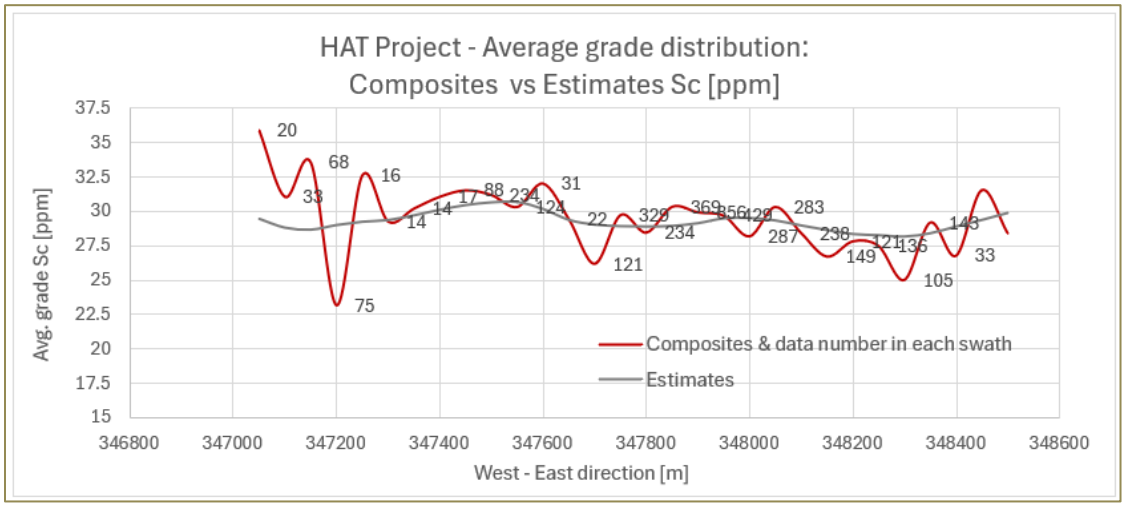
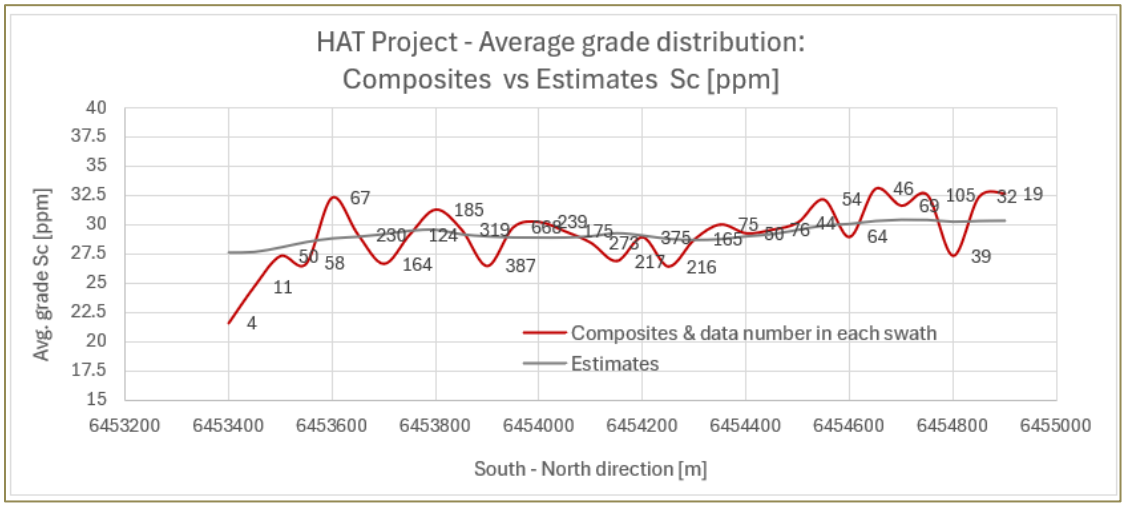


Figure 14-46: Cobalt Grade Validation—Swath Plot





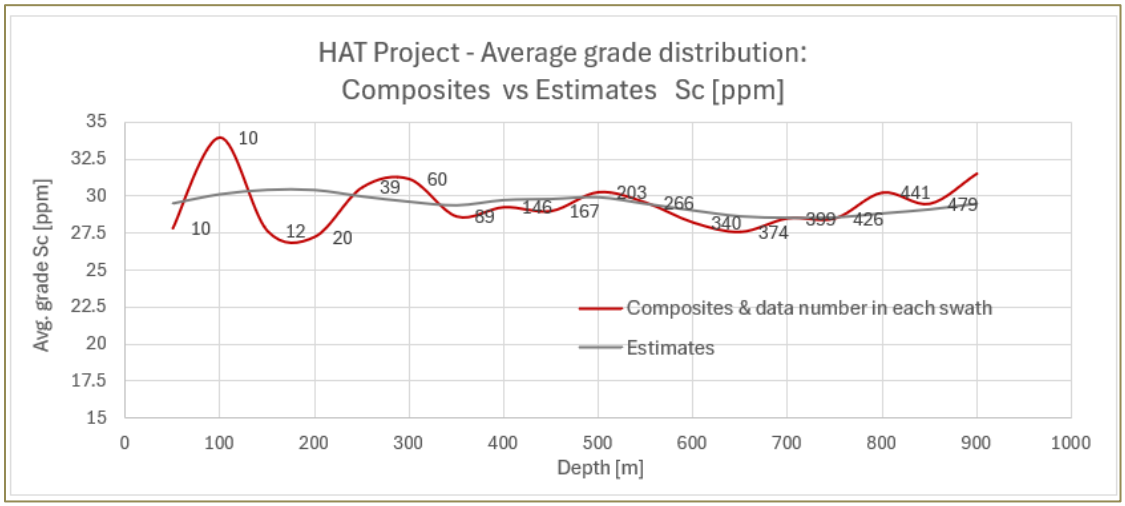


Figure 14-47: Scandium Grade Validation—Swatch Plot

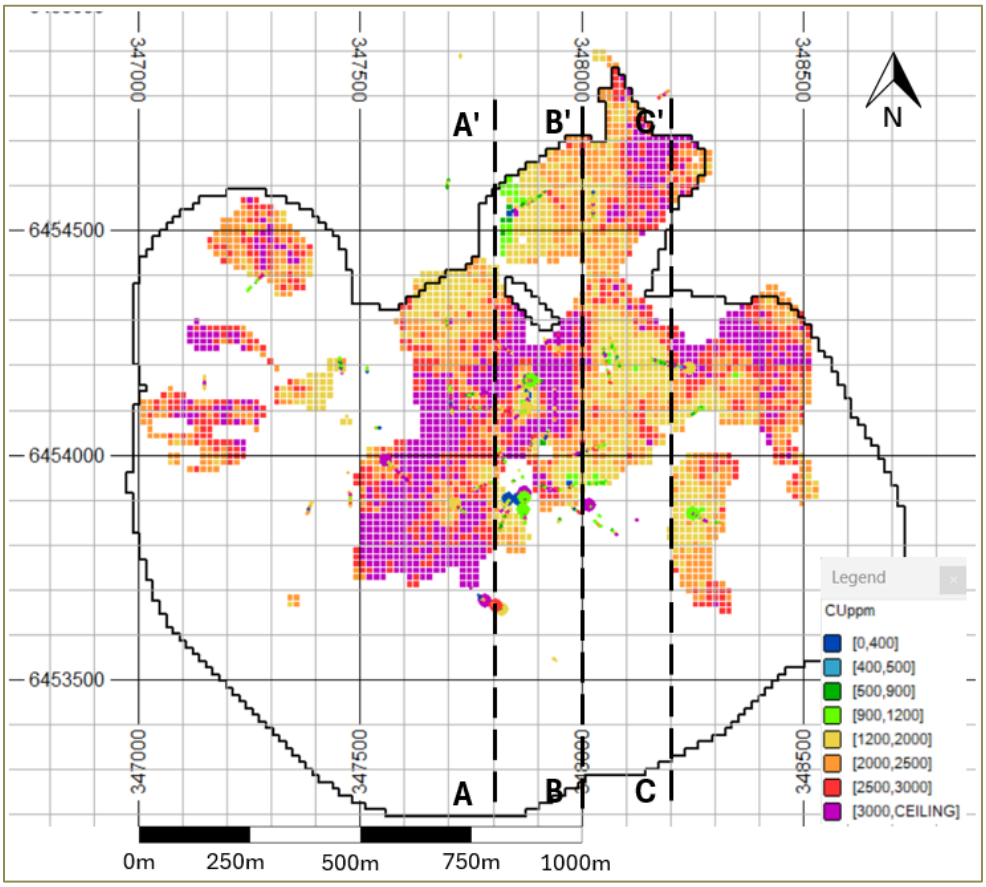
Swath Plot Validation

Average grade validation through swath plots demonstrates a reasonable reproduction of input data, with the already discussed in Section 14.12.2 smoothing effect attributable to block size.

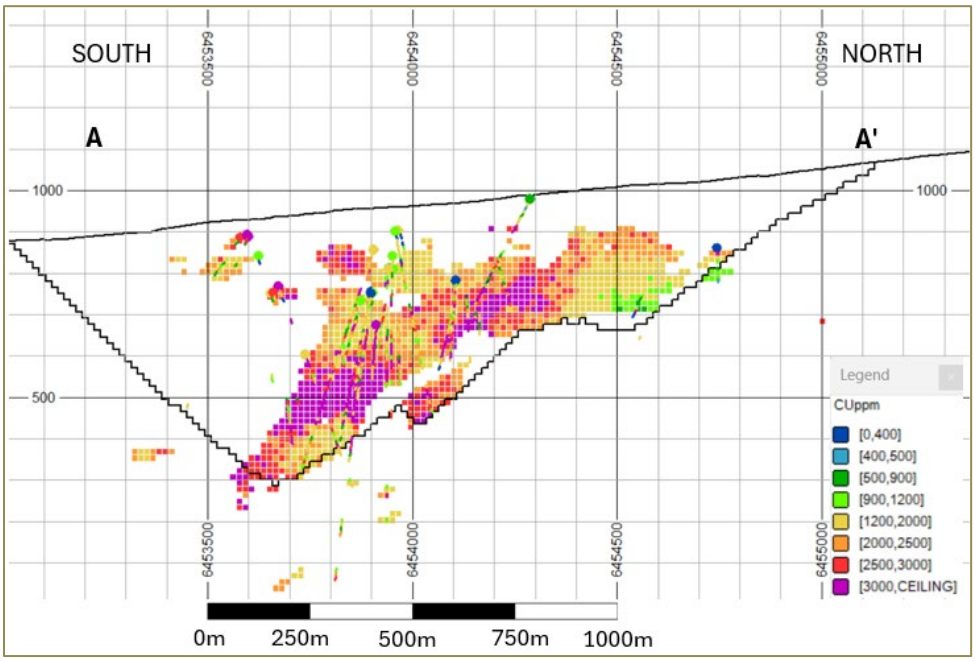
In the south–north direction, certain intervals with limited data density do not exert sufficient spatial influence on the block estimates. This limitation highlights that swath plots, while useful, may not always capture local variability when data are sparse or unevenly distributed.

It is important to emphasize that the overall reproduction is considered appropriate. The spatial effect on average grades arises from the underlying spatial correlation model, which incorporates samples that are not directly reflected in each swath plot. As such, swath plots should be interpreted as a validation tool within the broader context of geostatistical modelling, rather than as a standalone measure of accuracy.

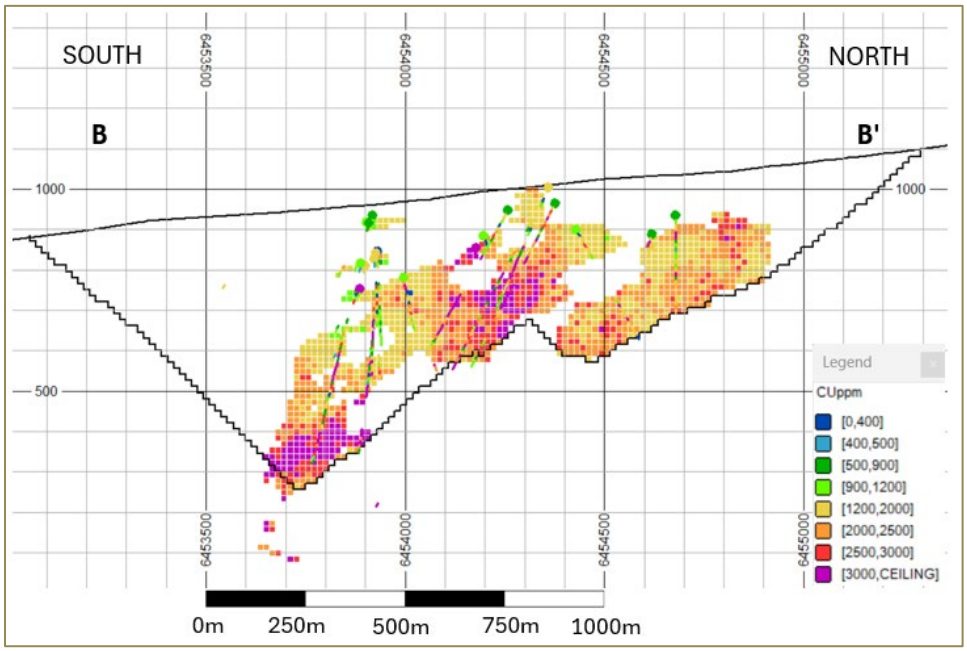
Figure 14-48 to Figure 14-53 presents examples of the spatial distribution of estimates by variable.



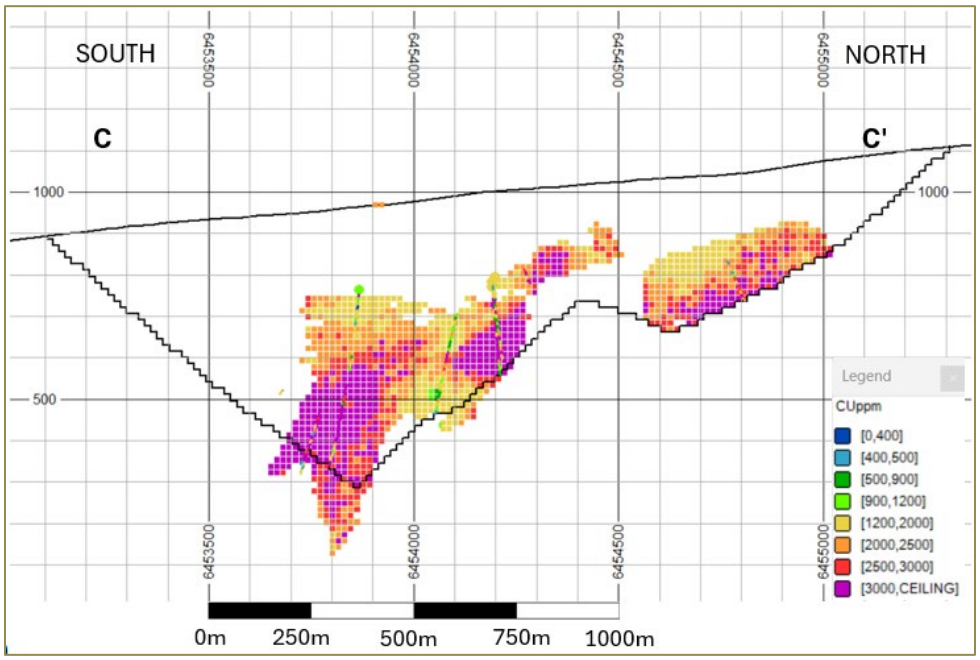
Plan View Elevation 700 masl; Cu (ppm) grade distribution within the modelled domain



East 347800 looking West; Cu (ppm) grade distribution within the modelled domain

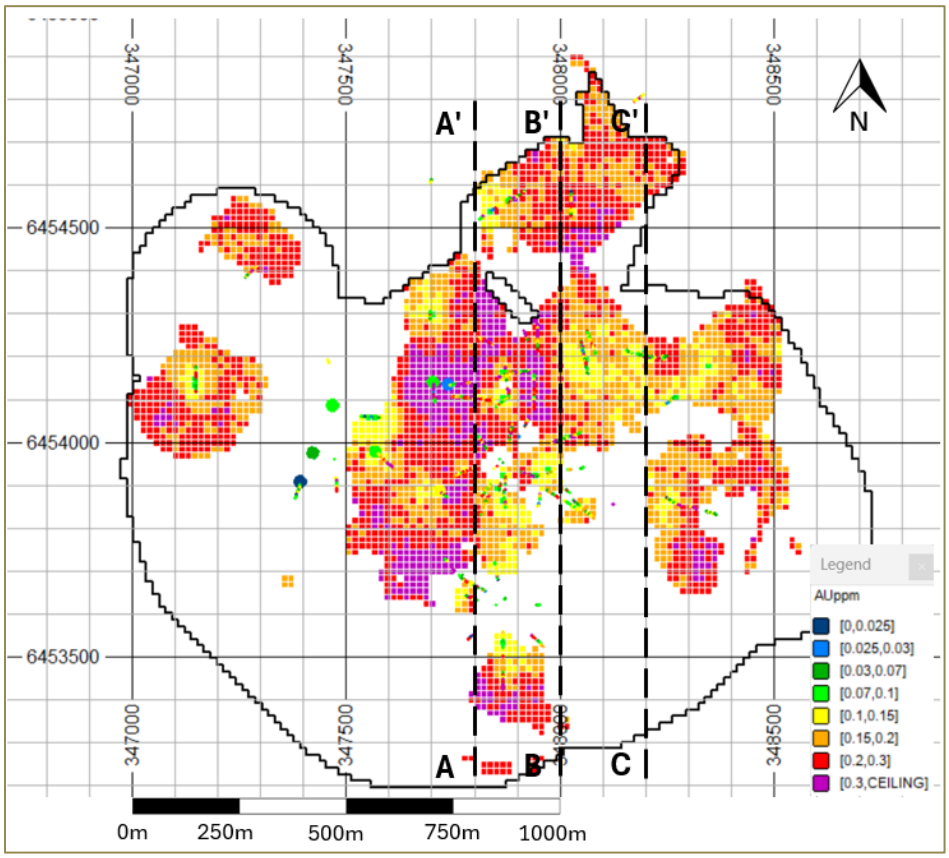


East 348000 looking West; Cu (ppm) grade distribution within the modelled domain

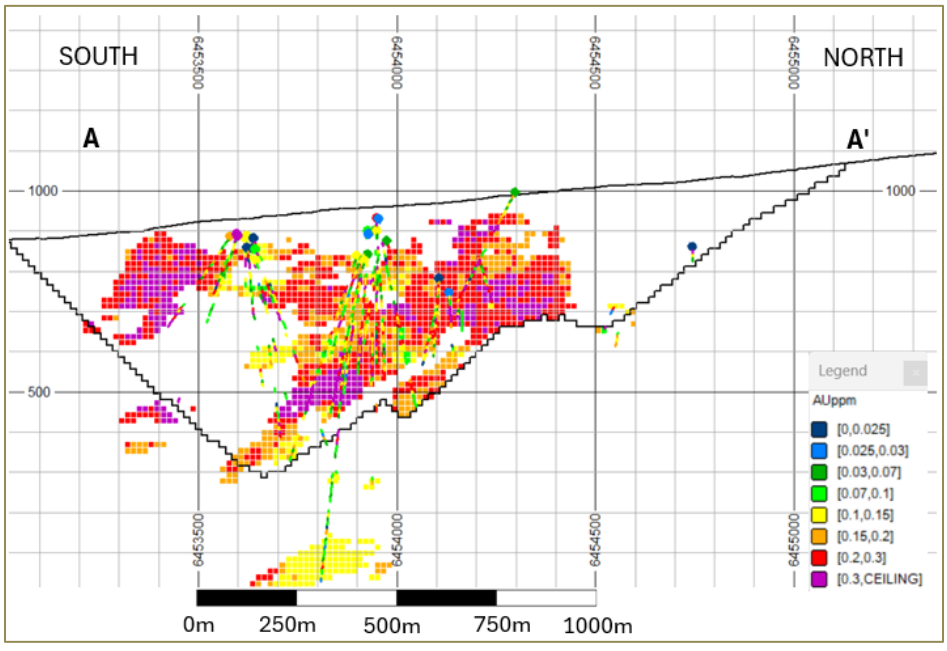


East 348200 looking West; Cu (ppm) grade distribution within the modelled domain

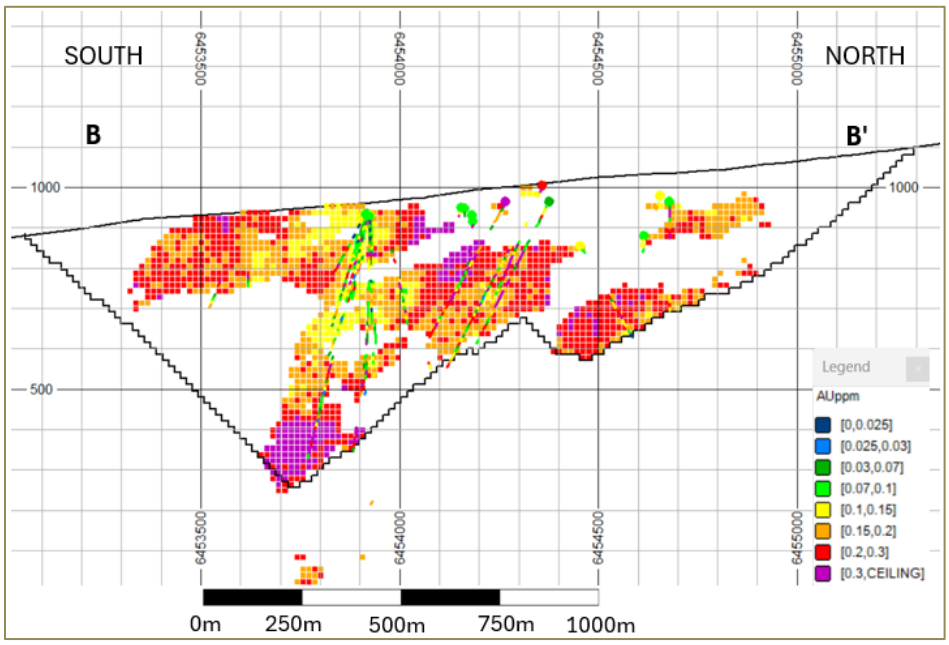
Figure 14-48: Plan View and Cross-Sections of Copper (Cu ppm) Grade Distribution within the planned 120 kt/d pit shell for the Hat Project. Block model: MDMRE_040226 (effective date 04/02/2026). Coordinate system: NAD83 / UTM Zone 9N (m). Drill holes are shown clipped to ±70 m from block-model intersection. Caption prepared by QP; data source: Doubleview Database.



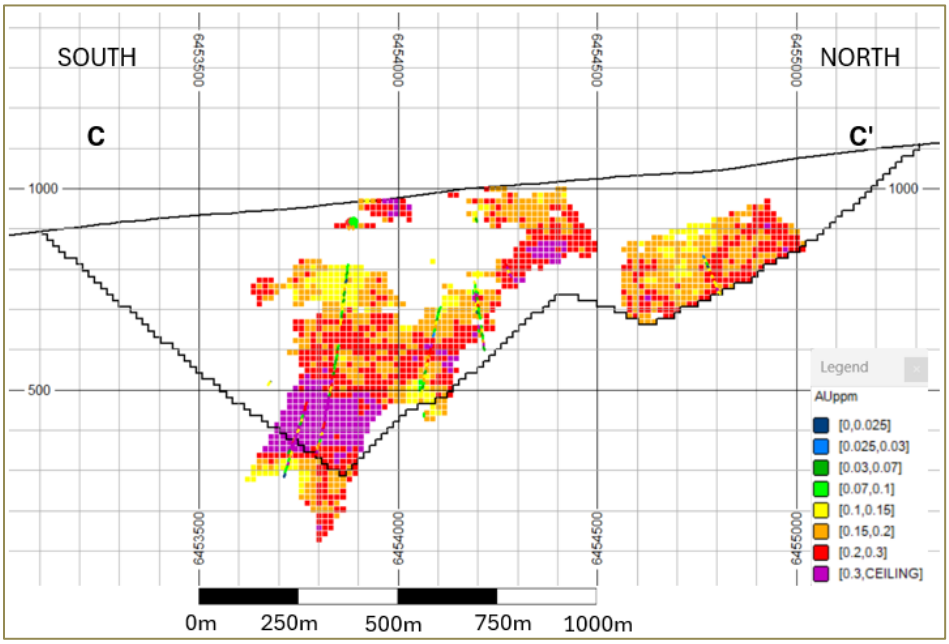
Plan View Elevation 700 masl; Au (ppm) grade distribution within the modelled domain



East 347800 looking West; Au (ppm) grade distribution within the modelled domain

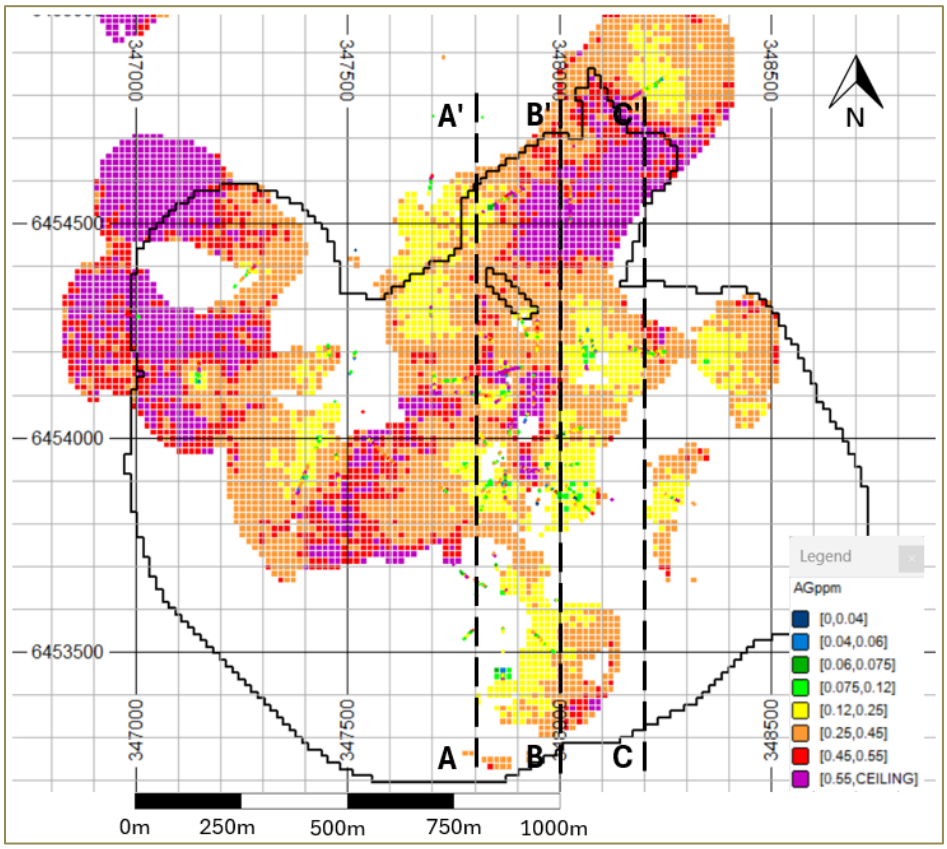


East 348000 looking West; Au (ppm) grade distribution within the modelled domain

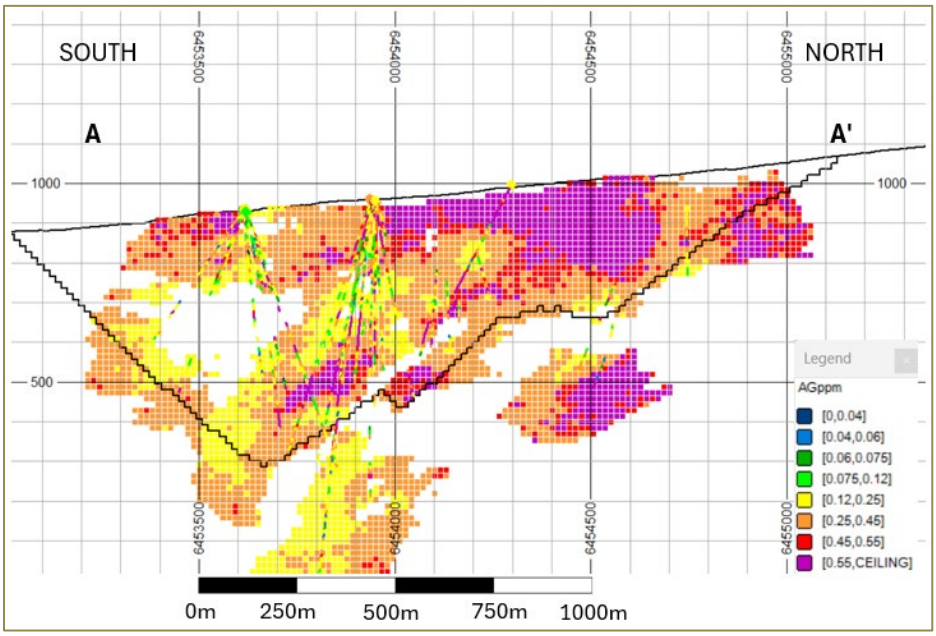


East 348200 looking West; Au (ppm) grade distribution within the modelled domain

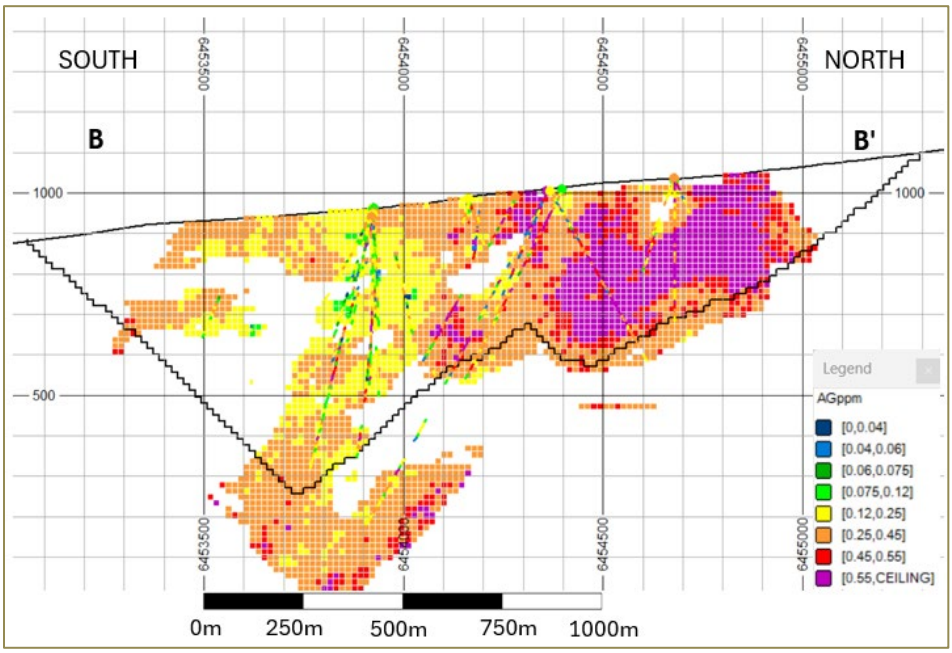
Figure 14-49: Plan View and Cross-Sections of Gold (Au ppm) Grade Distribution within the planned 120 kt/d pit shell for the Hat Project. Block model: MDMRE_040226 (effective date 04/02/2026). Coordinate system: NAD83 / UTM Zone 9N (m). Drill holes are shown clipped to ±70 m from block-model intersection. Caption prepared by QP; data source: Doubleview Database.



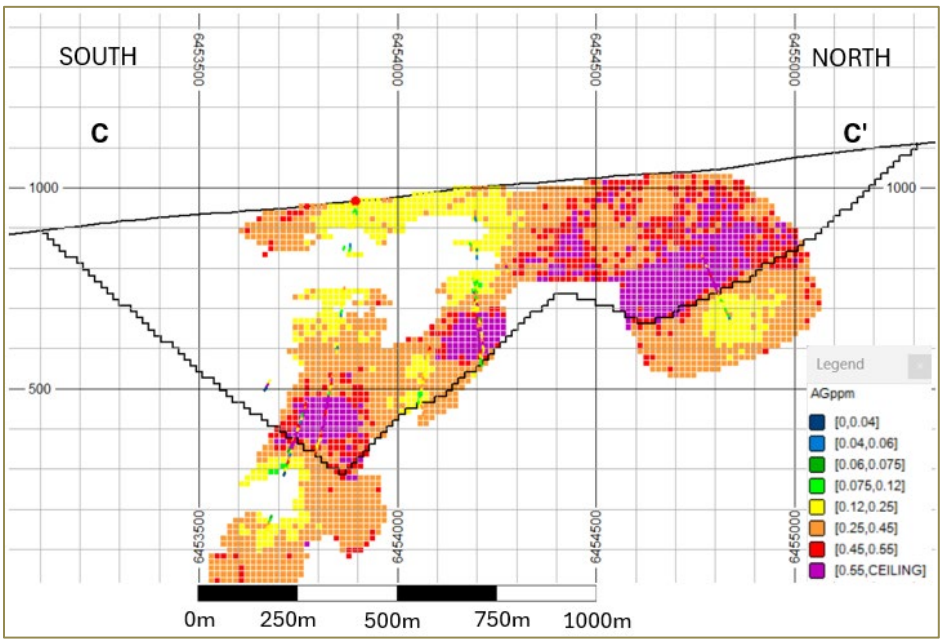
Plan View Elevation 700 masl; Ag (ppm) grade distribution within the modelled domain



East 347800 looking West; Ag (ppm) grade distribution within the modelled domain

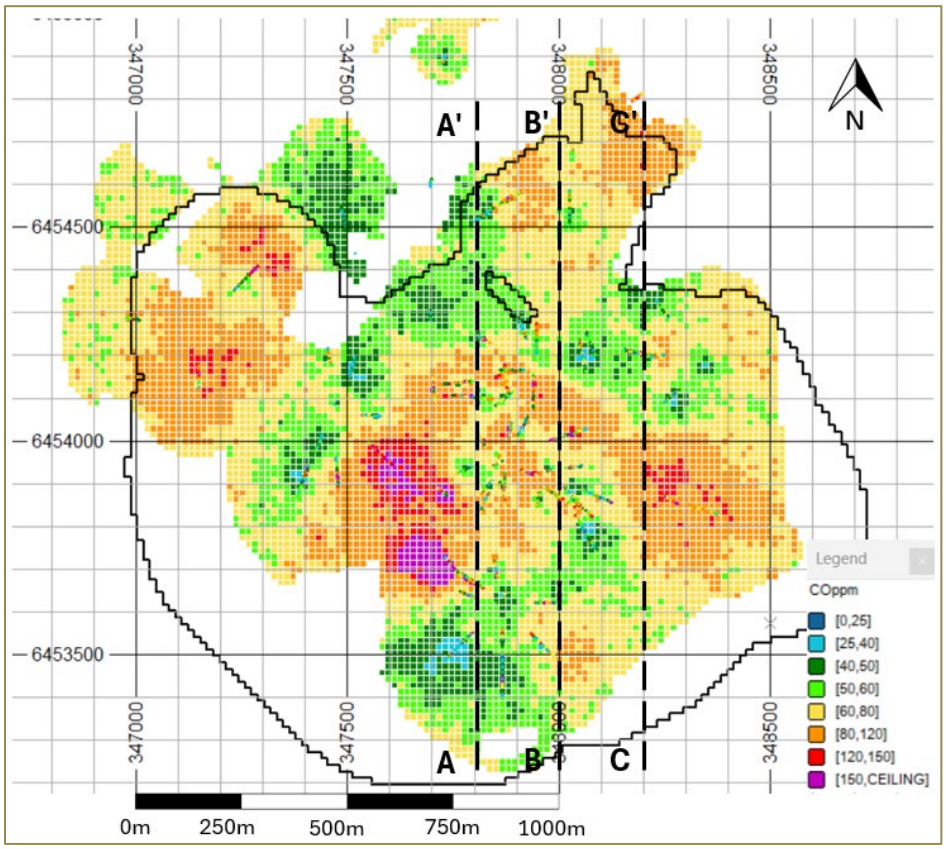


East 348000 looking West; Ag (ppm) grade distribution within the modelled domain

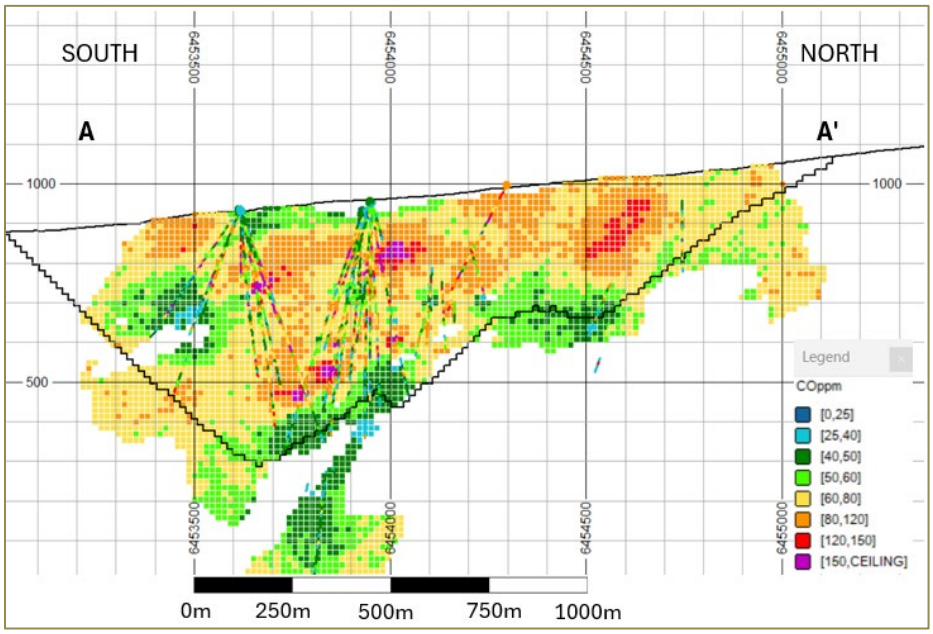


East 3482800 looking West; Ag (ppm) grade distribution within the modelled domain

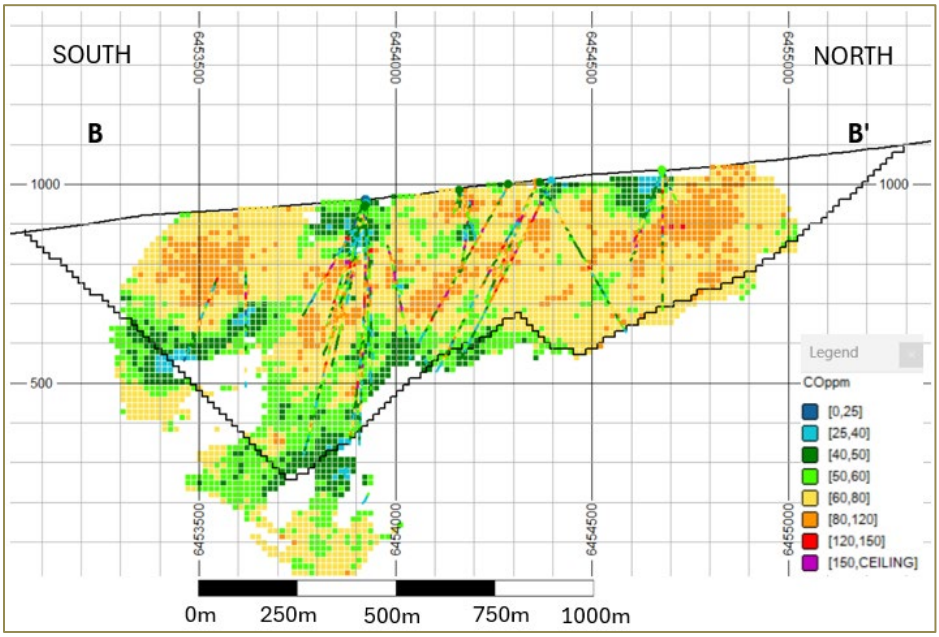
Figure 14-50: Plan View and Cross-Sections of Silver (Ag ppm) Grade Distribution within the planned 120 kt/d pit shell for the Hat Project. Block model: MDMRE_040226 (effective date 04/02/2026). Coordinate system: NAD83 / UTM Zone 9N (m). Drill holes are shown clipped to ±70 m from block-model intersection. Caption prepared by QP; data source: Doubleview Database.



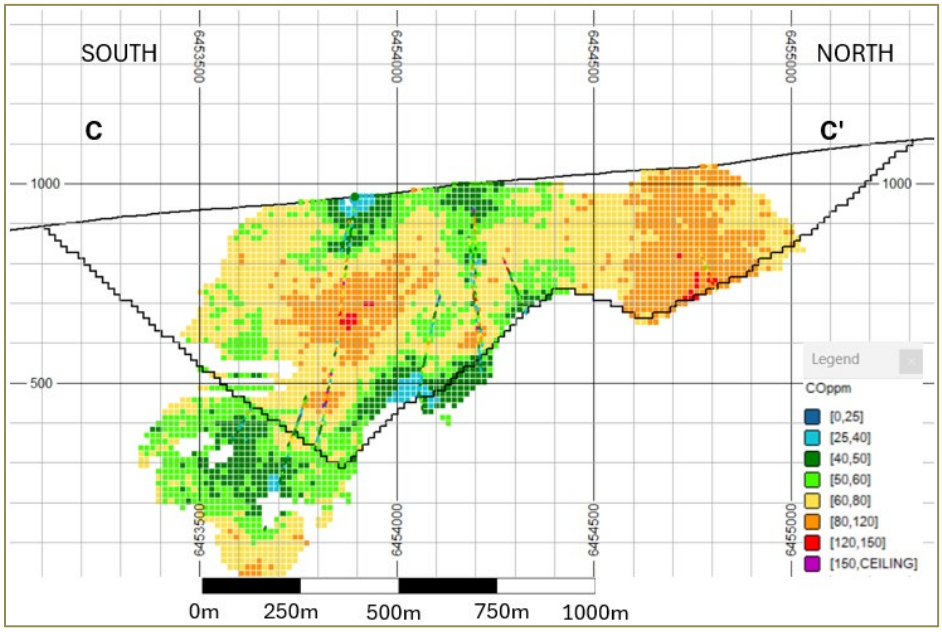
Plan View Elevation 700 masl; Co (ppm) grade distribution within the modelled domain



East 347800 looking West; Co (ppm) grade distribution within the modelled domain

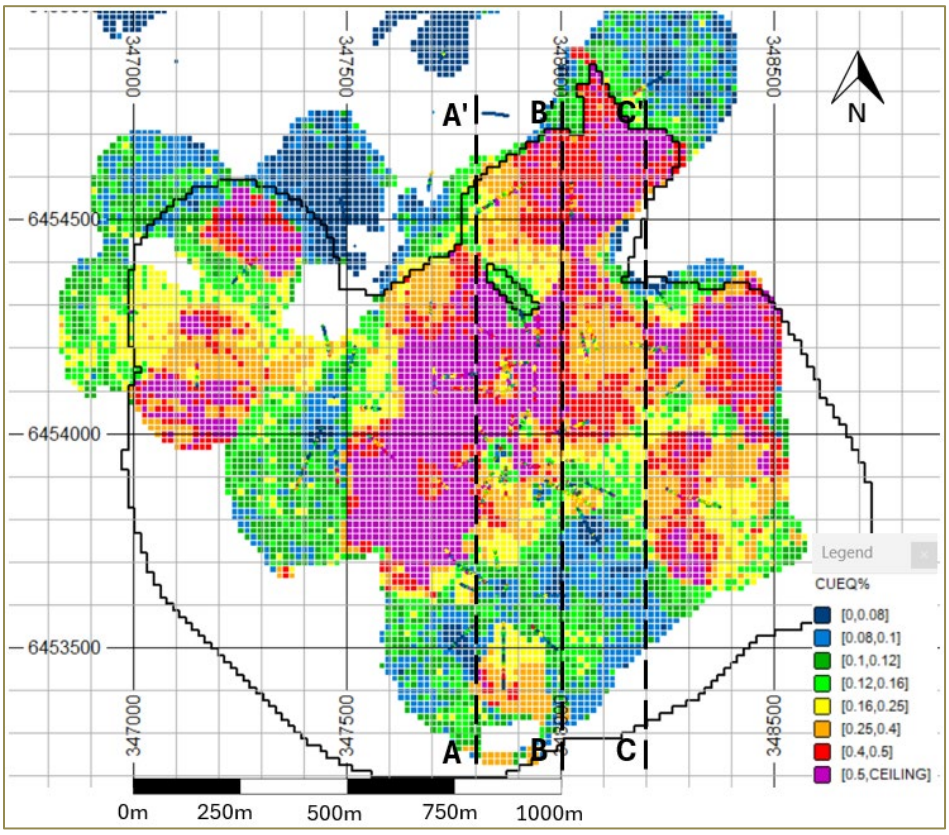


East 348000 looking West; Co (ppm) grade distribution within the modelled domain

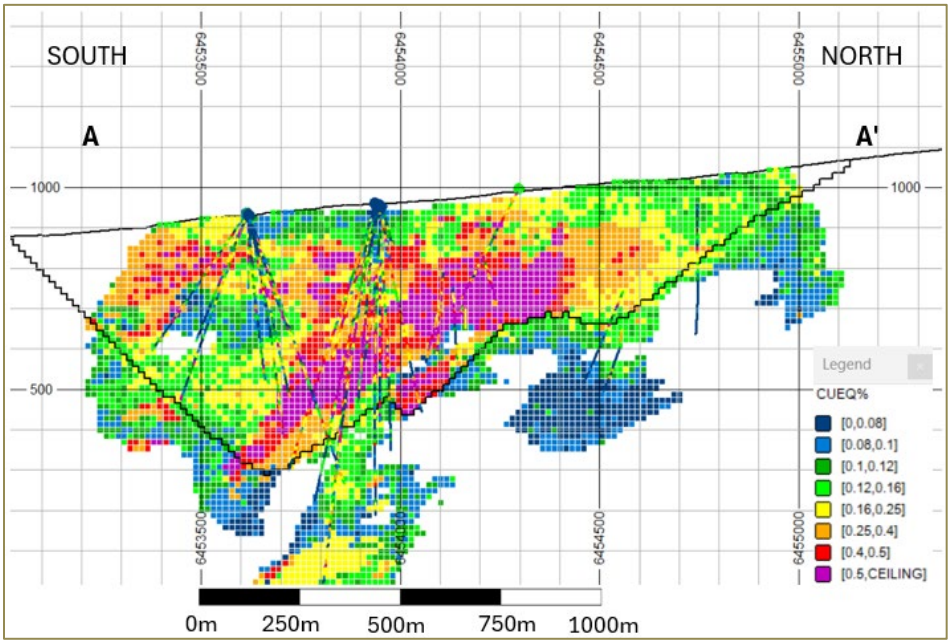


East 348200 looking West; Co (ppm) grade distribution within the modelled domain

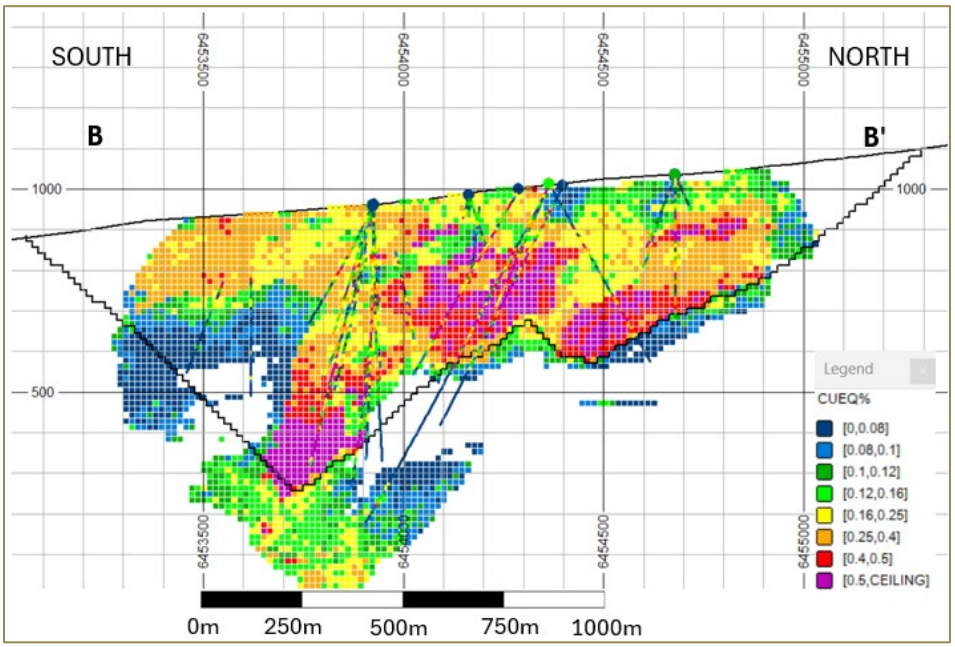
Figure 14-51: Plan View and Cross-Sections of Cobalt (Co ppm) Grade Distribution within the planned 120 kt/d pit shell for the Hat Project. Block model: MDMRE_040226 (effective date 04/02/2026). Coordinate system: NAD83 / UTM Zone 9N (m). Drill holes are shown clipped to ±70 m from block-model intersection. Caption prepared by QP; data source: Doubleview Database.



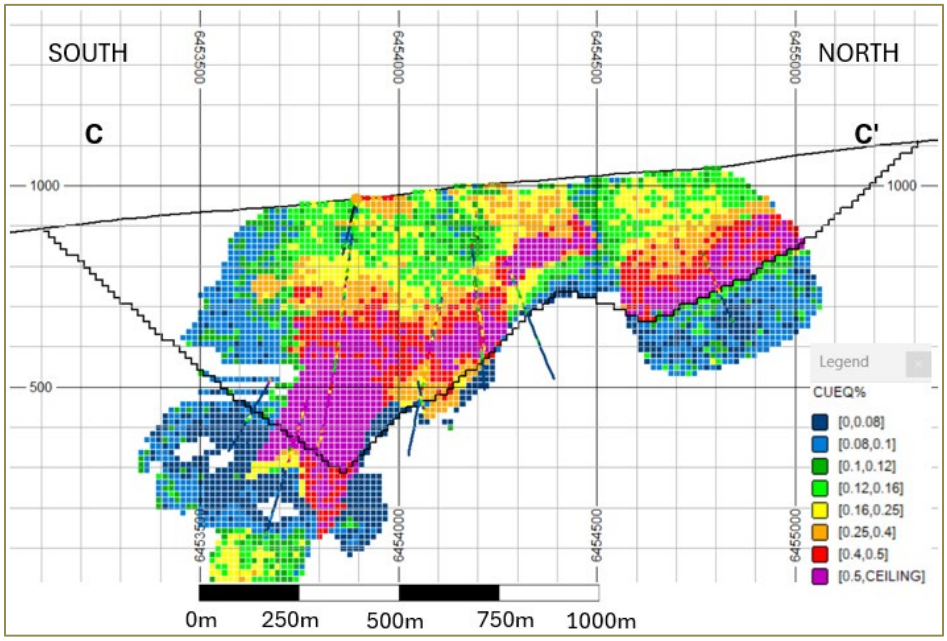
Plan View Elevation 700 masl; CuEq (%) grade distribution within the modelled domain



East 347800 looking West; CuEq (%) grade distribution within the modelled domain

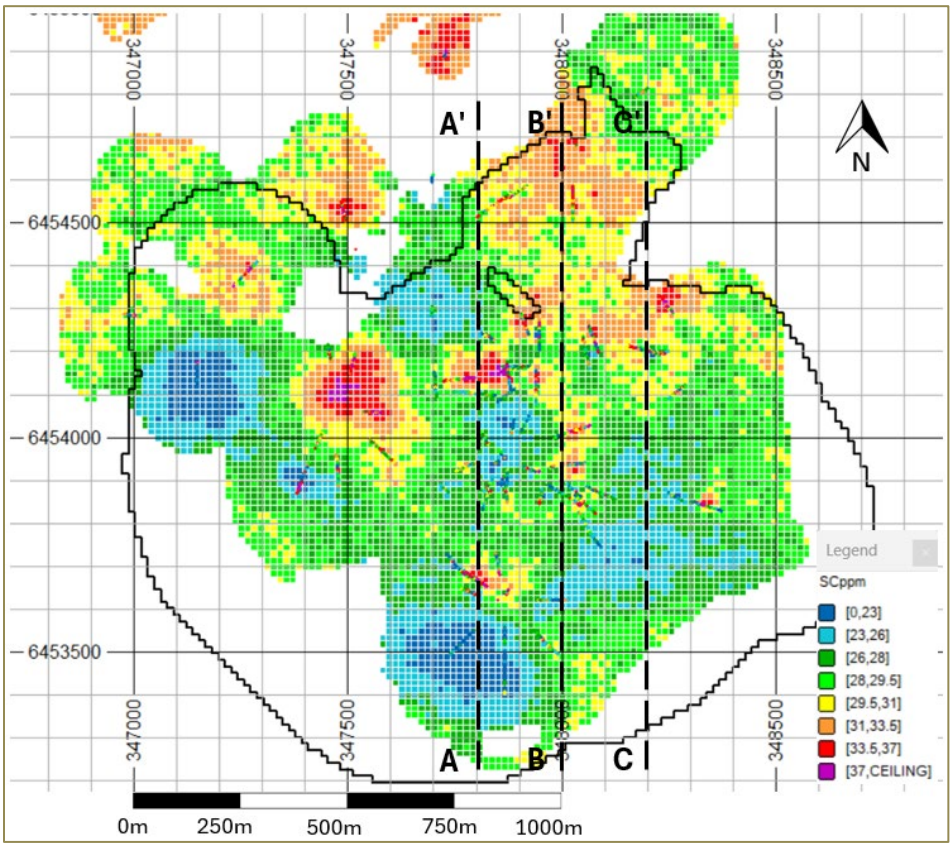


East 348000 looking West; CuEq (%) grade distribution within the modelled domain

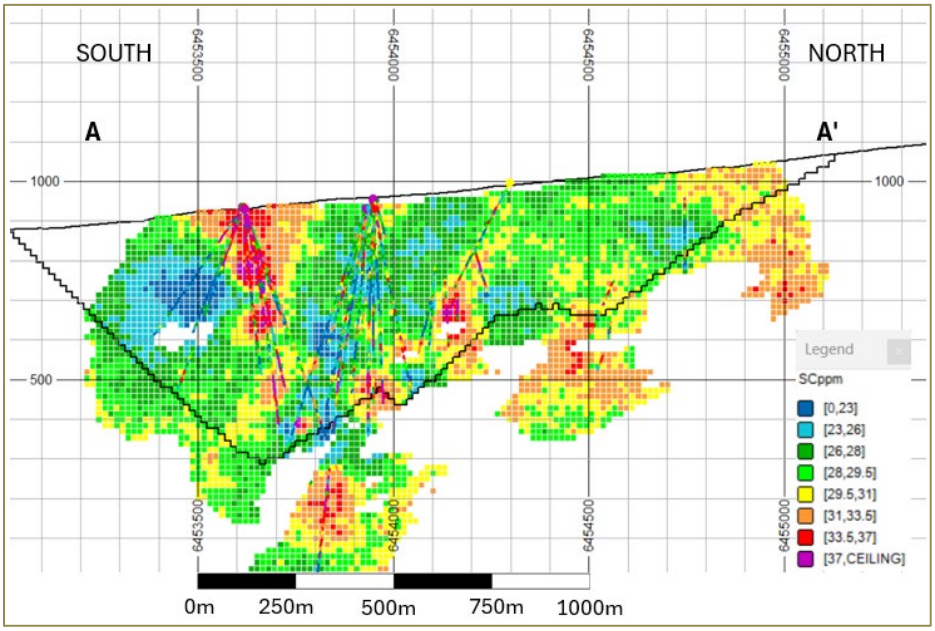


East 348200 looking West; CuEq (%) grade distribution within the modelled domain

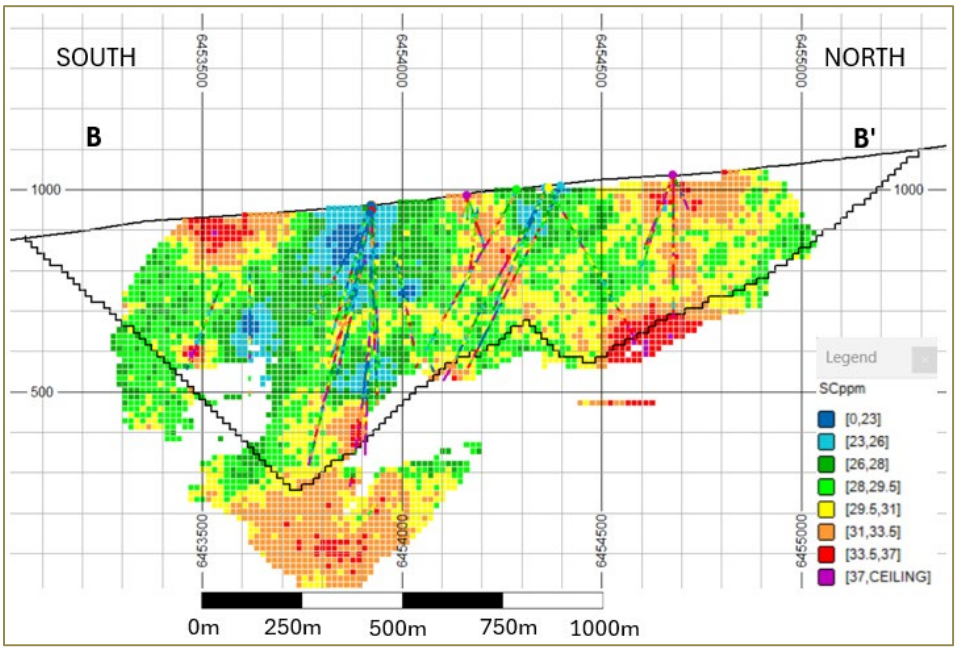
Figure 14-52: Plan View and Cross-Sections of Copper Equivalent (CuEq %) Grade Distribution within the planned 120 kt/d pit shell for the Hat Project. Block model: MDMRE_040226 (effective date 04/02/2026). Coordinate system: NAD83 / UTM Zone 9N (m). Drill holes are shown clipped to ±70 m from block-model intersection. Caption prepared by QP; data source: Doubleview Database.



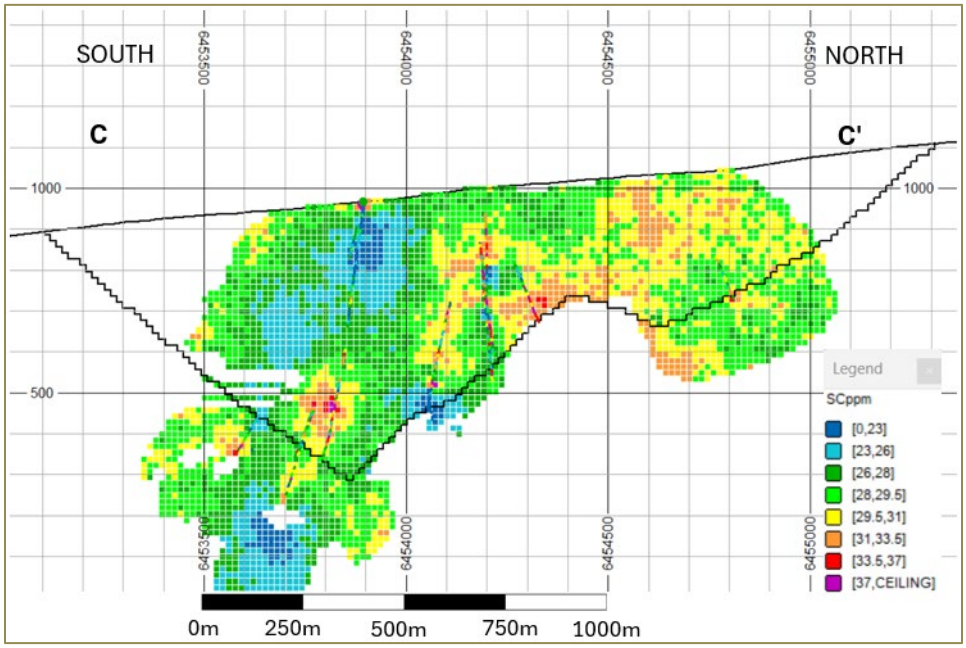
Plan View Elevation 700 masl; Sc (ppm) grade distribution within the modelled domain



East 347800 looking West; Sc (ppm) grade distribution within the modelled domain



East 348000 looking West; Sc (ppm) grade distribution within the modelled domain



East 348200 looking West; Sc (ppm) grade distribution within the modelled domain

Figure 14-53: Plan View and Cross-Sections of Scandium (Sc ppm) Grade Distribution within the planned 120 kt/d pit shell for the Hat Project. Block model: MDMRE_040226 (effective date 04/02/2026). Coordinate system: NAD83 / UTM Zone 9N (m). Drill holes are shown clipped to ±70 m from block-model intersection. Caption prepared by QP; data source: Doubleview Database.

14.12.4 Density Measurement and Estimation

As of the database cut-off date of November 9, 2025, a total of 3,002 density measurements were available. Of these, ALS measured 208 samples under an analytical laboratory protocol, and 2,794 samples were obtained through in-house procedures Hat conducted. The ALS samples averaged 1.65 m long (range: 0.38 to 3.70 m), whereas the Hat in-house samples averaged 0.17 m (range: 0.07 to 0.35 m).

Density was determined using Archimedes' principle, which assumes full saturation of the sample and infers volume from water displacement. Despite the shared principle, the two datasets differ in method. ALS laboratory procedures included drying, wax coating, and full saturation, while Hat in-house measurements did not involve drying, wax coating, or complete saturation. These procedural differences created two subtly distinct data populations.

A comparative analysis was undertaken prior to receipt of the final batch of samples. For this exercise, 208 ALS samples and 2,595 Hat samples were considered.

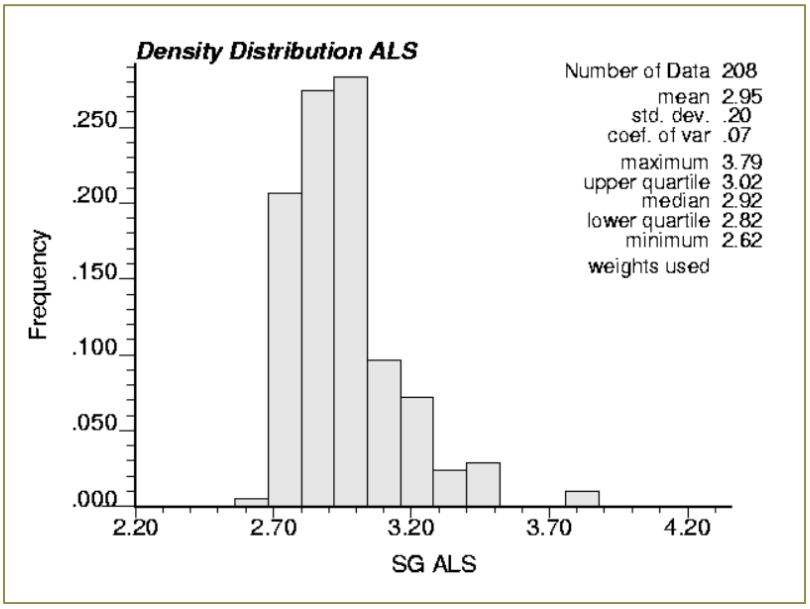


Figure 14-54: Density Results from ALS Laboratory Analysis

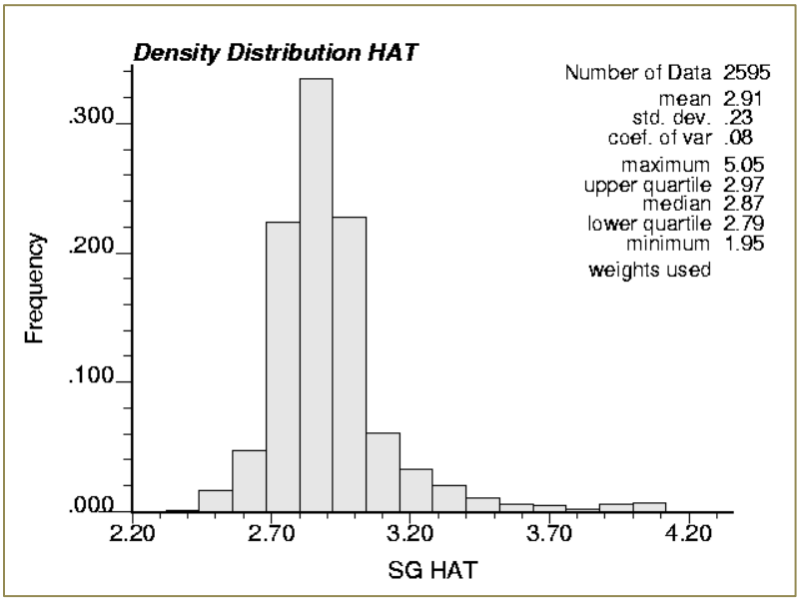


Figure 14-55: Density Results from Hat In-House Measurements

The Q-Q plot in Figure 14-55 reveals slight but persistent discrepancies. Up to approximately 3.5 g/cm³, ALS measurements are consistently higher than the Hat in-house values. In contrast, at the upper tail, the Hat results display a sudden dispersion into high densities. This comparison was conducted for analytical purposes only and does not represent a concern for the reliability of the density database. The Hat Project benefits from a dataset to support spatial density estimation across the ore body.

Within the dataset of 3,002 samples, four instances were identified where two density measurements followed immediately within the same drill hole. These pairs were merged by averaging, resulting in a final database of 2,998 samples used for estimation.

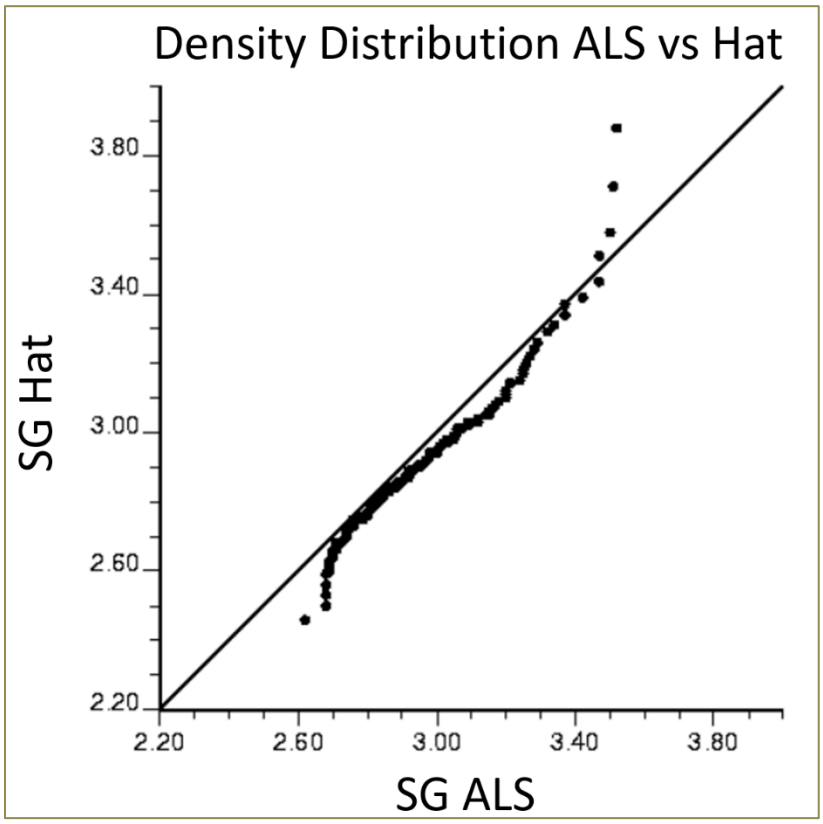


Figure 14-56: Q-Q Plot where ALS Laboratory Results are Compared to Hat In-House Measurements

Single outliers were identified at both ends of the density distribution. At the lower tail, the sample with a density of 1.95 g/cm³ was capped to 2.38 g/cm³, and at the upper tail, the sample with a density of 5.05 g/cm³ was capped to 4.75 g/cm³. The dataset was then declustered using an ordinary-kriging weighting scheme and transformed into Gaussian variables for spatial correlation analysis. Experimental correlogram points were generated using the correlogram formula, and the fitted correlogram model is shown in Figure 14-56, with numerical parameters summarized in Table 14-14.

Density estimation across the Project domain was performed using Sequential Conditional Gaussian Simulation.

The following figures summarize the parameters used in the Gaussian simulation process:

- Minimum number of composites—1
- Maximum number of composites—12
- Previously simulated nodes—6
- Search radius—full-radius search applied.

A limited number of samples was deliberately chosen to avoid excessive smoothing of a low-variability population. In total, 51 simulations were completed, reproducing the input data histogram and validated against cumulative distribution function plots.

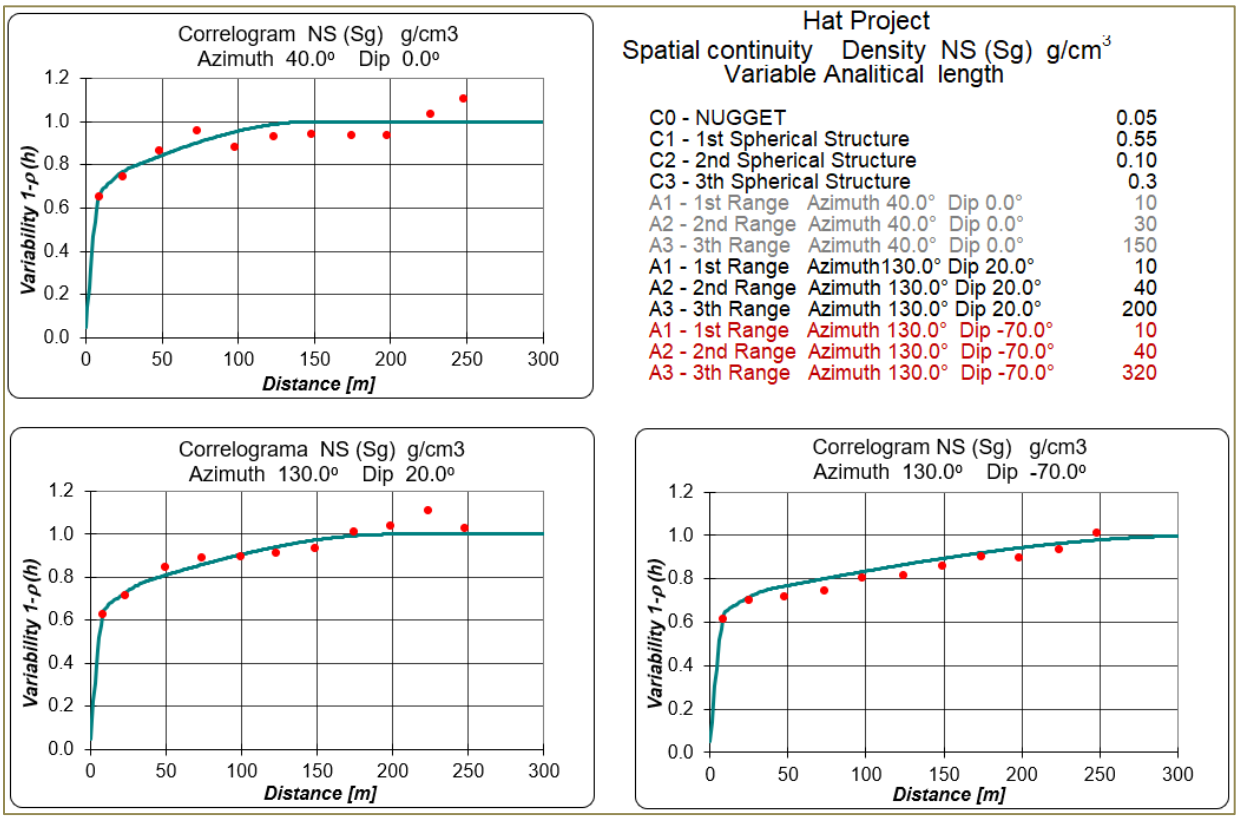


Figure 14-57: Spatial Correlogram for Density (SG)

Table 14-14: Summary of Numerical Parameters for Density Simulation

	Spherical	Axis Y (m)	Axis X (m)	Axis Z (m)
Density SG				
C0	0.05	1st Rot. 130° by Z	2nd Rot. −70° by X	
C1	0.55	10	10	10
C2	0.10	40	30	40
C3	0.30	320	150	200
Search		360	180	250

Notes: Rotation about the Z-axis is applied clockwise when viewed from above. Rotation about the X-axis is applied clockwise when viewed from the positive X direction, resulting in a downward rotation.

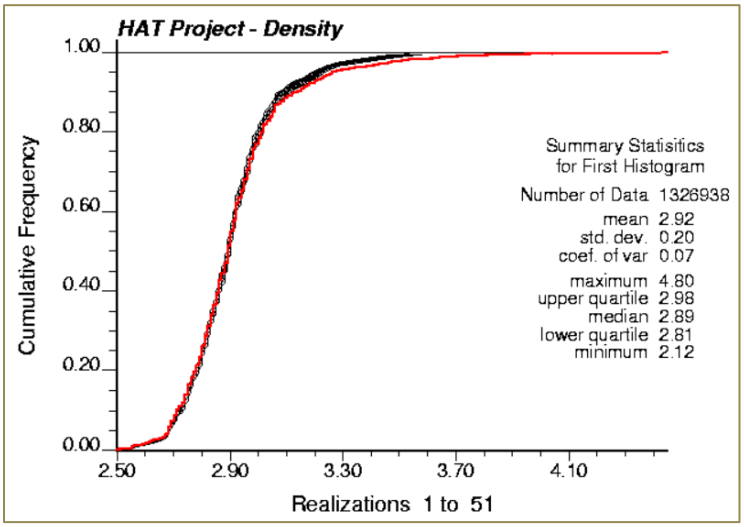


Figure 14-58: Validation of Successful Reproduction of 51 Simulations of Input Data Red Line

The upper 10% of the simulated density distribution deviated slightly from the original data; however, the simulations consistently reproduced the upper-tail behaviour, extending to 4.8 g/cm³, which is consistent with values present in the database. The mean density was accurately reproduced, and the overall distribution is considered satisfactory for Mineral Resource estimation purposes.

14.13 Mineral Resource Classification

14.13.1 Principles of Reliable Classification

The following principles have been established for the Mineral Resource classification applied to the Hat Project:

- Transparency and objectivity—achieved through a defined set of parameters consistent with geometry/grade modelling and interpolation strategy.
- Reproducibility and auditability—methods must be based on quantifiable principles applicable at production scale.
- Production scale reconciliation—classification should consider the production scale over a defined period, reflecting the relationship between support size and estimation error.
- Acceptable prediction discrepancy—tolerances between predicted and actual results must be explicitly defined according to the nature of mineralization and operational requirements.
- Identification of improvement areas—classification outputs should highlight zones where confidence in geometry or grade estimates require strengthening.

14.13.2 Industry Rules and Adaptation to Exploration Stage

In the mining industry, widely accepted rules for Mineral Resource classification (particularly in the base metals sector) are:

- Measured Resources are local estimates, adjusted to 1–3-month production units, that demonstrate an estimation error of ±15% at the 90% confidence level.
- Indicated Resources are local estimates, adjusted to annual production units, that demonstrate an estimation error of ±15% at the 90% confidence limit.
- Inferred Resources: Estimates that do not meet the above criteria.

However, the Project under development is still at the exploration stage, where drilling density and geometry do not yet support production-scale confidence levels. The requirement of a 90% confidence limit is too demanding under current conditions, particularly where drilling is irregular and often conducted in fan-shaped patterns from a single drill pad. Such configurations present difficulties in achieving an optimal drilling pattern and do not allow expected reconciliation at production scale.

14.13.3 Adjustment of Classification Criteria for the Exploration Stage

For this Project, the confidence limits used for classification have been modified to reflect the practical constraints of the exploration dataset. Rather than applying the industry-standard 10%–15% estimation error at a 90% confidence level, a 20% estimation error at an 80% confidence level has been adopted as a more realistic criterion. This modification accounts for irregular drill spacing and limited data density, while preserving transparency, reproducibility, and defensibility of the classification approach.

Proposed Method

Wawruch and Betzhold's (2004) classification method, applied here, is based on:

- Multiple realizations of conditional simulation.
- Calculation of the CV for local mining units.
- Change of support related to mining extraction over defined production periods. In the Hat Project two production periods are considered: 2 months and 12 months for Measured and Indicated Mineral Resources respectively.
- Classification according to established requirements in terms of percentage error and confidence limits, adjusted for exploration conditions. In the Hat Project this criterion is established as 20% error prediction at 80% confidence limit.

Treatment of Variability

A meaningful approach to classification accounts for variability of local blocks within larger production volumes. Local geometry/grade variability is expressed at the scale of mining units through a variability reduction factor, calculated from averages of CV for local blocks and production panels. Confidence limits are addressed by correcting the local CV using the variance reduction factor.

Strategic Importance

Mineral Resource classification is an integral part of evaluation and reconciliation. It is a strategic tool that allows assessment of tonnage and grade uncertainty within the exploration development or mining program. The applied method integrates transparency, objectivity, and geostatistical tools commonly used in modelling and evaluation.

Numerical Components Supporting the Process

1. Average local coefficient of variability
2. Average panel 1 coefficient of variability—panel 1 represents 2 months of production; dimensions: 175 × 175 × 45 m
3. Average panel 2 coefficient of variability—panel 2 represents 12 months of production; dimensions: 360 × 360 × 60 m
4. Variance-reduction factor (1) applied to the bi-monthly production panel
5. Variance-reduction factor (2) applied to the annual production panel
6. Local variability coefficient transformed through factor 1 into panel 1 variability coefficient
7. Local variability coefficient transformed through factor 2 into panel 2 variability coefficient.

14.13.4 Classification Results

Each variable is classified independently because their statistical and geostatistical characteristics differ, even when they share the same spatial locations. Their spatial behaviour, continuity, and variability are distinct and therefore require separate treatment.

The first aspect to consider is the quantity of data supporting each variable and the cut-off values used for domain definition. Using average–variance relationships, the cut-offs separating background mineralization from probable economic mineralization were established as follows:

- Copper—900 ppm → 33% of data above cut-off
- Gold—0.07 ppm → 38% of data above cut-off
- Cobalt—35 ppm → 76% of data above cut-off
- Silver—0.1 ppm → 59% of data above cut-off.

Scandium does not manifest behaviour like the other variables, and therefore no natural cut-off between background and probable economic mineralization was detected.

This analysis provided the basis for defining the mineralized domains. Within these domains, grade continuity varies among the different components of the mineralization system, and the nugget effects differ substantially. These considerations are discussed in detail in Section 14.10.

Classification is carried out independently for each variable, and the resulting classifications are subsequently merged to produce the combined classification for the CuEq variable.

Summary of Classification Results

The results expressed in terms of proportional percentages within corresponding domain are presented in Table 14-15 and Figure 14-59 for all classified variables.

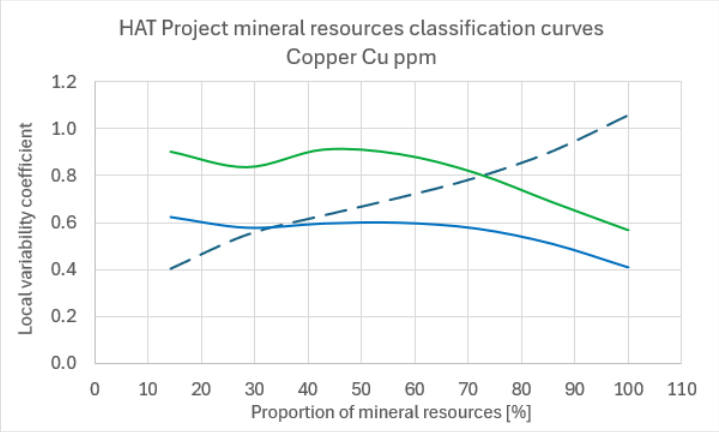
Table 14-15: Proportional Percentage Distribution of Classified Variables by Estimation Domain

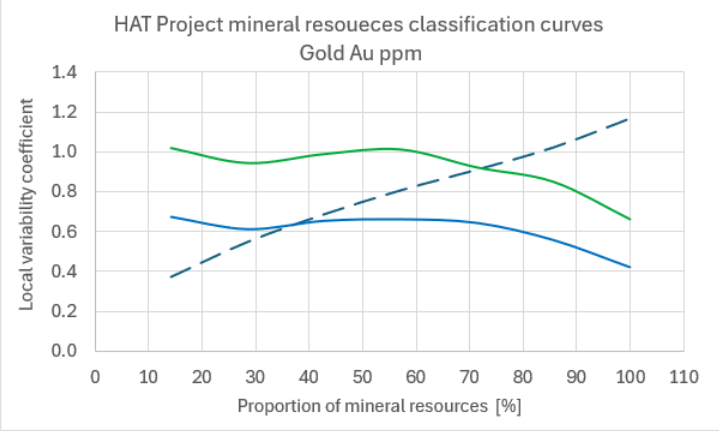
Category	Cu	Au	Co	Ag	Sc
Measured %	32.7	37.8	75.4	29.9	100.0
Indicated %	41.5	36.6	14.2	24.1	0.0
Inferred %	25.9	25.6	10.4	46.0	0.0

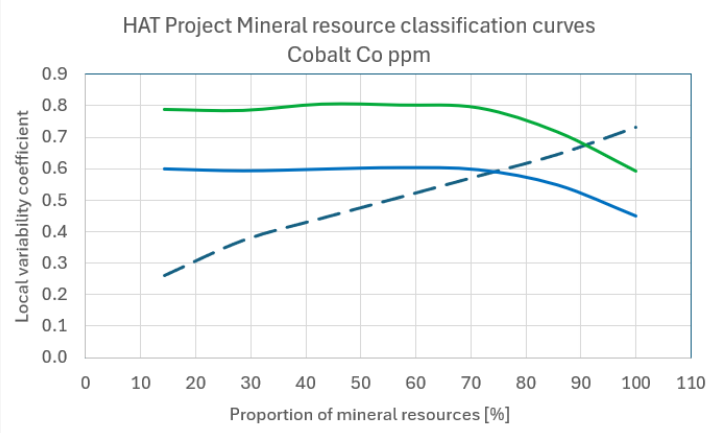
The applied classification method quantifies confidence based on the local variability of each variable. Although all variables share the same drilling pattern, each commodity exhibits distinct spatial continuity and mineralized volumes. These differences directly influence estimation confidence and must be reflected in the classification. In this case, the resulting categories appropriately capture the differing behaviours of the variables.

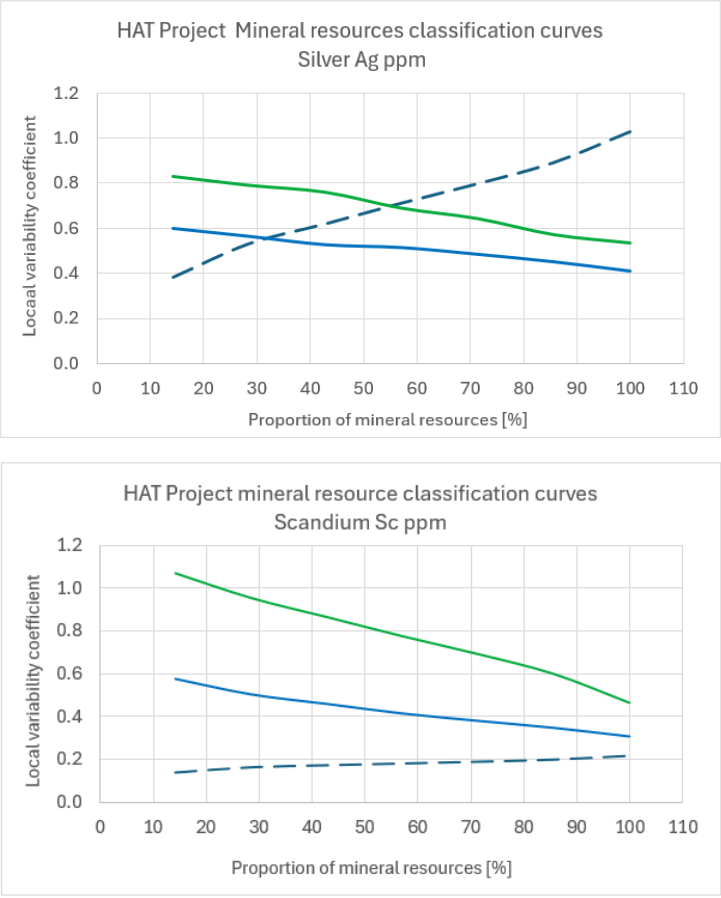
Because the Hat Project classification method is based on block-scale variability rather than drilling-distance rules or fixed spatial thresholds, the differences among variables are more clearly expressed. For example, scandium, which exhibits a stationary spatial distribution without strong boundary effects (average grade and variance is constant across the domain); thus, its distribution is the easiest to predict and its production scale can be estimated with high confidence under the proposed criteria. Cobalt also displays low variability, and when evaluated at the two-month production scale, its estimation error is reduced to below the 20% classification threshold across most of the deposit. Silver, gold, and copper exhibit higher estimation uncertainty, as reflected in their respective confidence-category proportions. Among them, silver has the greatest grade variability and estimation uncertainty, which results in the highest proportion of Inferred Mineral Resources. Gold and Copper show similar classification behaviour. Their Measured proportions (37.8% for Au and 32.7% for Cu) are lower than cobalt (75.4%) but higher than silver (29.9%). Both metals also have comparable proportions of Indicated Mineral Resources (36.6% for Au and 41.5% for Cu). This similarity in classification distribution indicates that gold and copper have broadly comparable levels of estimation confidence and that similar additional drilling density would be required to improve their classification.

The classification method is designed to reflect the statistical and geostatistical behaviour of each variable. Because the variables exhibit different levels of variability, continuity, and data support, the resulting classification appropriately captures these differences in confidence. The outcomes therefore align with the expected behaviour of the variables, demonstrating that the method is functioning as intended.


HAT Project mineral resources classification curves
Copper Cu ppm
Local variability coefficient
1.2
1.0
0.8
0.6
0.4
0.2
0.0
0 10 20 30 40 50 60 70 80 90 100 110
Proportion of mineral resources [%]

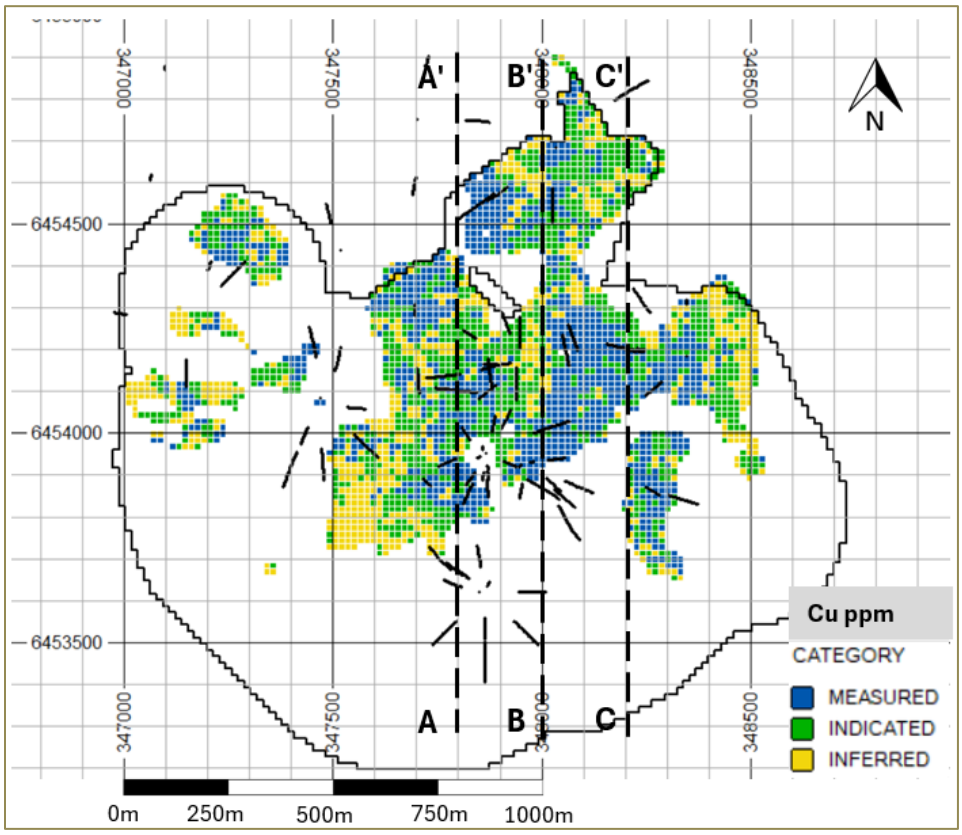

HAT Project mineral resoueces classification curves
Gold Au ppm
Local variability coefficient
1.4
1.2
1.0
0.8
0.6
0.4
0.2
0.0
0 10 20 30 40 50 60 70 80 90 100 110
Proportion of mineral resources [%]


HAT Project Mineral resource classification curves
Cobalt Co ppm
Local variability coefficient
0.9
0.8
0.7
0.6
0.5
0.4
0.3
0.2
0.1
0.0
0 10 20 30 40 50 60 70 80 90 100 110
Proportion of mineral resources [%]

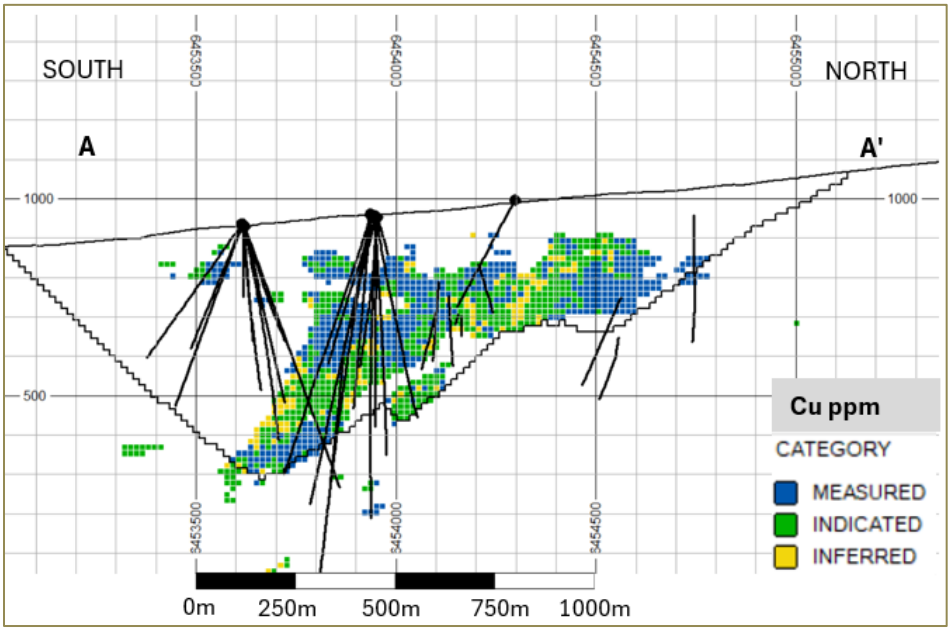


Note: Classification curves are showing the relationship between local variability and confidence category assignment. The dashed line represents the local coefficient of variability, while the solid curves correspond to the proportions of Measured (blue) and Indicated (green) Resources. The points where the solid curves intersect the variability curve mark the classification thresholds: blocks with lower variability are assigned to higher confidence. The horizontal axis approximates the cumulative proportion of Mineral Resources, starting from zero, and reflects how much of the deposit falls into each category.

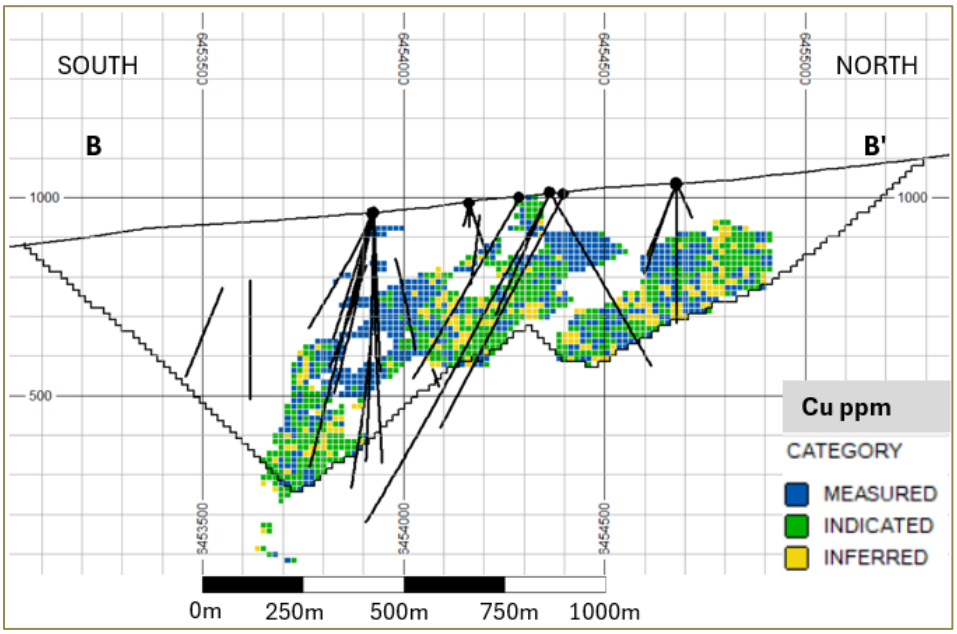
Figure 14-59: Classification Curves (Blue = Measured; Green = Indicated; Dashed = Local Coefficient of Variability)



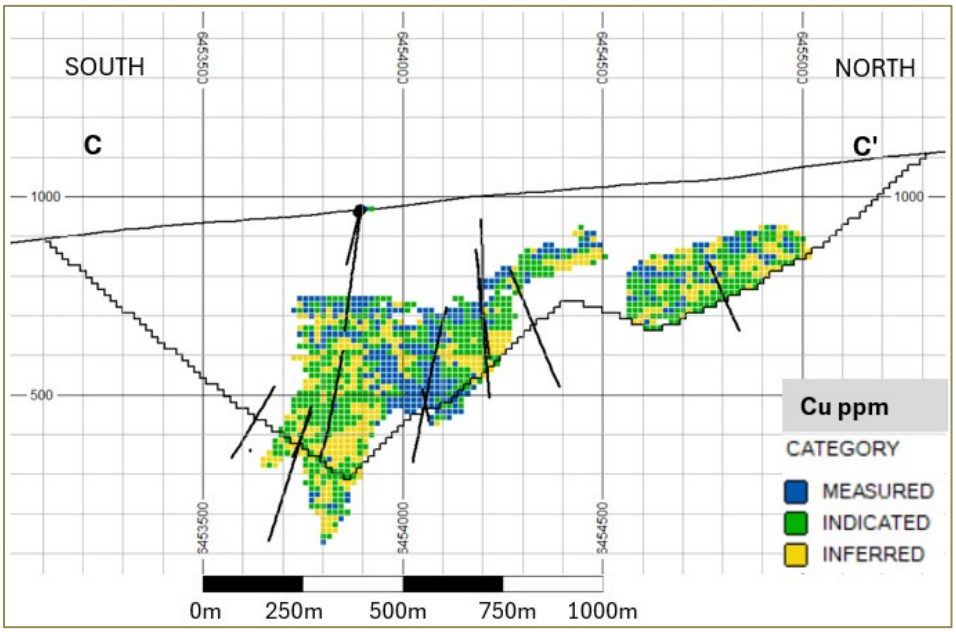
Plan View Elevation 700 masl—Block Model Showing Copper Mineral Resource Categories distribution



East 347800 Looking West—Block Model Showing Copper Mineral Resource Categories distribution

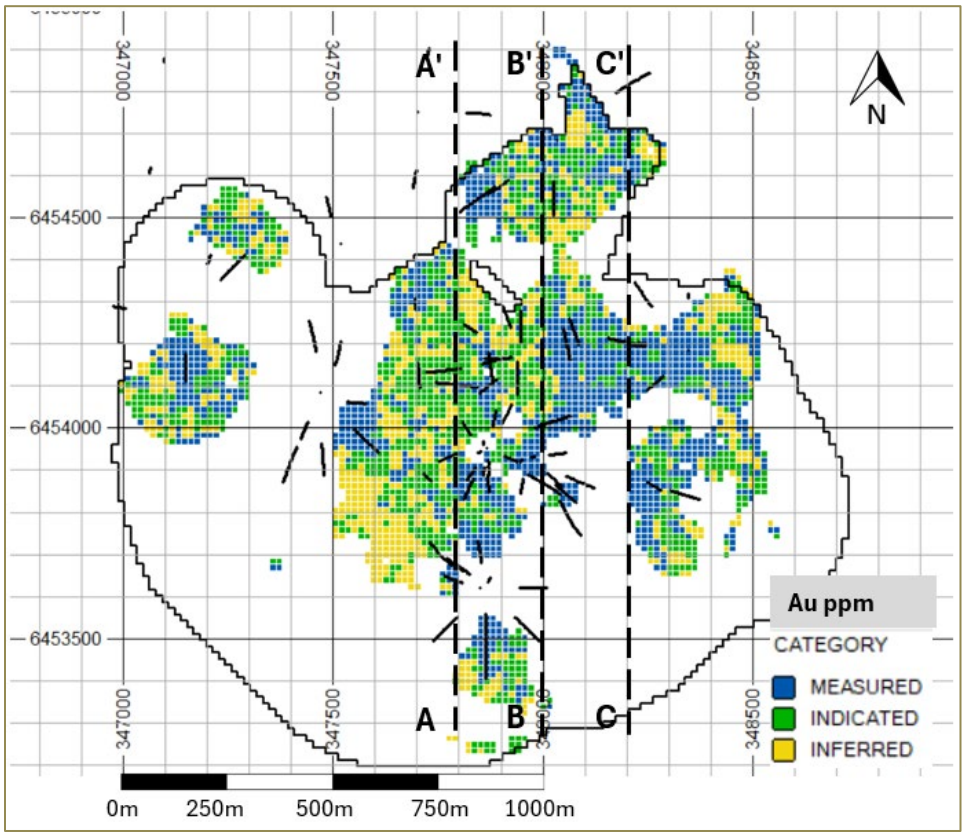


East 348000 Looking West—Block Model Showing Copper Mineral Resource Categories distribution

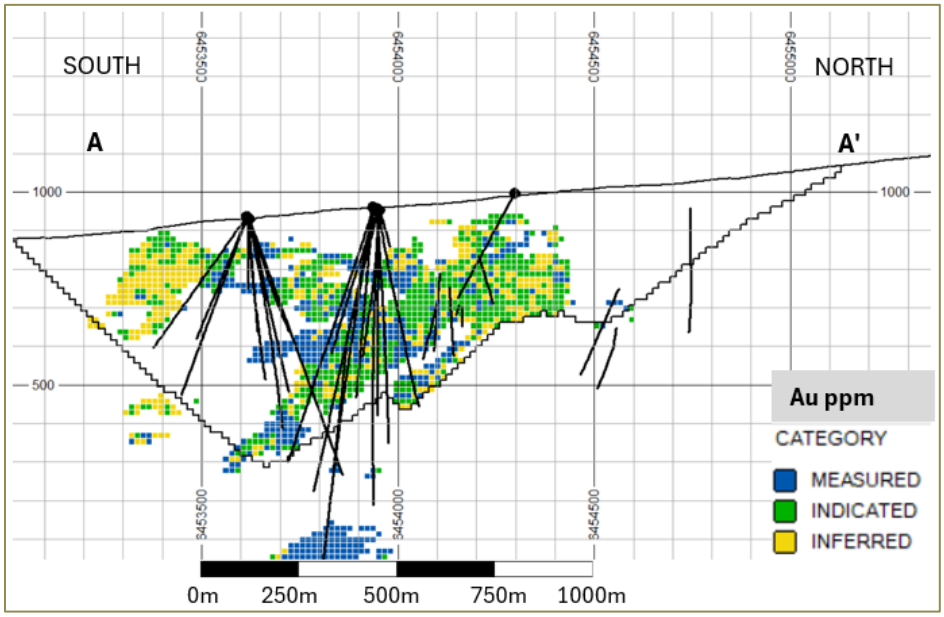


East 348200 Looking West—Block Model Showing Copper Mineral Resource Categories distribution

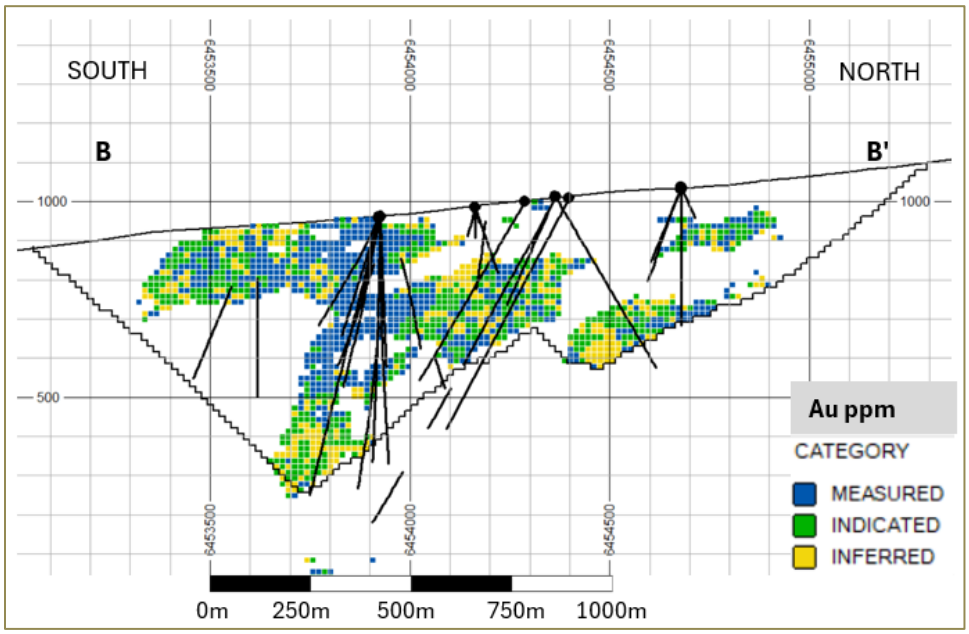
Figure 14-60: Copper Domain (Cu ppm) mineral-resource distribution by category (Measured, Indicated, Inferred) within the planned 120 kt/d pit shell for the Hat Project. Block model: MDMRE_040226 (effective date 04/02/2026). Coordinate system: NAD83 / UTM Zone 9N (m). Drill holes are shown clipped to ±70 m from block-model intersection. Caption prepared by QP; data source: Doubleview Database.



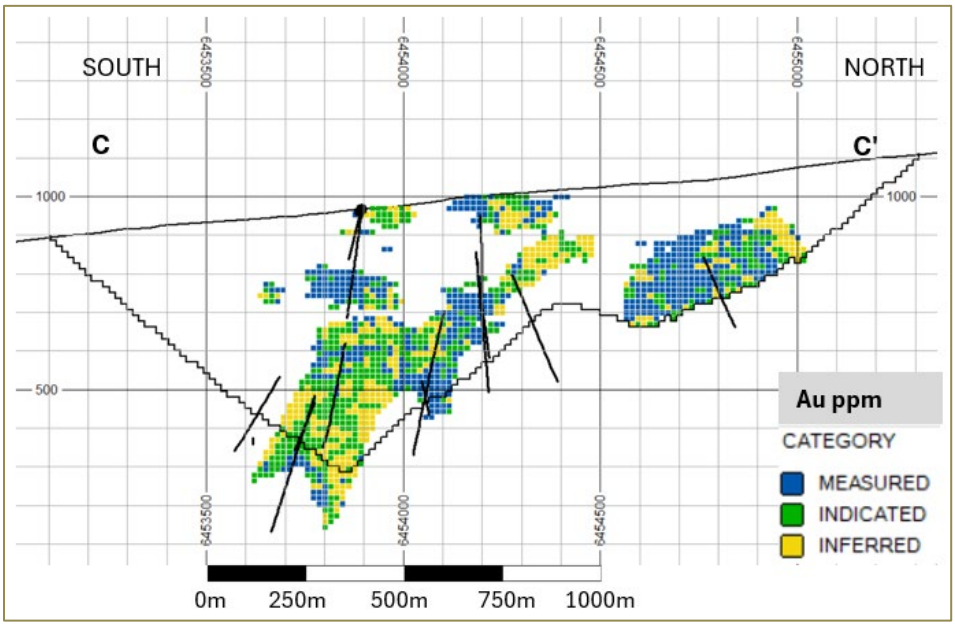
Plane View Elevation 700 masl—Block Model Showing Gold Mineral Resource Categories distribution



East 347800 Looking West—Block Model Showing Gold Mineral Resource Categories distribution

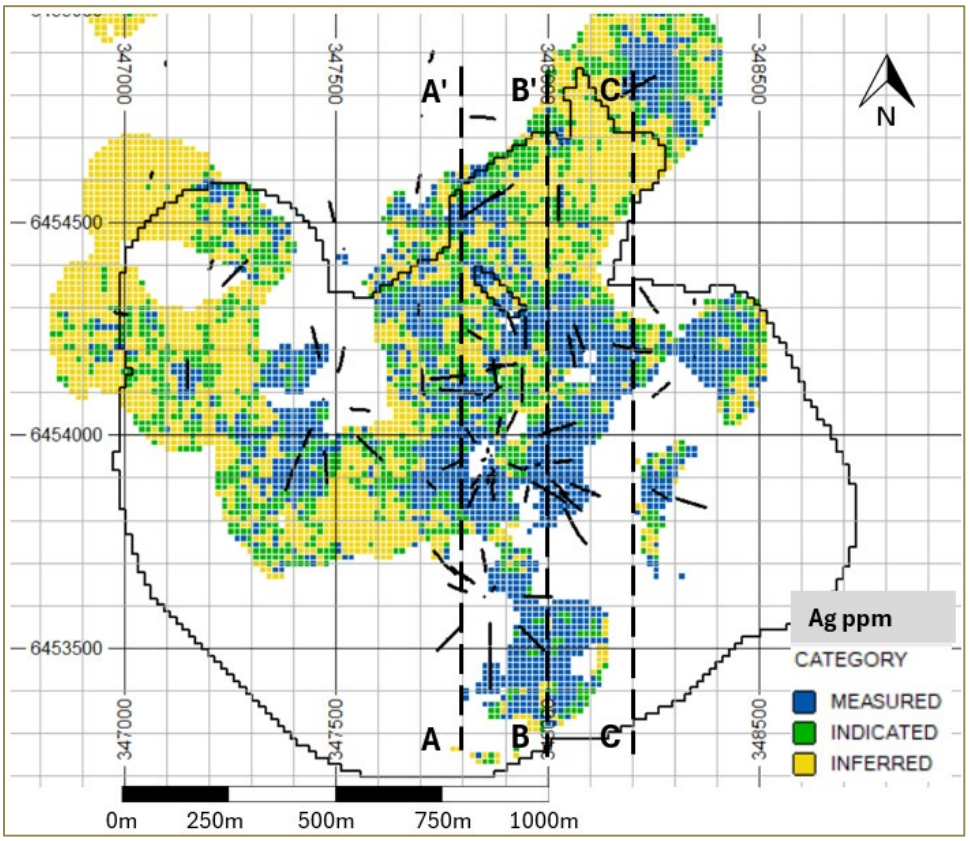


East 348000 Looking West—Block Model Showing Gold Mineral Resource Categories distribution

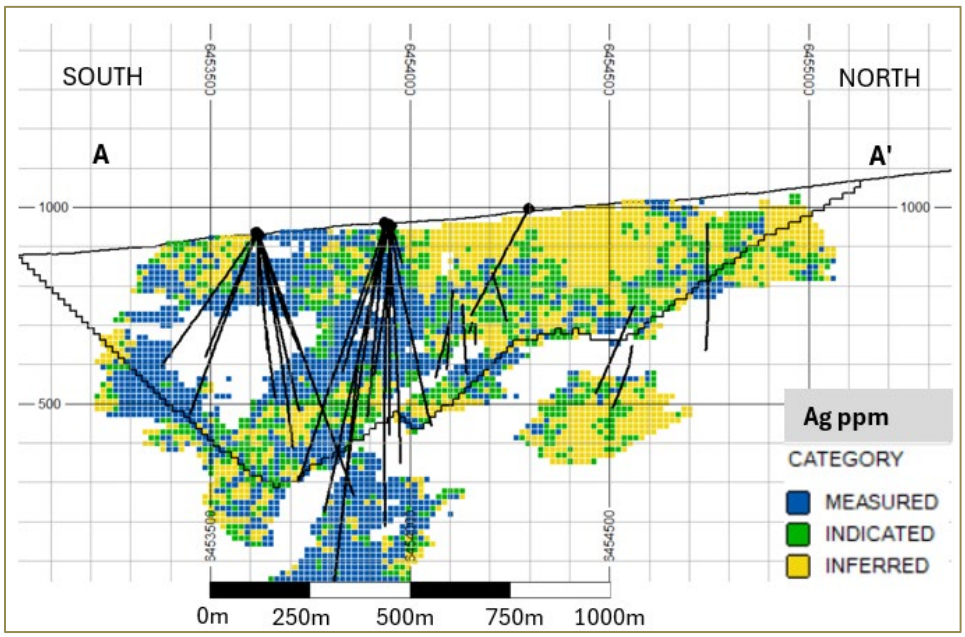


East 348200 Looking West—Block Model Showing Gold Mineral Resource Categories distribution

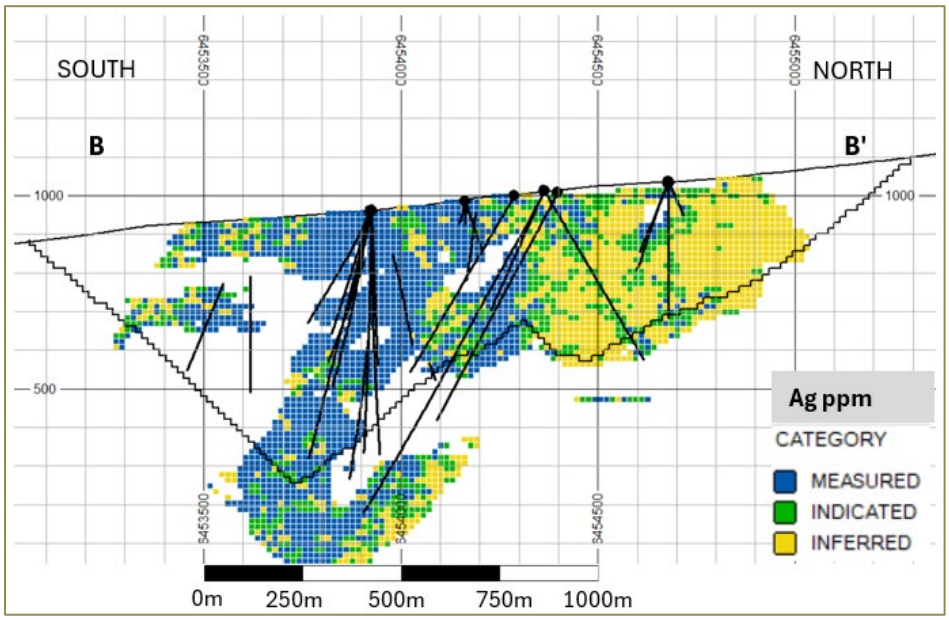
Figure 14-61: Gold Domain (Au ppm) mineral-resource distribution by category (Measured, Indicated, Inferred) within the planned 120 kt/d pit shell for the Hat Project. Block model: MDMRE_040226 (effective date 04/02/2026). Coordinate system: NAD83 / UTM Zone 9N (m). Drill holes are shown clipped to ±70 m from block-model intersection. Caption prepared by QP; data source: Doubleview Database.



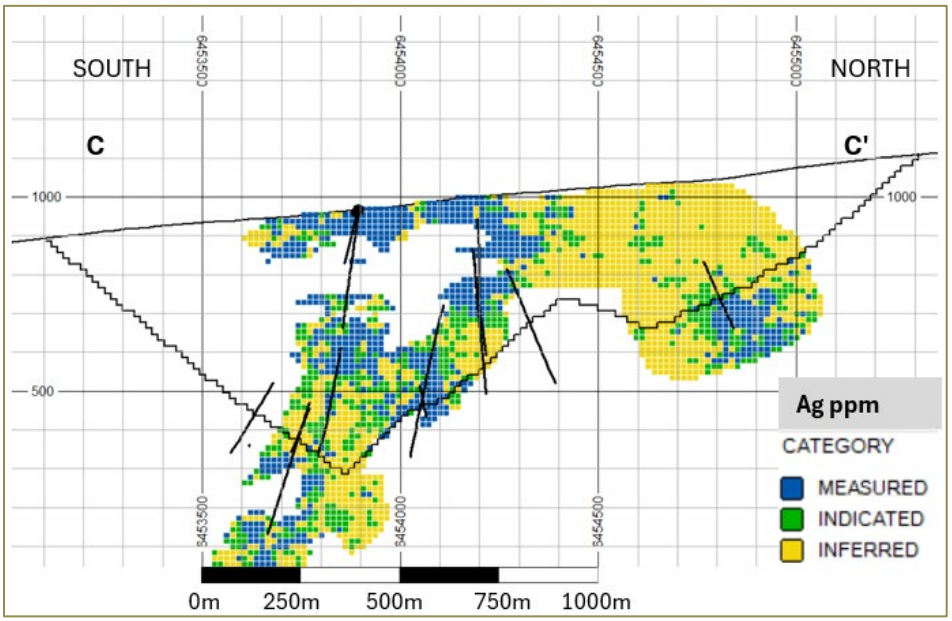
Plan View Elevation 700 masl—Block Model Showing Silver Mineral Resource Categories distribution



East 347800 Looking West—Block Model Showing Silver Mineral Resource Categories distribution

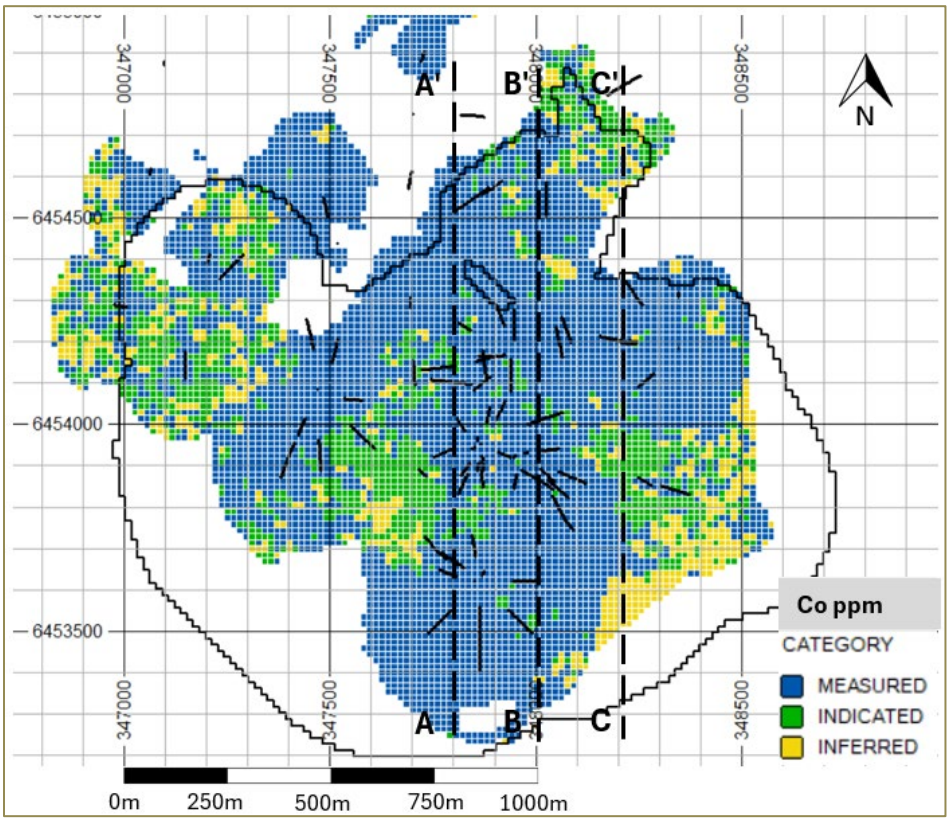


East 348000 Looking West—Block Model Showing Silver Mineral Resource Categories distribution

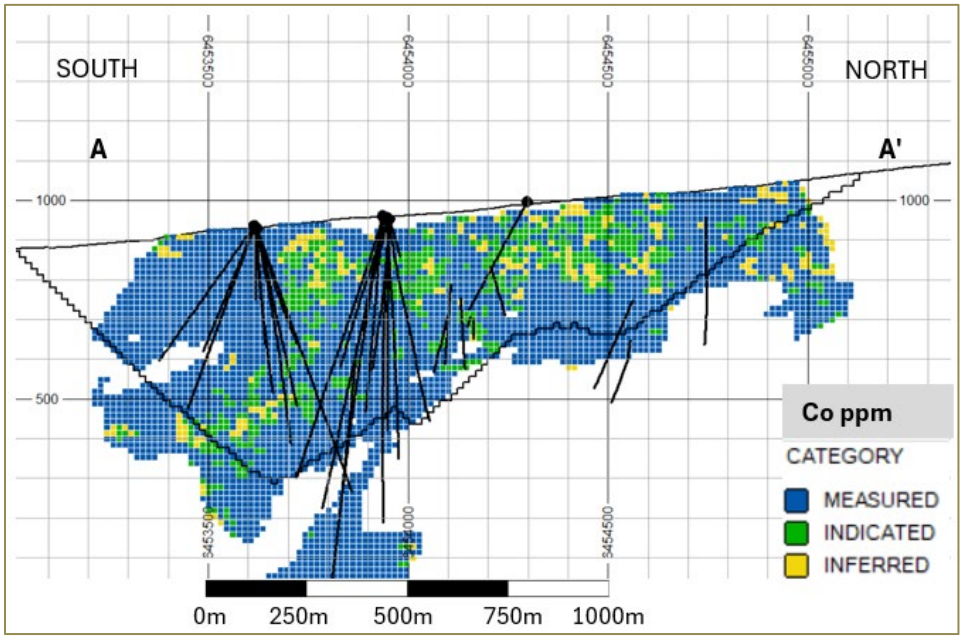


East 348200 Looking West—Block Model Showing Silver Mineral Resource Categories distribution

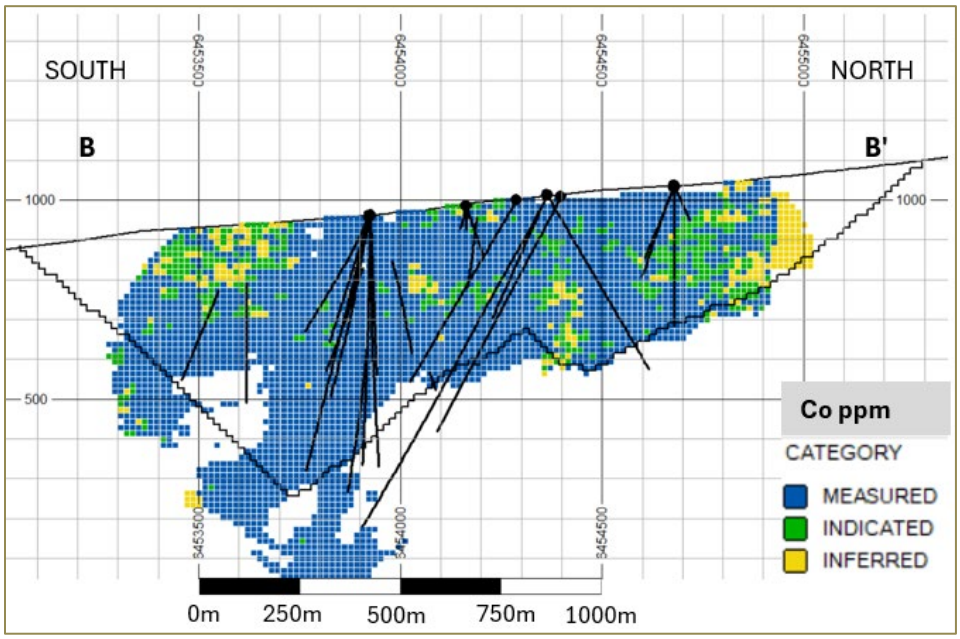
Figure 14-62: Silver Domain (Ag ppm) mineral-resource distribution by category (Measured, Indicated, Inferred) within the planned 120 kt/d pit shell for the Hat Project. Block model: MDMRE_040226 (effective date 04/02/2026). Coordinate system: NAD83 / UTM Zone 9N (m). Drill holes are shown clipped to ±70 m from block-model intersection. Caption prepared by QP; data source: Doubleview Database.



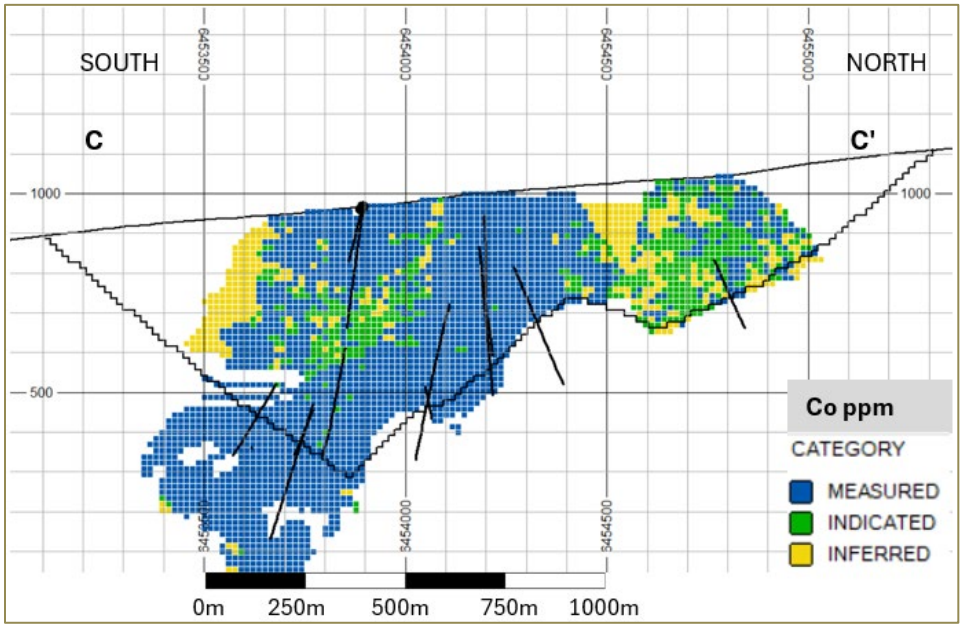
Plan View Elevation 700 masl—Block Model Showing Cobalt Mineral Resource Categories distribution



East 347800 Looking West—Block Model Showing Cobalt Mineral Resource Categories distribution

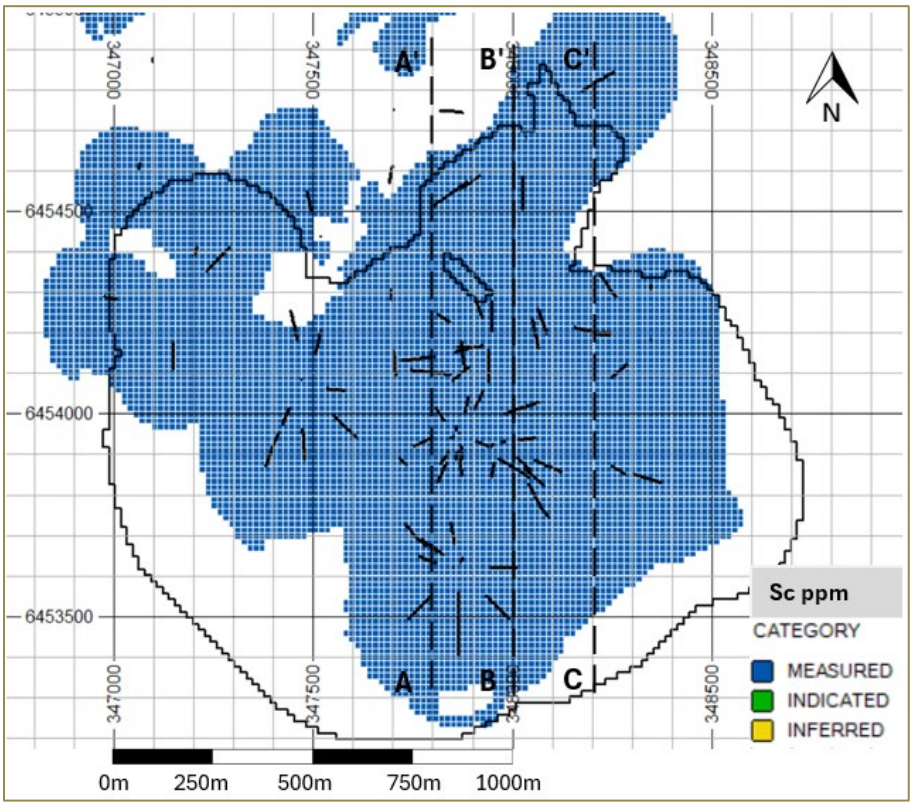


East 348000 Looking West—Block Model Showing Cobalt Mineral Resource Categories distribution

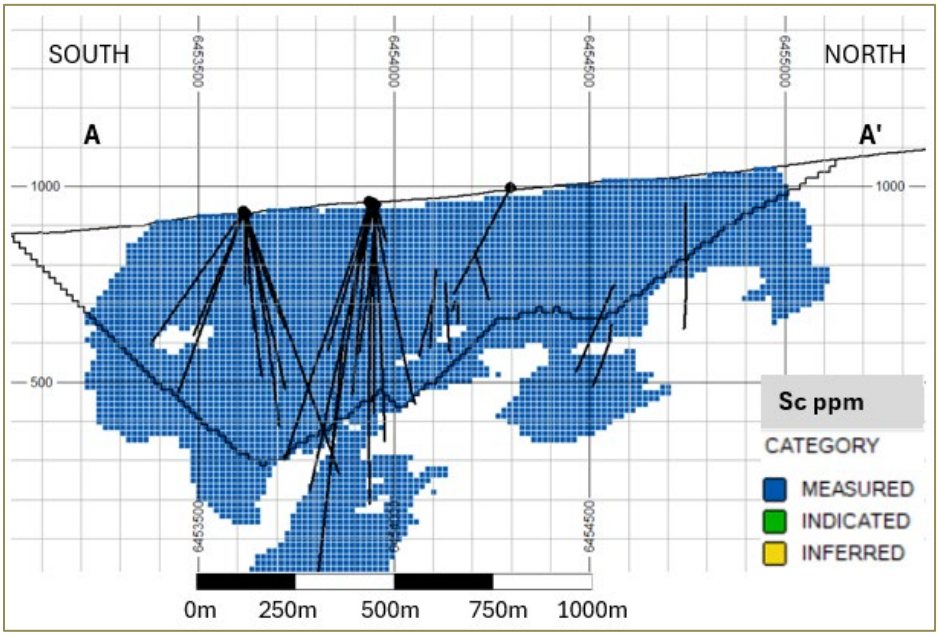


East 348200 Looking West—Block Model Showing Cobalt Mineral Resource Categories distribution

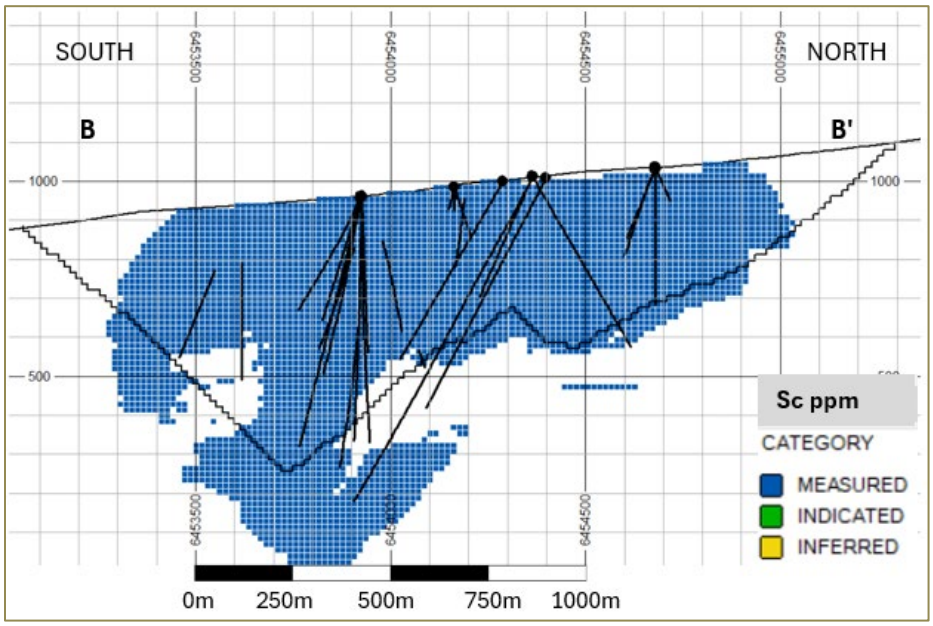
Figure 14-63: Cobalt Domain (Co ppm) mineral-resource distribution by category (Measured, Indicated, Inferred) within the planned 120 kt/d pit shell for the Hat Project. Block model: MDMRE_040226 (effective date 04/02/2026). Coordinate system: NAD83 / UTM Zone 9N (m). Drill holes are shown clipped to ±70 m from block-model intersection. Caption prepared by QP; data source: Doubleview Database.



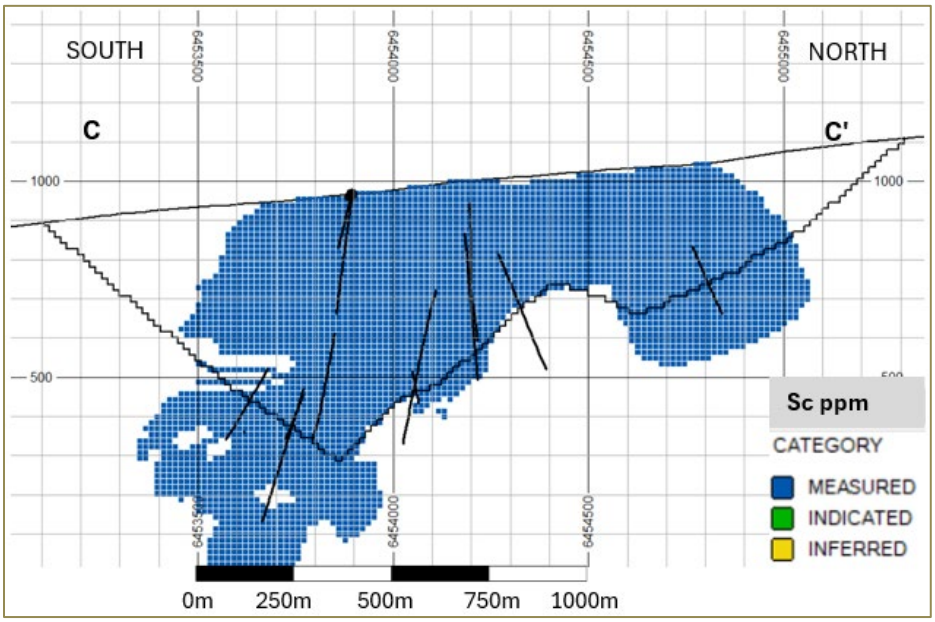
Plan View Elevation 700 masl—Block Model Showing Scandium Mineral Resource Categories distribution



East 347800 looking West—Block Model Showing Scandium Mineral Resource Categories distribution



East 348000 looking West—Block Model Showing Scandium Mineral Resource Categories distribution



East 348200 looking West—Block Model Showing Scandium Mineral Resource Categories distribution

Figure 14-64: Scandium Domain (Sc ppm) mineral-resource distribution by category (Measured, Indicated, Inferred) within the planned 120 kt/d pit shell for the Hat Project. Block model: MDMRE_040226 (effective date 04/02/2026). Coordinate system: NAD83 / UTM Zone 9N (m). Drill holes are shown clipped to ±70 m from block-model intersection. Caption prepared by QP; data source: Doubleview Database.

14.13.5 Validation of Mineral Resource Classification

The classification methods used for the Hat Project, like any approach to assigning confidence to estimates, act as a transfer function. Data and analytical parameters are transferred into a three-dimensional grade distribution. The classification should be considered an asset of the Hat Project, and the quantified confidence assigned to each block must be validated.

Validation consists of using a set of conditional sequential Gaussian simulations used for the estimation of each variable, as discussed in Section 14.12.1. These 51 simulations are evaluated to verify whether the conditions imposed on Mineral Resource classification, specifically estimation error and confidence limits, achieve their objectives at a hypothetical production scale.

Validation through Production-Scale Simulation

Each variable (Au, Ag, Cu, Co, Sc) is validated independently by simulating production-scale panels and translating these panels across the domain, with statistics compiled at each location to build a robust set of production-scale results. The process begins at the scale of a single block and progressively increases the production volume until bimonthly and annual supports are reached. The resulting production-scale statistics are then compared with the initial theoretical assumptions to determine whether the classification thresholds are appropriate or require adjustment.

Production Scale and Classification Assumptions

The classification assumes that, at the appropriate production support, deviations in grade and tonnage should fall within 20% at an 80% confidence limit. Here, deviation refers to the quantified estimation error at different production scales, derived from the tabulation of 51 conditional simulations. These simulations provide average grades and confidence limits for progressively larger production volumes, showing that both estimation error and confidence-interval width decrease as production support increases. Measured Resources correspond to a target production support of approximately two months (3.5 Mt), while Indicated Resources correspond to roughly twelve months (22.5 Mt).

Classification method explicitly incorporates production rate as a factor mitigating individual block estimation error. The base case production rate used for the classification was 60 kt/d. This satisfies the requirements of the classification framework of 20% error at an 80% confidence limit. It is important to note that increasing the production rate only strengthens this conclusion. Larger production supports reduce estimation error and narrow confidence intervals, meaning that any scenario above 60 kt/d will continue to meet or exceed the original classification thresholds.

Implications for Block-Scale Uncertainty

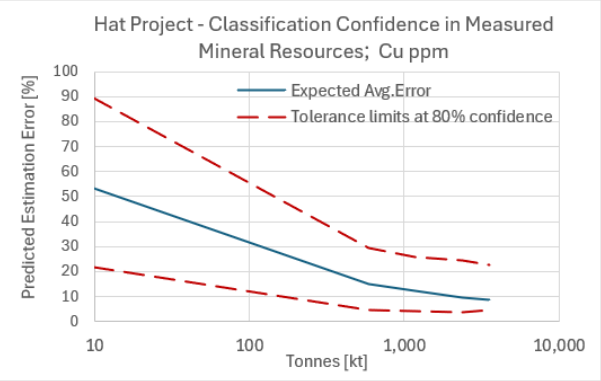
The validation highlights the inherently high uncertainty present at isotropic 15 m block-size support and demonstrates how this uncertainty can be reduced to an objective and measurable target by considering a scheduled production volume. A single block—commonly treated as a small mining unit (SMU)—is not an appropriate scale for expressing Mineral Resource confidence. Reliable predictions emerge only when multiple SMUs are aggregated, reflecting the natural averaging process that occurs at production scale.

Presentation of Results

The following section presents the numerical results and accompanying figures (Figure 14-65 to Figure 14-69) illustrating the behaviour of each variable individually.

Copper—Measured Mineral Resources

Tonnes (kt)	Production Time Equivalency	Avg. Error (%)	Min. Error (%)	Max. Error (%)
10	1 block	53.3	21.6	89.6
589	10 days	15.1	4.6	29.6
1,179	20days	12.1	4.2	25.7
2,357	38 days	9.5	3.9	24.4
3,536	2 months	8.7	4.6	22.5



Copper—Indicated Mineral Resources

Tonnes (kt)	Production Time Equivalency	Avg. Error (%)	Min. Error (%)	Max. Error (%)
10	1 block	61.3	22.9	123.0
589	10 days	23.3	13.0	42.8
1,179	20 days	18.7	9.1	33.5
2,357	38 days	14.8	7.4	31.0
3,536	2 months	13.1	7.6	26.7
22,589	1 year	10.2	5.8	14.4

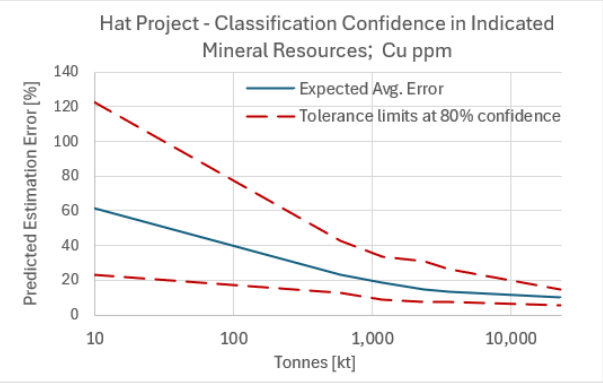
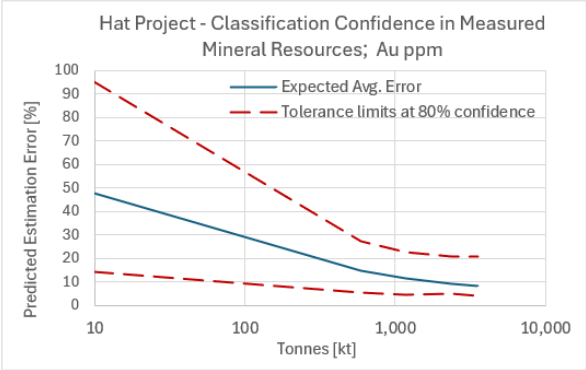


Figure 14-65: Copper Classification Confidence

Gold—Measured Mineral Resources

Tonnes (kt)	Production Time Equivalency	Avg. Error (%)	Min. Error (%)	Max. Error (%)
10	1 block	47.7	14.4	95.2
589	10 days	14.5	5.4	27.2
1,179	20 days	11.4	4.5	22.8
2,357	38 days	9.0	4.9	20.8
3,536	2 months	8.0	4.2	20.8



Gold—Indicated Mineral Resources

Tonnes (kt)	Production Time Equivalency	Avg. Error (%)	Min. Error (%)	Max. Error (%)
10	1 block	55.2	15.8	142.9
589	10 days	23.7	13.3	37.9
1,179	20 days	18.6	10.5	35.2
2,357	38 days	14.5	9.3	27.1
3,536	2 months	12.9	7.4	23.3
22,589	1 year	9.7	6.8	18.3

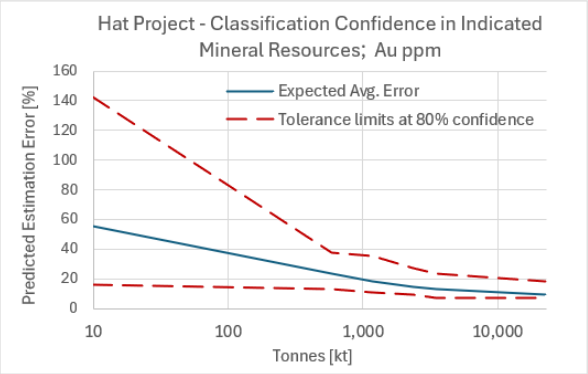
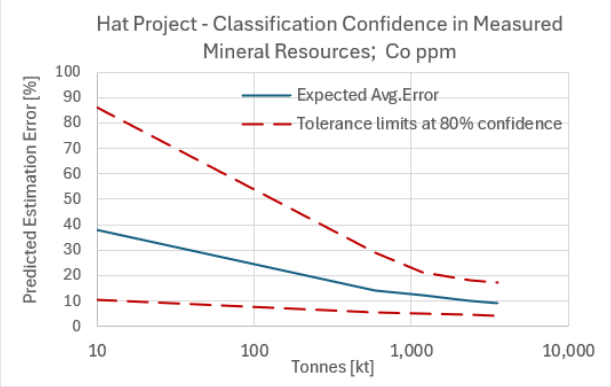


Figure 14-66: Gold Classification Confidence

Cobalt—Measured Mineral Resources

Tonnes (kt)	Production Time Equivalency	Avg. Error (%)	Min. Error (%)	Max. Error (%)
10	1 block	38.1	10.6	86.3
589	10 days	14.2	5.3	28.9
1,179	20 days	12.2	5.1	21.5
2,357	38 days	10.1	4.8	18.3
3,536	2 months	8.9	4.2	17.2



Cobalt—Indicated Mineral Resources

Tonnes (kt)	Production Time Equivalency	Avg. Error (%)	Min. Error (%)	Max. Error (%)
10	1 block	48.6	13.2	104.9
589	10 days	20.1	9.6	32.1
1,179	20 days	16.3	8.2	28.8
2,357	38 days	12.7	6.6	23.4
3,536	2 months	10.8	6.1	23.4
22,589	1 year	7.9	4.9	14.9

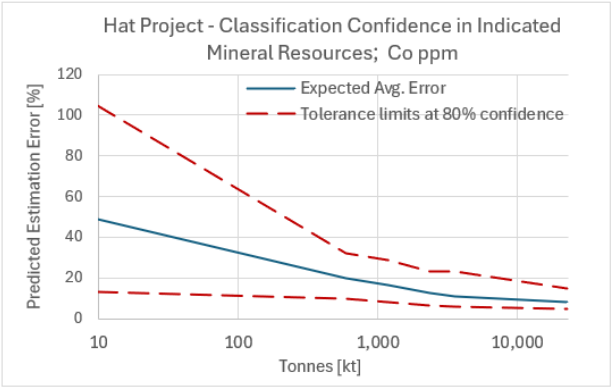
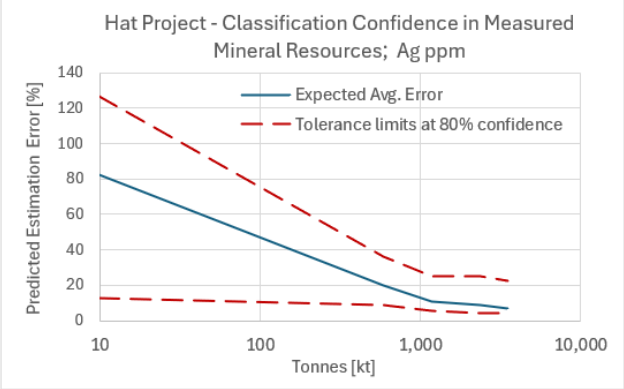


Figure 14-67: Cobalt Classification Confidence

Silver—Measured Mineral Resources

Tonnes (kt)	Production Time Equivalency	Avg. Error (%)	Min. Error (%)	Max. Error (%)
10	1 block	82.5	12.9	126.8
589	10 days	19.7	8.8	36.4
1,179	20 days	11.1	5.7	25.2
2,357	38 days	8.7	4.4	25.2
3,536	2 months	7.1	4.4	22.5



Silver—Indicated Mineral Resources

Tonnes (kt)	Production Time Equivalency	Avg. Error (%)	Min. Error (%)	Max. Error (%)
10	1 block	59.3	13.3	114.5
589	10 days	21.2	10.4	34.9
1,179	20 days	17.3	9.0	32.7
2,357	38 days	13.5	6.6	28.2
3,536	2 months	11.8	6.1	25.0
22,589	1 year	8.2	5.0	15.8

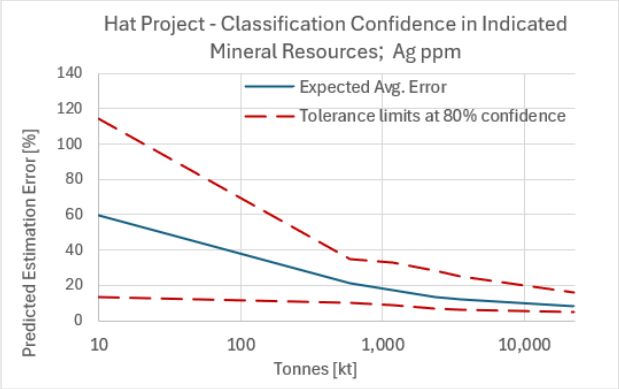


Figure 14-68: Silver Classification Confidence

Scandium—Measured Mineral Resources

Tonnes (kt)	Production Time Equivalency	Avg. Error (%)	Min. Error (%)	Max. Error (%)
10	1 block	22.3	9.9	40.9
589	10 days	7.6	2.5	17.0
1,179	20 days	6.8	2.2	16.9
2,357	38 days	5.8	2.3	13.3
3,536	2 months	5.2	2.2	11.6

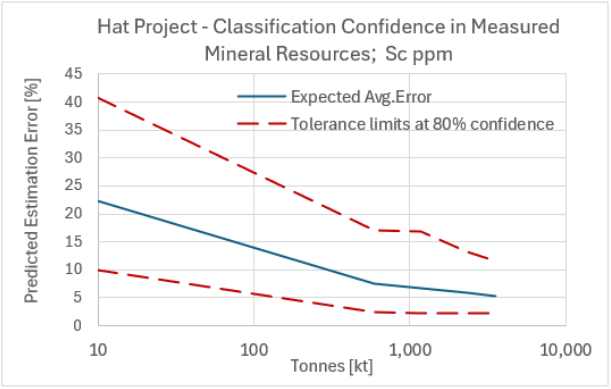


Figure 14-69: Scandium Classification Confidence

14.13.6 Conclusion on Classification Validation

In all the analyzed cases, it has been demonstrated that estimation error can be reduced to within the 20% threshold at the 80% confidence limit, confirming that the applied classification method is robust and reliable at the production scales considered. The validation exercise also highlights how uncertainty at block support can be effectively managed and transformed into predictable outcomes through increased production volume. However, it is important to recognize that future production requirements typically demand tighter performance—often in the range of 10%–15% error at approximately 90% confidence. Achieving these operational standards will require additional drilling to further constrain local variability and reduce uncertainty ahead of mine development. Together, these results provide confidence that the Hat Project Mineral Resource estimates are suitable for supporting mine planning decisions and future economic evaluations, consistent with the assumptions present in the current PEA.

14.13.7 Contribution of Different Elements to Resource Classification

The Hat Project is supported by five commodities: gold, copper, cobalt, silver, and scandium. The current classification framework considers the individual characteristics of each commodity. However, for the purposes of combined confidence supporting the CuEq variable, only copper (62%) and gold (38%) are considered to ensure meaningful interpretation.

To ensure consistent classification, a copper–gold category matrix was applied, whereby the CuEq category (CAT_EQ)—whose local configuration is shown in Figure 14-70—is derived from the combination of the individual copper (CAT_CU) and gold (CAT_AU) categories. The categorical values 1, 2, and 3 correspond to Measured, Indicated, and Inferred confidence levels, respectively.

The applied matrix is as follows:

if CAT_CU=1 and CAT_AU=1 → CAT_EQ=1

if CAT_CU=1 and CAT_AU=2 → CAT_EQ=1

if CAT_CU=1 and CAT_AU=3 → CAT_EQ=2

if CAT_CU=2 and CAT_AU=1 → CAT_EQ=1

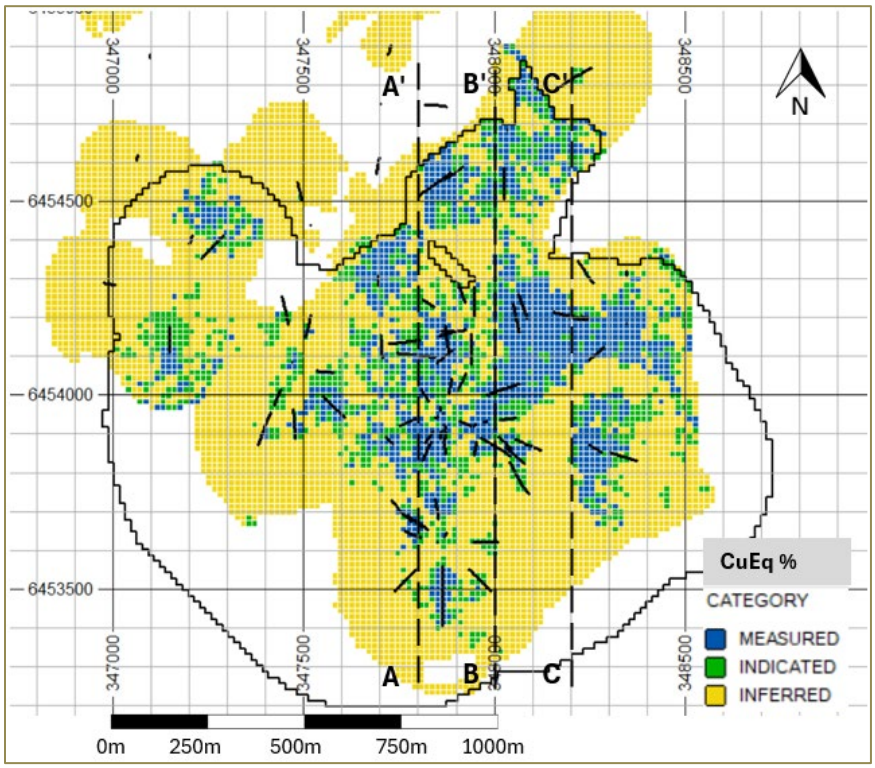
if CAT_CU=2 and CAT_AU=2 → CAT_EQ=2

if CAT_CU=2 and CAT_AU=3 → CAT_EQ=3

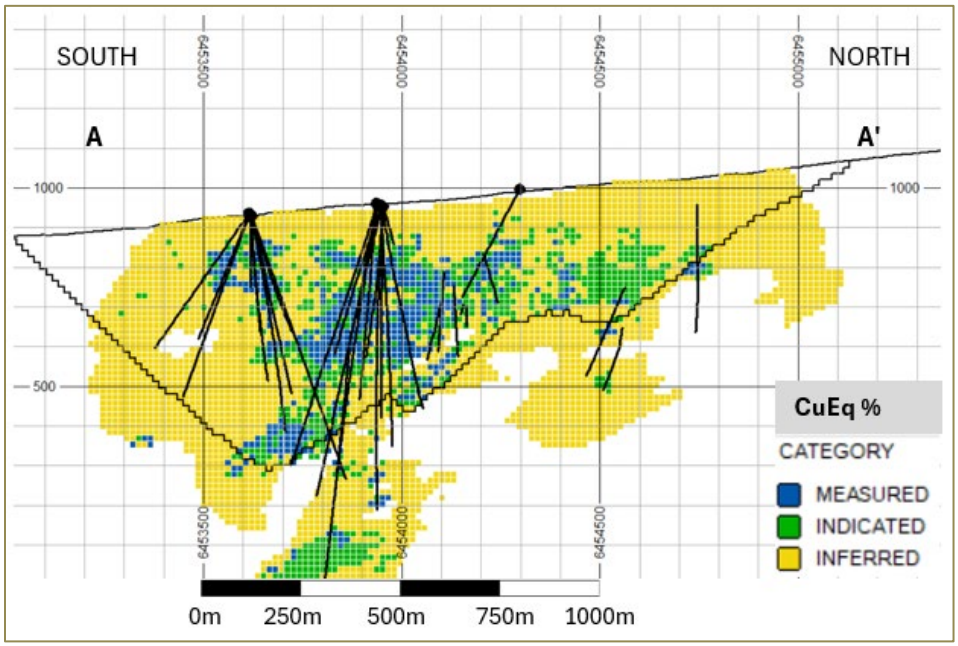
if CAT_CU=3 and CAT_AU=1 → CAT_EQ=2

if CAT_CU=3 and CAT_AU=2 → CAT_EQ=3

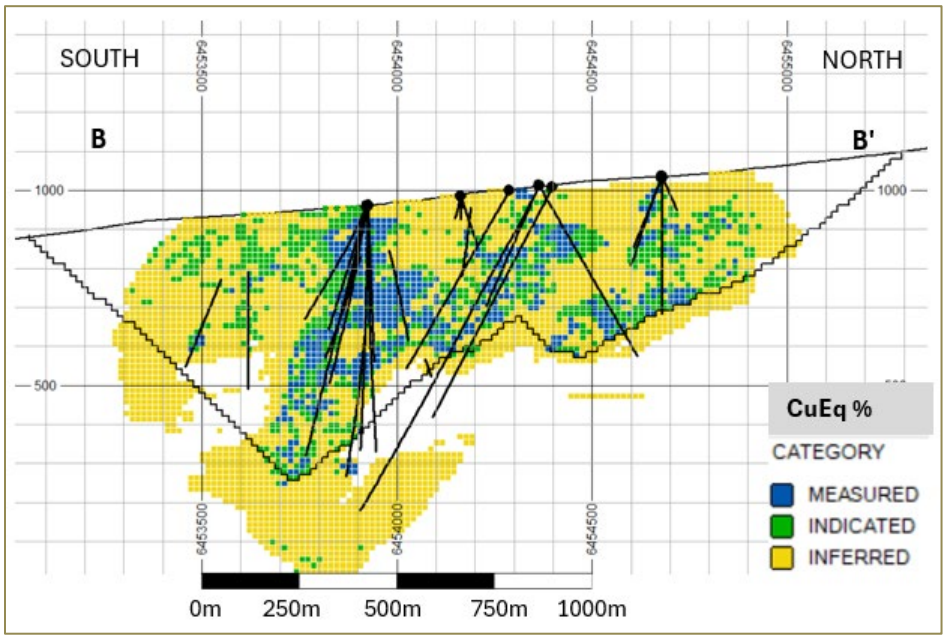
if CAT_CU=3 and CAT_AU=3 → CAT_EQ=3



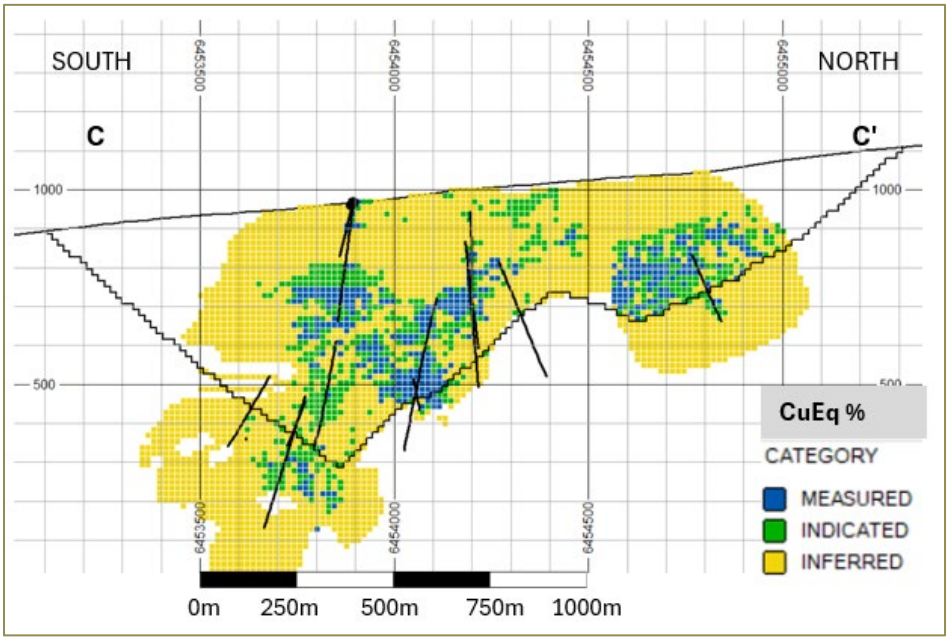
Plan View Elevation 700 masl—Block Model Showing Equivalent Copper Mineral Resource Categories distribution



East 347800 Looking West—Block Model Showing Equivalent Copper Mineral Resource Categories distribution



East 348000 Looking West—Block Model Showing Equivalent Copper Mineral Resource Categories distribution



East 348200 Looking West—Block Model Showing Equivalent Copper Mineral Resource Categories distribution

Figure 14-70: Equivalent-Copper (CuEq) mineral-resource distribution by category (Measured, Indicated, Inferred) within the planned 120 kt/d pit shell for the Hat Project. Block model: MDMRE_040226 (effective date 04/02/2026). Coordinate system: NAD83 / UTM Zone 9N (m). Drill holes are shown clipped to ±70 m from block-model intersection. Caption prepared by QP; data source: Doubleview Database.

14.14 Copper Equivalent Formula

Individual grades of silver, gold, copper, and cobalt estimates were consolidated into the CuEq formula. The formula integrates the contribution of each variable grade, price, and metallurgical recovery into a single copper-equivalent grade, allowing direct comparison and resource tabulation.

The formula is presented below, with input variables expressed in ppm and the output variable expressed in % CuEq.

$$CuEq = (((AG \times 46.0 \times 0.68) / 31.1035) + ((AU \times 3200 \times 0.89) /31.1035) + 0.0001 \times (CO \times 20.0 \times 0.78 \times 22.0462) + 0.0001 \times (CU \times 4.8 \times 22.0462 \times 0.85)) / (4.8 \times 22.0462 \times 0.85)$$

14.14.1 Copper Equivalent Domain and Reasonable Prospects for Eventual Economic Extraction

The CuEq domain is a compound-grade domain derived from the combined contributions of the copper, gold, silver, and cobalt mineralized domains. CuEq represents the integrated economic value of the individual gold, silver, copper and cobalt grades, calculated using metal prices and metallurgical recoveries.

The CuEq domain fulfills the requirement for reasonable prospects for eventual economic extraction (RPEEE) in the same manner as the individual metal domains because:

- All blocks included in the CuEq domain lie within the same Lerchs–Grossmann pit shell used to demonstrate RPEEE for the Mineral Resource.
- The CuEq calculation uses the same metal prices, metallurgical recoveries, and operating cost assumptions applied to the individual metal domains.
- The CuEq domain does not introduce material outside the economic pit shell and does not increase tonnage beyond what is supported by the RPEEE pit constraint.
- The CuEq domain is fully supported by geological interpretation and honours the same lithological, structural, and alteration controls used to define the individual metal domains.
- The CuEq domain is used solely as an economic composite for classification and reporting purposes and does not replace the underlying gold, silver, copper, and cobalt grade domains.

On this basis, the CuEq domain meets the CIM requirement for RPEEE and is considered appropriate for use in the reporting of Mineral Resources.

14.15 Mineral Resources Tabulation and Sensitivity to Cut-Off Grade

A summary of the estimation and classification exercises is presented in the form of a Mineral Resources tabulation in Table 14-16, Table 14-17, Table 14-18and Table 14-19, reported as a function of resource category and CuEq cut-off grade.

14.15.1 Basis of Tabulation

- Copper-equivalent (CuEq) grades were applied as the cut-off criterion and used as the classification threshold for the tabulation of Mineral Resources.
- An optimized pit design (PIT120k) was applied to constrain resources, ensuring that the reported figures reflect RPEEE.

Table 14-16to Table 14-19present the Measured, Indicated, Measured + Indicated, and Inferred Mineral Resources, including CuEq, individual metal grades, and density, consistent with the estimation method and classification criteria within an optimized pit design of 120 kt/d.

14.15.2 Tonnage/Grade Curve for Single vs. Compound Variable

When dealing with a single variable, tonnage/grade curves typically present as simple and smooth functions, reflecting the behavior of a single grade population. By contrast, the CuEq is a compound measure that integrates four elements (Ag, Au, Cu, Co). As a result, the tonnage/grade curves for CuEq in Figure 14-71 may display uneven shapes. This should not be interpreted as an error, but rather as a natural outcome of combining multiple grade distributions into a single equivalent value. Subsequently, the underlying grades and tonnages do not behave as a single homogeneous population, and variability in curve shapes is expected.

Table 14-16: Pit-Constrained Measured Mineral Resource Reported Across CuEq Cut-Off Grades

Measured Resources CuEq Cut-Off	Tonnage (Mt)	Average Grade							Metal Content				
		CuEq (%)	Cu (%)	Au (g/t)	Co (%)	Ag (g/t)	Sc (g/t)	Sg (g/cm³)	Cu (Mlb)	Au (koz)	Co (Mlb)	Ag (koz)	Sc[1] (Mt)
0.05	296	0.41	0.20	0.17	0.007	0.35	28.82	2.93	1,134	1,449	38.0	2,259	37
0.10	289	0.42	0.21	0.17	0.008	0.35	28.80	2.93	1,130	1,444	37.5	2,240	36
0.15	281	0.43	0.21	0.18	0.008	0.36	28.79	2.93	1,123	1,433	36.6	2,207	35
0.20 b.c.	**272**	**0.44**	**0.22**	**0.18**	**0.008**	**0.37**	**28.79**	**2.93**	**1,113**	**1,408**	**35.6**	**2,170**	**34**
0.25	257	0.45	0.23	0.18	0.008	0.37	28.79	2.93	1,091	1,359	33.9	2,103	32
0.30	242	0.46	0.23	0.19	0.008	0.38	28.79	2.93	1,059	1,306	32.2	2,020	30
0.35	216	0.47	0.24	0.19	0.008	0.39	28.74	2.93	982	1,201	29.0	1,843	27
0.40	172	0.50	0.26	0.20	0.008	0.40	28.66	2.93	828	1,000	23.5	1,514	21
0.45	114	0.54	0.28	0.22	0.008	0.42	28.55	2.93	595	709	16.0	1,053	14
0.50	62	0.59	0.31	0.24	0.009	0.45	28.41	2.93	359	426	9.1	611	8

Notes: [1] Scandium tonnages represent 12.5% of the mineralized material by category, reflecting the proportion of tailings expected to be processed through a dedicated scandium leach circuit under current metallurgical design constraints.
b.c.= base case.
Rounding may result in minor variations between individual values and totals; such differences are not considered material to the Mineral Resource estimate.
Metal contents for Cu, Au, Co, and Ag have been calculated using the following metallurgical recovery factors: Cu = 0.85, Au = 0.89, Co = 0.78, and Ag = 0.68.
CuEq = [(((Ag × 46.0 × 0.68)/31.1035) + ((Au × 3200 × 0.89)/31.1035) + 0.0001 × (Co × 20.0 × 0.78 × 22.0462) + 0.0001 × (Cu × 4.8 × 22.0462 × 0.85))/(4.8 × 22.0462 × 0.85)] where all input variables are expressed in (ppm), and CuEq is expressed in percent (%).

Table 14-17: Pit-Constrained Indicated Mineral Resource Reported Across CuEq Cut-Off Grades

Indicated Resources CuEq Cut-Off	Tonnage (Mt)	Average Grade							Metal Content				
		CuEq (%)	Cu (%)	Au (g/t)	Co (%)	Ag (g/t)	Sc (g/t)	Sg (g/cm³)	Cu (Mlb)	Au (koz)	Co (Mlb)	Ag (koz)	Sc[1] (Mt)
0.05	375	0.40	0.19	0.17	0.007	0.37	28.82	2.92	1,343	1,859	48.3	3,037	47
0.10	358	0.42	0.20	0.18	0.008	0.38	28.80	2.92	1,333	1,846	46.8	2,978	45
0.15	344	0.43	0.20	0.19	0.008	0.39	28.77	2.92	1,319	1,829	45.3	2,914	43
0.20 b.c.	**337**	**0.43**	**0.21**	**0.19**	**0.008**	**0.39**	**28.76**	**2.92**	**1,311**	**1,811**	**44.5**	**2,882**	**42**
0.25	297	0.46	0.22	0.20	0.008	0.40	28.77	2.92	1,252	1,675	39.8	2,626	37
0.30	240	0.50	0.25	0.21	0.008	0.42	28.85	2.93	1,138	1,461	32.6	2,188	30
0.35	207	0.53	0.27	0.23	0.008	0.42	28.92	2.93	1,042	1,332	27.9	1,889	26
0.40	184	0.55	0.28	0.23	0.008	0.42	28.93	2.93	962	1,233	24.9	1,685	23
0.45	150	0.58	0.29	0.25	0.008	0.43	28.90	2.93	827	1,064	20.7	1,402	19
0.50	103	0.63	0.32	0.27	0.008	0.44	28.86	2.93	616	796	14.7	1,000	13

Notes: [1] Scandium tonnages represent 12.5% of the mineralized material by category, reflecting the proportion of tailings expected to be processed through a dedicated scandium leach circuit under current metallurgical design constraints.
b.c.= base case.
Rounding may result in minor variations between individual values and totals; such differences are not considered material to the Mineral Resource estimate.
Metal contents for Cu, Au, Co, and Ag have been calculated using the following metallurgical recovery factors: Cu = 0.85, Au = 0.89, Co = 0.78, and Ag = 0.68.
CuEq = [(((Ag × 46.0 × 0.68)/31.1035) + ((Au × 3200 × 0.89)/31.1035) + 0.0001 × (Co × 20.0 × 0.78 × 22.0462) + 0.0001 × (Cu × 4.8 × 22.0462 × 0.85))/(4.8 × 22.0462 × 0.85)] where all input variables are expressed in (ppm), and CuEq is expressed in percent (%).

Table 14-18: Pit-Constrained Measured and Indicated Mineral Resources Reported Across CuEq Cut-Off Grades

M + I Resources CuEq Cut-off	Tonnage (Mt)	Average Grade							Metal Content				
		CuEq (%)	Cu (%)	Au (g/t)	Co (%)	Ag (g/t)	Sc (g/t)	Sg (g/cm³)	Cu (Mlb)	Au (koz)	Co (Mlb)	Ag (koz)	Sc[1] (Mt)
0.05	671	0.41	0.20	0.17	0.007	0.36	28.82	2.92	2,477	3,308	86.3	5,296	84
0.10	647	0.42	0.20	0.18	0.008	0.37	28.80	2.92	2,463	3,290	84.2	5,218	81
0.15	625	0.43	0.21	0.18	0.008	0.37	28.78	2.92	2,442	3,263	81.9	5,121	78
0.20 b.c.	**609**	**0.43**	**0.21**	**0.18**	**0.008**	**0.38**	**28.77**	**2.92**	**2,424**	**3,219**	**80.1**	**5,052**	**76**
0.25	554	0.45	0.23	0.19	0.008	0.39	28.78	2.93	2,343	3,034	73.7	4,728	69
0.30	483	0.48	0.24	0.20	0.008	0.40	28.82	2.93	2,197	2,767	64.8	4,207	60
0.35	423	0.50	0.26	0.21	0.008	0.40	28.82	2.93	2,024	2,533	57.0	3,731	53
0.40	355	0.53	0.27	0.22	0.008	0.41	28.80	2.93	1,789	2,232	48.4	3,199	44
0.45	264	0.56	0.29	0.23	0.008	0.43	28.75	2.93	1,422	1,773	36.7	2,455	33
0.50	165	0.61	0.31	0.26	0.008	0.45	28.69	2.93	975	1,222	23.7	1,611	21

Notes: [1] Scandium tonnages represent 12.5% of the mineralized material by category, reflecting the proportion of tailings expected to be processed through a dedicated scandium leach circuit under current metallurgical design constraints.
b.c.= base case.
Rounding may result in minor variations between individual values and totals; such differences are not considered material to the Mineral Resource estimate.
Metal contents for Cu, Au, Co, and Ag have been calculated using the following metallurgical recovery factors: Cu = 0.85, Au = 0.89, Co = 0.78, and Ag = 0.68.
CuEq = [(((Ag × 46.0 × 0.68)/31.1035) + ((Au × 3200 × 0.89)/31.1035) + 0.0001 × (Co × 20.0 × 0.78 × 22.0462) + 0.0001 × (Cu × 4.8 × 22.0462 × 0.85))/(4.8 × 22.0462 × 0.85)] where all input variables are expressed in (ppm), and CuEq is expressed in percent (%).

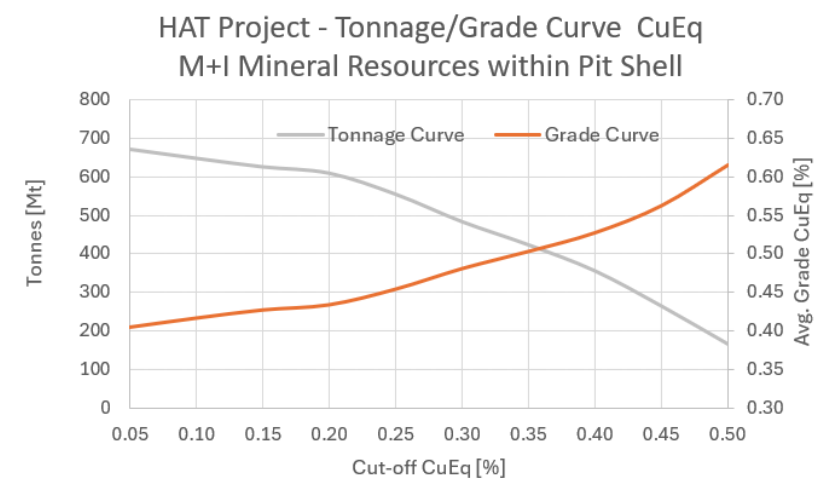


Figure 14-71: Measured and Indicated Mineral Resources Tonnage/Grade Curve

Table 14-19: Pit-Constrained Inferred Mineral Resource Reported Across CuEq Cut-Off Grades

Inferred Resources CuEq Cut-off	Tonnage (Mt)	Average Grade							Metal Content				
		CuEq (%)	Cu (%)	Au (g/t)	Co (%)	Ag (g/t)	Sc (g/t)	Sg (g/cm³)	Cu (Mlb)	Au (koz)	Co (Mlb)	Ag (koz)	Sc[1] (Mt)
0.05	1,438	0.22	0.10	0.10	0.007	0.32	28.86	2.91	2,619	3,916	168.5	10,117	180
0.10	1,199	0.25	0.11	0.11	0.007	0.34	28.85	2.91	2,468	3,735	147.2	8,943	150
0.15	702	0.34	0.15	0.16	0.008	0.37	28.76	2.91	1,996	3,128	91.1	5,695	88
0.20 b.c.	**503**	**0.41**	**0.18**	**0.19**	**0.008**	**0.38**	**28.69**	**2.92**	**1,720**	**2,768**	**66.2**	**4,192**	**63**
0.25	433	0.44	0.20	0.21	0.008	0.38	28.65	2.92	1,595	2,582	57.2	3,639	54
0.30	355	0.48	0.22	0.22	0.008	0.39	28.66	2.92	1,438	2,282	47.6	3,036	44
0.35	272	0.53	0.24	0.24	0.008	0.40	28.73	2.92	1,242	1,894	37.2	2,398	34
0.40	221	0.56	0.27	0.25	0.008	0.42	28.83	2.93	1,106	1,607	30.5	2,003	28
0.45	183	0.59	0.29	0.26	0.008	0.43	28.89	2.93	980	1,384	25.6	1,709	23
0.50	141	0.62	0.30	0.28	0.008	0.44	28.90	2.93	802	1,122	20.1	1,345	18

Notes:
[1] Scandium tonnages represent 12.5% of the mineralized material by category, reflecting the proportion of tailings expected to be processed through a dedicated scandium leach circuit under current metallurgical design constraints.
b.c.= base case.
Rounding may result in minor variations between individual values and totals; such differences are not considered material to the Mineral Resource estimate.
Metal contents for Cu, Au, Co, and Ag have been calculated using the following metallurgical recovery factors: Cu = 0.85, Au = 0.89, Co = 0.78, and Ag = 0.68.
CuEq = [(((Ag × 46.0 × 0.68)/31.1035) + ((Au × 3200 × 0.89)/31.1035) + 0.0001 × (Co × 20.0 × 0.78 × 22.0462) + 0.0001 × (Cu × 4.8 × 22.0462 × 0.85))/(4.8 × 22.0462 × 0.85)] where all input variables are expressed in (ppm), and CuEq is expressed in percent (%).

14.16 Reasonable Prospect for Eventual Economic Extraction

The RPEEE assessment integrates geological, geostatistical, metallurgical, mining, economic, environmental, and jurisdictional considerations, consistent with the CIM Definition Standards and

NI 43-101 requirements. The following discussion summarizes the basis on which the QP considers the Hat Project's Mineral Resources to have reasonable prospects for eventual economic extraction.

14.16.1 Economic Parameters

To satisfy the requirement for RPEEE, the Mineral Resource is reported within a conceptual open pit shell generated using the Lerchs-Grossmann algorithm. The optimization parameters used are consistent with the economic assumptions of this PEA.

Table 14-20: Pit Optimization Parameters

Parameters	Values
Au Price US$/oz	3200
Ag Price US$/oz	46.0
Cu Price US$/lb	4.8
Co Price US$/lb	20.0
Au Recovery %	89.0
Ag Recovery %	68.0
Cu Recovery %	85.0
Co Recovery %	78.0
Pit Slope Angles	Defined per geotechnical Zone (OSA 36-45)
Mining Cost $/t (OP)	3.01 +0.015/Bench
Processing Cost $/t Milled	7.93
G&A Cost $/t Milled	2.90

The formula used to calculate CuEq for the 2025 Mineral Resource estimate is:

$$CuEq = [(((AG \times 46.0 \times 0.68) / 31.1035) + ((AU \times 3200 \times 0.89) / 31.1035) + 0.0001 \times (CO \times 20.0 \times 0.78 \times 22.0462) + 0.0001 \times (CU \times 4.8 \times 22.0462 \times 0.85)) / (4.8 \times 22.0462 \times 0.85)]$$

14.16.2 Resource Block Model

The resource block model for the Hat Project was constructed using a 15 × 15 × 15 m block size. The model incorporates estimation domains for the targeted commodities (Ag, Au, Cu, Co, and Sc), estimated grades, density, geological units, individual classification fields for each variable, and a compound classification for the CuEq variable. A technical summary of the Hat Project Mineral Resource model is provided in Table 14-21.

Table 14-21: Technical Summary of the Hat Project Block Model

Coordinates	X	Y	Z
Model Origin UTM	346750	6453000	0
Block Size (m)	15	15	15
Number of Blocks	136	150	75
Model Extents (m)	2,040	2,250	1,125
Rotation	0°	0°	0°

14.16.3 Mineral Resource Statement

The updated Mineral Resource Estimate for the Hat Project is summarized in Table 14-22 at a base case cut-off grade of 0.20% CuEq.

Table 14-22: Hat Project Mineral Resource Estimate at 0.20% CuEq (Effective February 4, 2026)

Resources Classification	Tonnage (Mt)	Average Grade							Metal Content				
		CuEq (%)	Cu (%)	Au (g/t)	Co (%)	Ag (g/t)	Sc (g/t)	Sg (g/cm³)	Cu (Mlb)	Au (koz)	Co (Mlb)	Ag (koz)	Sc[1] (Mt)
Measured	272	0.44	0.22	0.18	0.008	0.37	28.79	2.93	1,113	1,408	35.6	2,170	34
Indicated	337	0.43	0.21	0.19	0.008	0.39	28.76	2.92	1,311	1,811	44.5	2,882	42
Total M+I	609	0.43	0.21	0.18	0.008	0.38	28.77	2.92	2,424	3,219	80.1	5,052	76
Inferred	503	0.41	0.18	0.19	0.008	0.38	28.69	2.92	1,720	2,768	66.2	4,192	63

Notes: [1]Scandium tonnages represent 12.5% of the mineralized material by category, reflecting the proportion of tailings expected to be processed through a dedicated scandium leach circuit under current metallurgical design constraints.

- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
- Inferred Mineral Resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that any part of the Inferred Mineral Resource will be converted to a higher category.
- The effective date of the Mineral Resource estimate is February 4, 2026.
- Metal contents for Cu, Au, Co, and Ag have been calculated using the following metallurgical recovery factors: Cu = 0.85, Au = 0.89, Co = 0.78, and Ag = 0.68.
- Mineral Resources are reported within optimized open pit constraints and 0.2% CuEq cut-off grade, based on a $7.93/t milled processing cost and C$2.9/t milled general and administrative (G&A) cost, with a mining cost of C$3.01/t + incremental mining cost increasing by C$0.015/t for every bench below the reference level of 1,125 mRL.
- CuEq calculations do not include scandium. The formula used to calculate CuEq is:
  CuEq = [(((Ag × 46.0 × 0.68)/31.1035) + ((Au × 3200 × 0.89)/31.1035) + 0.0001 × (Co × 20.0 × 0.78 × 22.0462) + 0.0001 × (Cu × 4.8 × 22.0462 × 0.85))/(4.8 × 22.0462 × 0.85)] where all input variables are expressed in (ppm), and CuEq is expressed in percent (%).
- Rounding may result in minor variations between individual values and totals; such differences are not considered material to the Mineral Resource estimate.
- Mineral Resource classification reflects the level of geological confidence and satisfies the uncertainty criteria appropriate for exploration and resource development. Additional drilling will be required to reduce uncertainty to the level expected for production planning.
- The Mineral Resource estimate reflects the geological interpretation, drill-hole spacing, and estimation parameters available at the time of modelling. Any additional drilling is expected to influence the current outcome by improving confidence in the estimates and refining the geometry of the mineralized domains.

14.16.4 Comparison to Previous Estimate

The July 2024 Mineral Resource estimate did not report any Measured Mineral Resources. Relative to the 2024 Indicated category, this updated estimate includes a 125% increase in tonnage and a 138% increase in contained CuEq metal. These gains reflect the addition of 18,323 m of new drilling, which expanded knowledge of the mineralized system at depth and along the eastern portion of the deposit.

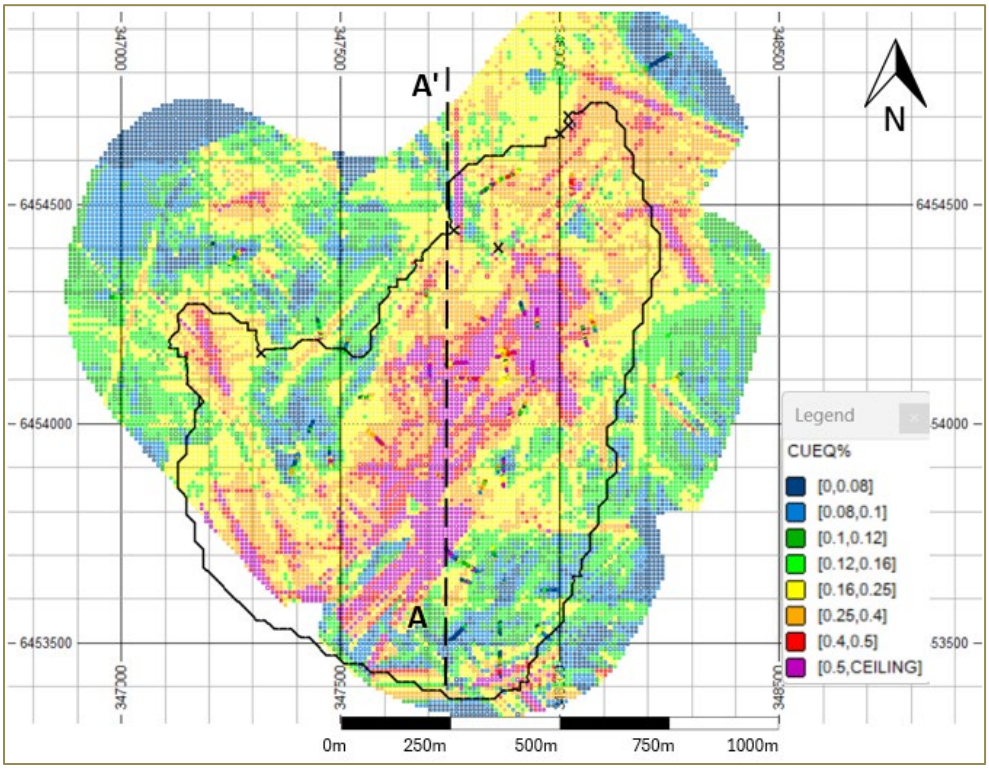
When considering the combined Measured and Indicated categories, this Mineral Resource estimate shows a 306% increase in total tonnes and an 331% increase in contained CuEq metal. Figure 14-72 provides the plan-view and cross-section comparison between the 2024 Mineral Resource estimate and this updated Mineral Resource estimate.

Within the Inferred category, the 2026 Mineral Resource estimate reports a 5% increase in total tonnage and a 26% increase in contained CuEq.

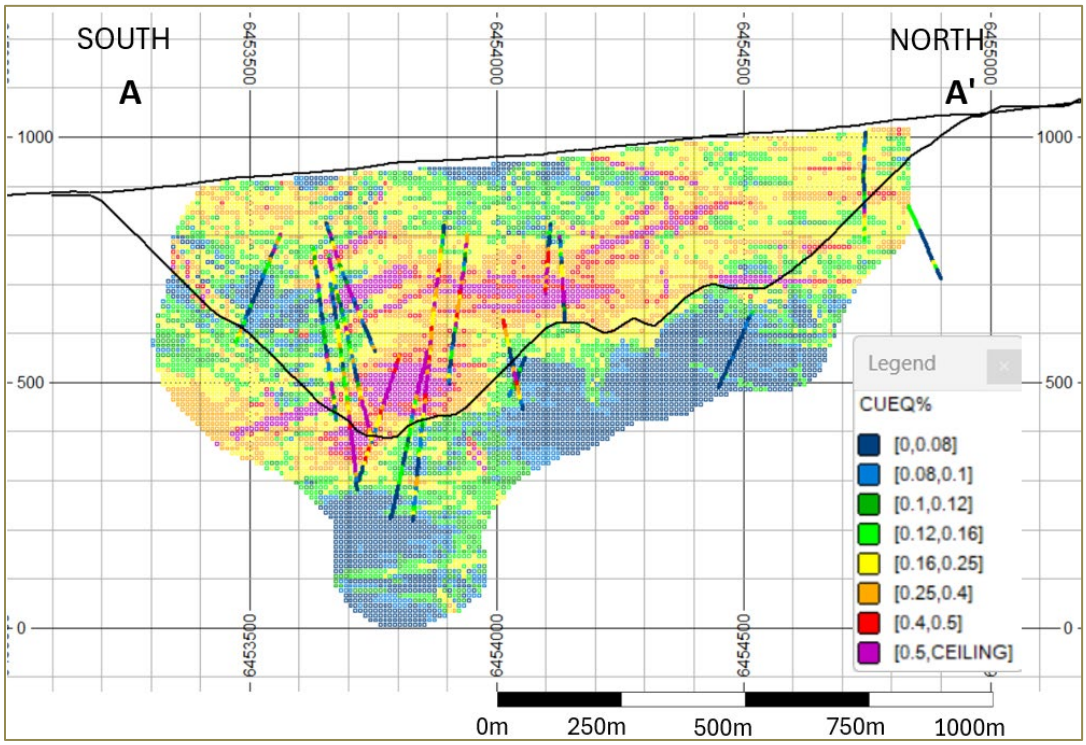
It is important to note that the CuEq formulation used in Table 14-23 differs from that applied in the 2024 Mineral Resource estimate. The two estimates were completed independently by different QPs, and the current method represents the approach adopted for this update rather than a revision of the previous work.

Table 14-23: Comparison of the 2024 Mineral Resource Estimate and this Technical Report's Estimate at a 0.2% CuEq Cut-Off, Constrained by the Respective Pit Shells that Micon (2024) and Mineit (2026) Generated

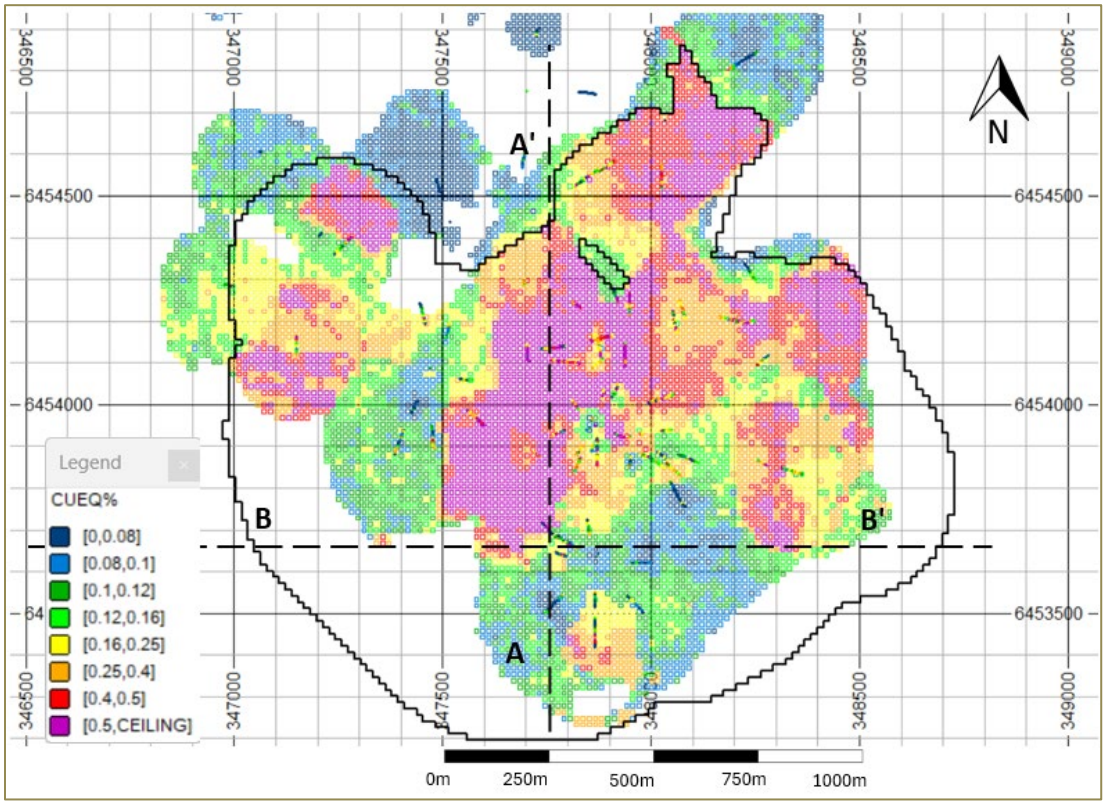
Category	Parameter	2024	2026	% Change
Measured	Tonnage (Mt)	-	272	-
	CuEq Grade (%)	-	0.44	-
	Contained CuEq (MLb)	-	2,612	-
Indicated	Tonnage (Mt)	150	337	125
	CuEq Grade (%)	0.41	0.43	-
	Contained CuEq (MLb)	1,353	3,214	138
M + I	Tonnage (Mt)	150	609	306
	CuEq Grade (%)	0.41	0.43	-
	Contained CuEq (MLb)	1,353	5,826	331
Inferred	Tonnage (Mt)	477	503	5
	CuEq Grade (%)	0.34	0.41	-
	Contained CuEq (MLb)	3,619	4,566	26



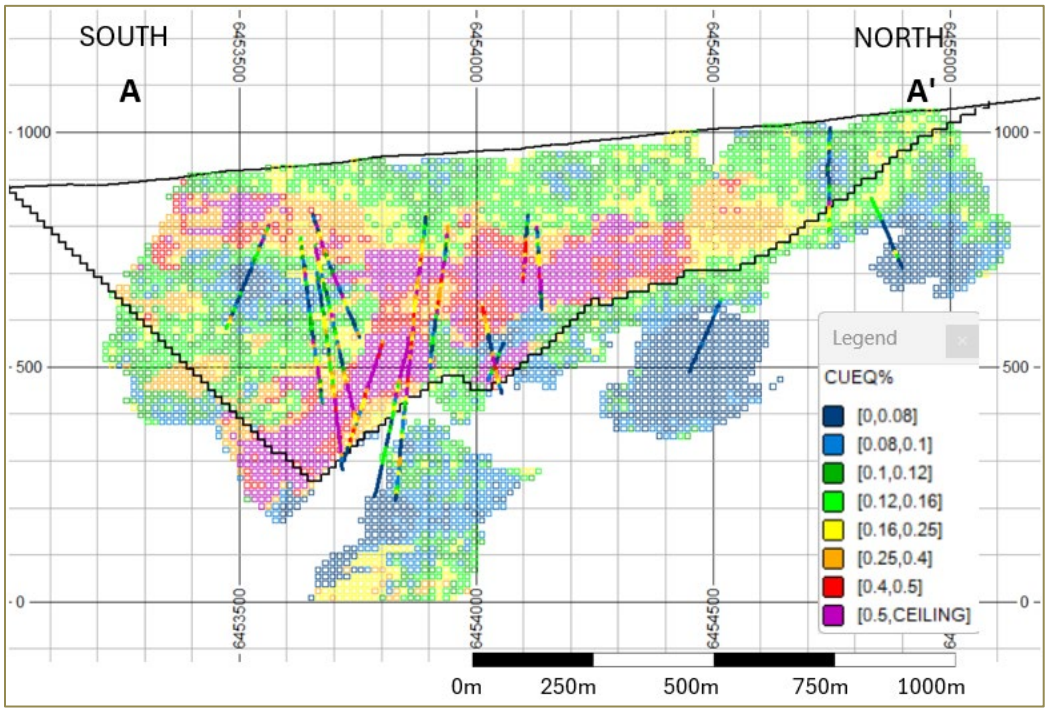
Plan View, Elevation 700 masl, Mineral Resource Estimate 2024



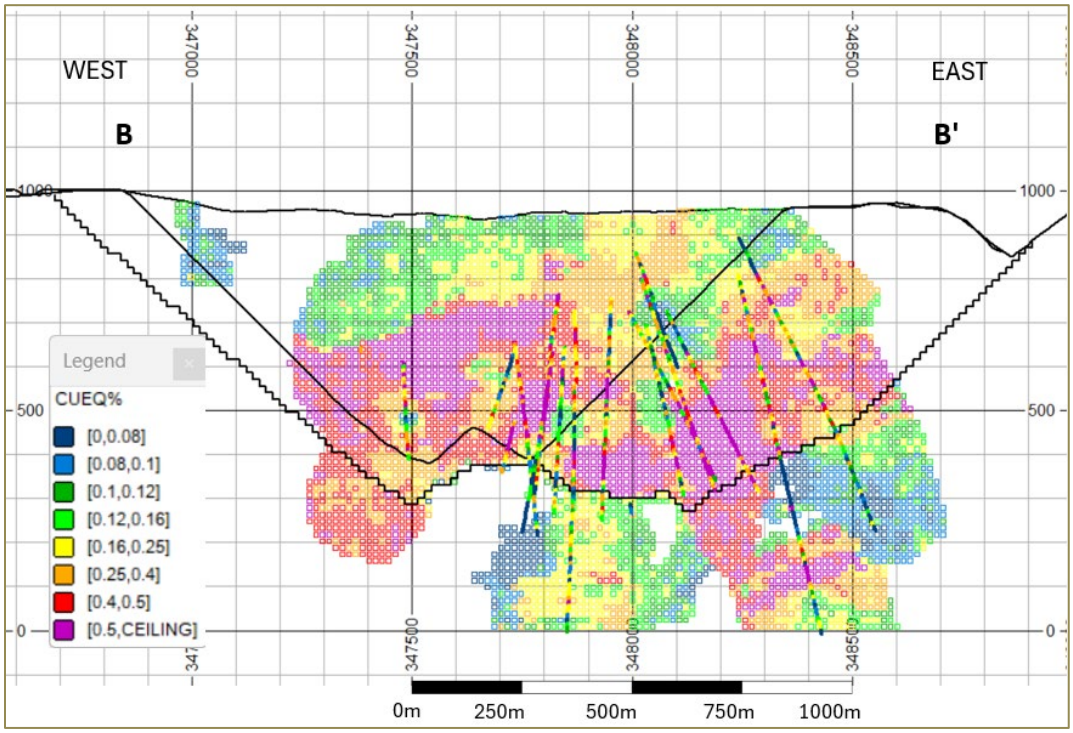
East 347760 Looking West, Mineral Resource Estimate 2024



Plan View, Elevation 700 masl, Mineral Resource Estimate 2026



East 347760 Looking West, Mineral Resource Estimate 2026



Cross-Section at North 6453800 looking North. The zone between the 2024 pit shell (smooth line) and the 2026 pit shell (stepped line 120 kt/d) corresponds to the portion of the deposit contributing to the increase in the 2026 MRE

Figure 14-72: Plan View and Cross-Section Comparison of the 2024 Mineral Resource Estimate and the 2026 Mineral Resource Estimate

14.16.5 Geological Reasonableness

Established Mineralization Model

Extensive drilling, geological mapping, geophysics, and geochemical analysis confirm that the Hat deposit represents a well-developed alkalic porphyry system hosted within a large intrusive complex. The mineralization style, alteration patterns, and metal associations (Cu–Au–Ag–Co ± Sc) are consistent with recognized porphyry systems in British Columbia.

The intrusive-hosted mineralization exhibits:

- Predictable vertical continuity
- Coherent mineralized domains for copper and gold
- Pervasive cobalt and silver envelopes
- A spatially stable scandium distribution.

This geological understanding provides a strong basis for modelling and supports the assumption that mineralization extends in a continuous and mineable geometry.

Genetic Model and Grade Distribution Patterns

The spatial distribution of metals follows a coherent and predictable pattern:

- Copper and gold show strong spatial correlation and occur predominantly in the middle and lower portions of the mineralized system, forming a continuous core. Their co-location reflects the primary magmatic–hydrothermal pulse associated with the intrusive system.
- Silver and Cobalt display a more pervasive and volumetrically extensive distribution, forming broad envelopes that encapsulate the copper–gold core. Their omnipresent character is consistent with late-stage hydrothermal dispersion and sulphide saturation patterns.
- Scandium is present as a background, spatially stable element, with a near-normal distribution and no evidence of structural or lithological control. Sc behaves as a pervasive accessory element throughout the intrusive complex.

This genetic consistency reinforces the geological continuity required for RPEEE and supports the assumption that mineralization extends in a mineable geometry.

14.16.6 Data Quality and Estimation Reliability

Drill Database Integrity

The current Mineral Resource estimate is supported by:

- 97 drill holes used for estimation
- Systematic QA/QC programs
- Validated density measurements
- Consistent geological logging
- A well-defined lithological framework.

The data density is appropriate for the current stage of project development and supports the classification of Measured, Indicated, and Inferred Mineral Resources.

Drill-Hole Survey Coverage and Its Impact on RPEEE

Approximately 36% of drill holes in the current database lack downhole survey measurements. This introduces positional uncertainty at depth; however, the QP considers this uncertainty non-fatal for the following reasons:

- Open-pit mining geometry
- The Hat Project is currently evaluated under an open-pit mining scenario, where extraction begins at surface.
  - Near-surface mineralization during the first years of production is unaffected by survey uncertainty.

  - The lack of survey data affects only the deeper portions of the model, well below the initial pit benches.
- The magnitude of positional uncertainty is low; even at depths of approximately 300 m, the expected deviation of unsurveyed holes remains small relative to the production block size (15 × 15 × 15 m).
- The disseminated character of mineralization further reduces sensitivity to small spatial deviations, unlike vein-hosted systems where precise drill-hole positioning is critical. Even where downhole surveys are absent, small spatial offsets do not materially affect grade continuity or the ability to model the deposit at the scale relevant to open-pit mining.

Probabilistic Modelling Mitigates Survey Uncertainty

The probabilistic domain approach assigns blocks to mineralized volumes only when they exhibit a ≥50% probability of exceeding the natural cut-off grade. This method inherently incorporates spatial uncertainty:

- Blocks with insufficient support or inconsistent spatial correlation are naturally excluded.
- The model quantifies volumes that are well supported and geologically coherent.
- Uncertainty is quantified rather than ignored.

Future Drilling Reduces Uncertainty

All future drilling will include full downhole surveying, progressively reducing positional uncertainty as the Project advances. The QP considers the current level of uncertainty acceptable for the present stage of exploration, and additional drilling will be optimized to improve confidence without over-drilling the deposit.

Given these factors, the survey gap does not compromise the ability to declare Mineral Resources nor the RPEEE conclusion.

Cut-Off Grade Determination

The cut-off grades used for the Hat Project were established through a quantitative process that integrates:

- The volume–variance relationship
- The proportional effect (variance increasing with grade)
- The behaviour of grade variance across the cumulative distribution function.

For gold, copper, and cobalt, a mobile-panel variance analysis was performed using 100 × 100 × 30 m moving windows. This method identifies the inflection point where variance begins to increase more rapidly, marking the transition between background values and the onset of economically meaningful mineralization. These inflection points form the basis for defining natural cut-off grades, ensuring that thresholds are grounded in the statistical behaviour of the deposit rather than arbitrary selection.

The resulting cut-offs—0.07 ppm Au, 0.1 ppm Ag, 900 ppm Cu, and 35 ppm Co—are defensible, reproducible, and aligned with the intrinsic grade distribution of the deposit.

Scandium did not exhibit the same proportional-effect behaviour. For this reason, scandium was treated as background mineralization and estimated across the entire intrusive domain without a probabilistic cut-off constraint.

This approach ensures that the cut-off grades used in the Mineral Resource estimate are consistent with CIM Definition Standards, reflect the geostatistical characteristics of the Hat deposit, and support the Reasonable Prospect for Eventual Economic Extraction.

14.16.7 Probabilistic Domain Modelling

The Hat Project uses a probabilistic indicator-kriging approach to define mineralized domains. Only blocks with a ≥50% probability of exceeding the natural cut-off grade are included in the estimation domain. This approach:

- Quantifies geological uncertainty
- Avoids subjective boundary drawing
- Ensures defensibility of domain geometry
- Aligns with modern geostatistical best practices.

The probabilistic method provides a transparent and reproducible basis for RPEEE.

14.16.8 Volume–Variance Principles and Classification

The classification strategy is grounded in volume–variance relationships, using production-scale volumes to evaluate estimation error. This approach is consistent with the *CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines* (CIM, 2019), which emphasize that confidence levels must reflect the scale of mining and the reliability of estimates at the appropriate production support.

Current Classification Thresholds

At the present stage of exploration, the current classification thresholds are:

- Measured Resources
  - Based on a 2-month production volume
  - 20% estimation error at 80% confidence.
- Indicated Resources
  - Based on a 1-year production volume
  - 20% estimation error at 80% confidence.

These thresholds are appropriate for the current drilling density and project maturity.

Future Requirements for Production-Ready Confidence

To fully satisfy production-level criteria:

- Measured Resources should target
  - 10%–15% error at 90% confidence
  - Evaluated at monthly production volumes.
- Indicated Resources should target
  - 10%–15% error at 90% confidence
  - Evaluated at annual production volumes.

To fully satisfy production-level confidence criteria, Mineral Resources should ultimately target an estimation error of 10%–15% at the 90% confidence level, evaluated at monthly and annual production volumes. Achieving these thresholds will require additional, but not excessive, drilling to improve confidence. The objective is to optimize drilling to reduce uncertainty while maintaining capital efficiency and ensuring that investment aligns with the Project's development stage.

14.16.9 Scandium Potential

Role of Scandium Within the Mineral Resource Model

Scandium is not a primary economic driver of the Hat Project Mineral Resource. It does not influence the geometry of the resource model and is not included in the pit shell optimization, underground stope optimization, or the determination of RPEEE volumes.

Although scandium does not define its own mineralized domain, it was estimated within the existing copper–gold–silver–cobalt resource domains and therefore inherits the same geological constraints and classification framework. The estimation results show good continuity and high confidence, but scandium is considered background mineralization relative to the primary commodities.

Scandium reporting is therefore secondary and is tied to the potential for future extraction from flotation tailings, rather than from primary ore processing.

Geological Conditions Supporting Scandium Potential

Available geological, mineralogical, and geochemical data indicate that scandium is preferentially associated with mafic to intermediate, iron–magnesium silicate–rich domains, typically characterized by amphibole–pyroxene–chlorite–epidote ± carbonate assemblages and elevated iron oxide, magnesium oxide, and silicon dioxide with moderate aluminum oxide and calcium oxide.

These domains are favourable because scandium is commonly enriched in ferromagnesian silicates, and these silicate-hosted scandium phases show predictable leach behaviour under hydrogen fluoride-assisted sulphuric acid conditions. Within the Hat Project, these favourable conditions correspond to specific mapped lithologies, including amphibole–pyroxene–bearing diorite, monzodiorite,

gabbroic phases, and associated mafic to intermediate intrusive and volcanic units. Their propylitic alteration halos, dominated by chlorite–epidote ± carbonate, exhibit the iron–magnesium–silicate mineralogy and bulk-oxide chemistry consistent with elevated scandium and favourable leach response.

These geological characteristics support the concept that a subset of the flotation tailings will be suitable feedstock for scandium extraction.

To support future metallurgical domain modelling, drill-hole data should be compiled on a per-sample basis, including scandium grade, major oxides (Fe_2O_3, MgO, CaO, Al_2O_3, SiO_2, TiO_2), zirconium, lithology, and alteration codes. Simple indices (acid load proxy, scandium efficiency index, impurity index) may be developed to identify zones where scandium recovery is maximized while acid consumption and impurity dissolution are minimized.

Metallurgical Basis for Stand-Alone Scandium Leaching

1. Scandium in flotation tailings
   Metallurgical testwork (Section 13), demonstrates that 95%–98% of scandium reports to the flotation tailings, making the tailings stream the logical feed for scandium recovery.
2. High-acid leach performance
   A dedicated scandium leach circuit has been demonstrated at laboratory-scale testwork and mini-pilot scale. Key outcomes include:
   - Strong sulphuric acid leach (~1,050 kg/t H_2SO_4) with fluoride reagent
   - Operating conditions—~95°C, 20% solids, 250 g/L free acid, 5–8 hours
   - Scandium extraction of 85%–90%.
   - Co-dissolution of iron, aluminum, magnesium, and titanium.

This step is technically effective but reagent-intensive, requiring on-site sulphuric acid production.

3. Impurity removal and solvent extraction
   Selective precipitation removes iron, followed by solvent extraction (SX) using di-(2-ethylhexyl)phosphoric acid, which demonstrated:
   - 94% scandium extraction
   - Effective scrubbing of co-extracted titanium
   - Stable operation in a 63-hour mini-pilot run.

These results confirm that the SX circuit is scalable and capable of producing a purified scandium stream.

4. Final product
   Scandium is precipitated as a hydroxide or carbonate and calcined to scandium oxide, confirming the ability to produce a saleable final product.

Processing Constraints and Throughput Assumptions

Due to the high acid consumption of the scandium leach, the Project contemplates on-site sulphuric acid generation. Under current design assumptions, the acid plant capacity limits scandium processing to approximately 12.5% of the mineralized tailings. This proportion is used to estimate scandium tonnage potentially available for extraction.

Future optimization will require metallurgical domain modelling to identify geological conditions that minimize acid consumption and impurity dissolution.

RPEEE Conclusion for Scandium

Based on the geological evidence, metallurgical testwork, and processing constraints, scandium demonstrates reasonable prospects for eventual economic extraction as a secondary by-product recovered from a dedicated tailings leach circuit. Key supporting factors include:

- Favourable geological domains hosting scandium in silicate minerals amenable to hydrogen fluoride-assisted sulphuric leaching
- Demonstrated metallurgical performance, including high scandium extraction and successful production of scandium oxide
- Planned on-site sulphuric acid production to support the reagent-intensive leach
- A throughput assumption of 12.5% of mineralized tailings, consistent with current acid plant design constraints

Scandium is therefore appropriately reported as a secondary, tailings-based opportunity within the RPEEE framework, without influencing the primary Mineral Resource geometry or economic drivers.

14.16.10 Mining and Processing Considerations

Mining Method

The geometry, depth, and grade distribution of the Hat deposit support the assumption of a large-scale open pit mining scenario. Preliminary pit optimization, completed using reasonable long-term metal prices and realistic operating costs, demonstrates that a significant portion of the mineralized volume falls within an optimized resource shell. This provides a practical mining framework consistent with the RPEEE requirement.

Processing and Metallurgy

Metallurgical testwork completed to date indicates that:

- Copper, gold, silver, and cobalt are recoverable using conventional flotation
- Scandium shows potential for recovery as a by-product
- Mineralogy does not present prohibitive processing challenges.

These results support the assumption that mineralization is amenable to standard processing routes and that no fatal metallurgical flaws are present at this stage.

14.16.11 Economic and Jurisdictional Factors

Commodity Prices

The RPEEE assessment uses long-term consensus metal prices consistent with industry practice. These price assumptions are reasonable for evaluating the eventual economic extraction of the Hat deposit.

British Columbia as a Mining Jurisdiction

The Hat Project benefits from its location in British Columbia, a globally recognized mining jurisdiction characterized by:

- A stable regulatory environment
- Established mining infrastructure
- Proximity to transportation corridors
- A skilled workforce
- Strong precedent for permitting large open-pit operations.

These jurisdictional advantages materially strengthen the RPEEE justification.

14.16.12 Conclusion: RPEEE Justification

The QP concludes that the Hat Project's Mineral Resources have reasonable prospects for eventual economic extraction based on:

- The geological continuity of the alkalic porphyry system
- The quality and density of the drill database
- The probabilistic domain modelling
- The volume–variance classification framework
- The favourable mining and metallurgical characteristics
- The economic assumptions used in pit optimization
- The strong mining jurisdiction of British Columbia.

The current classification reflects both the stage of exploration and the quantified uncertainty. Future drilling will further refine confidence levels and support the eventual conversion of Mineral Resources to Mineral Reserves.

15 MINERAL RESERVE ESTIMATE

There are no current mineral reserve estimates for the Hat Project.

16 MINING METHODS

This PEA for the Hat Project evaluates developing a large-scale open pit mining operation. The mine plan envisions a conventional truck-and-shovel operation moving nominal mill feed of 120,000 t/d (approximately 42 Mt/a).

The mining operation is scheduled to support a 25-year mine life. The mine plan emphasizes a phased development strategy, using a higher-grade starter pit to maximize early cash flow and minimize the payback period.

Cautionary Statement: The mine plan and economic model presented in this section are preliminary in nature and include Inferred Mineral Resources that are considered too speculative geologically to have 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.

16.1 Pit Geotechnical Data Assessment

No site-specific geotechnical drilling or rock mechanics laboratory testing has been completed for the Hat Project to date. For this PEA, pit-slope design criteria were estimated based on:

- Geotechnical logging (rock quality designation [RQD] and core recovery) from the exploration campaigns—RQD data are available in 83 out of 101 drill holes.
- Benchmarking against similar alkalic copper–gold porphyry deposits in British Columbia.

Doubleview provided a preliminary interpretation of the principal fault systems in the Hat deposit area; however, based on the currently available information, it is not possible to reliably assess or verify the extent to which these structures may affect open pit slope stability at this stage of the study.

Because of the extensive overburden, vegetation cover, and the absence of detailed surface geological mapping, the fault systems have been interpreted primarily from satellite imagery, and their orientation, continuity, kinematics, and mechanical significance have not yet been adequately constrained. As a result, the potential influence of these structures on pit slope design, inter-ramp stability, and the location of sector-scale slope domains remains uncertain.

Evidence consistent with faulting has been identified in drill core in two principal forms:

- Crushed zones containing abundant clay minerals, in some cases associated with calcite veining (Figure 16-1).
- Crushed and brecciated rocks with little clay content (Figure 16-2).



Figure 16-1: Fractured Zone with Abundant Clay Minerals (BH092 147.2–156 m)

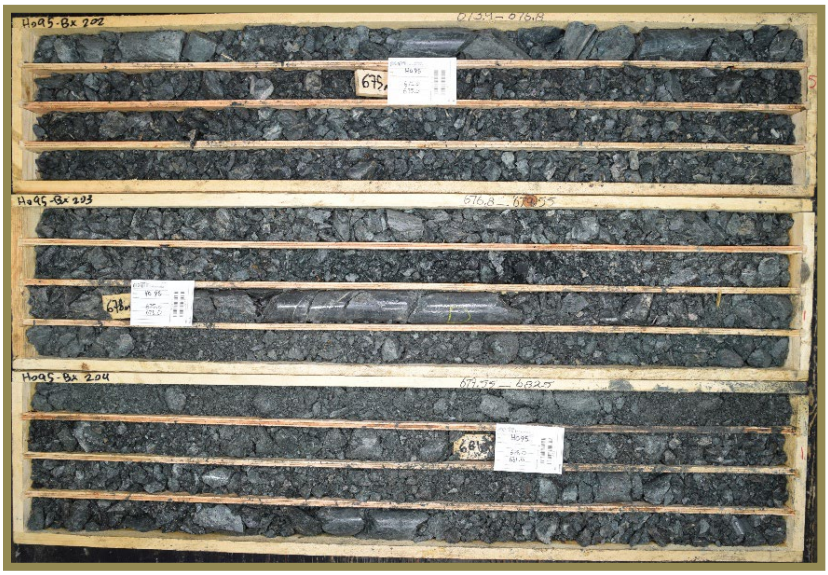


Figure 16-2: Fractured Zone with Fault Breccias (BH095 673.9–682.5 m)

These crushed rocks and argillitic zones in the drill holes vary from 20 m to more than 30 m thick, indicating that significant zones of structural weakness may be present within the pit area.

However, the available data are insufficient to define their geometry and engineering significance with confidence. Further structural interpretation, geological mapping, oriented core analysis, and targeted geotechnical investigation will be required in subsequent study phases to properly assess the impact of the fault systems on pit slope performance and to support refinement of slope design criteria.

16.1.1 Rock Quality Designation Data Analysis

RQD values were compiled for all drill holes with valid core-recovery data and matched to corresponding lithological intervals through weighted overlap methods. Both unweighted and length-weighted statistics were generated. Weighted metrics were used as the primary basis for geotechnical domain characterization.

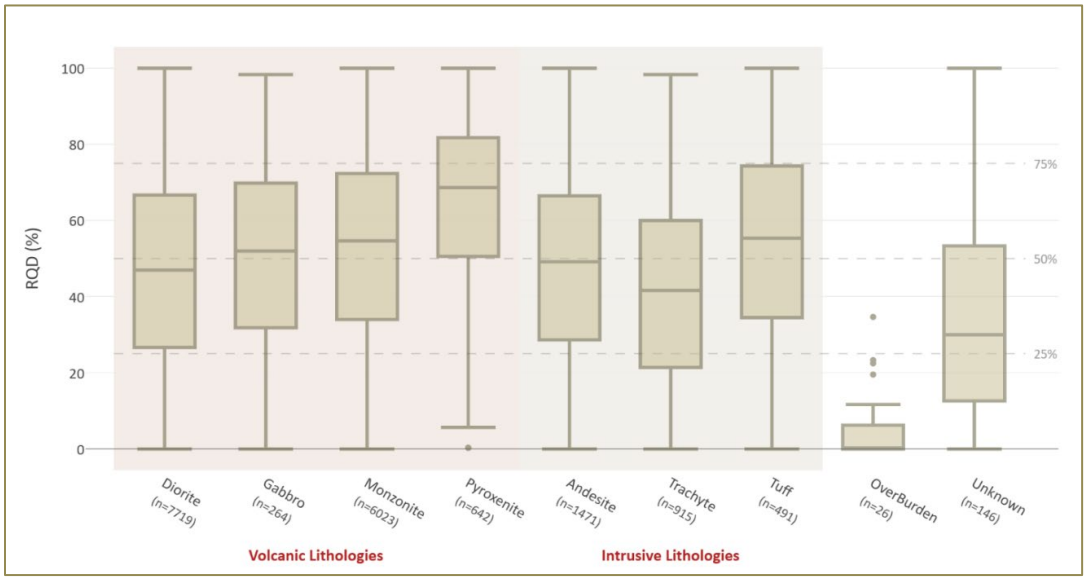


Figure 16-3: RQD Distribution Box Plots by Lithology Domains

Table 16-1: RQD Statistical Analysis by Lithology Domains

	Domain	Number Intervals	Total (m)	Avg. RQD (unw)	Avg. RQD (w)	P_{10} (w)	P_{25} (w)	Median (w)	P_{75} (w)	P_{90} (w)
Volcanic Lithology	Diorite	7,719	18,724	47	47	13	27	48	67	81
	Gabbro	264	630	50	50	3	30	52	73	84
	Monzonite	6,023	14,643	53	53	17	34	55	73	84
	Pyroxenite	642	1,661	65	66	35	52	70	83	91
Intrusive Lithology	Andesite	1,471	3,134	48	48	13	30	49	67	79
	Trachyte	915	2,145	42	42	10	21	42	60	73
	Tuff	491	1,116	53	53	15	35	55	75	86
Other	Fault Zone	467	829	24	18	0	0	12	27	48
	Overburden	26	111	6	2	0	0	0	0	6
	Unknown	146	323	34	30	0	10	26	50	65
	Unclassified	2	2	40	37	35	35	35	35	44

Notes: "Avg RQD (w)" = length-weighted mean RQD.
"Avg RQD (unw)" = unweighted mean RQD.
Percentiles (P_{10}, P_{25}, P_{75} and P_{90}) shown are length-weighted (w).
"Unclassified" and "Unknown" contain intervals lacking lithology codes; total influence negligible.
RQD measurements are available for approximately 87% of the total drilled metres (43,316 m out of 49,885 m).

The volcanic intrusive suite (diorite, gabbro, monzonite, and pyroxenite) shows weighted median RQD values ranging from approximately 48% to 70%. Monzonite and pyroxenite exhibit the highest RQD within this group, with weighted medians of 55% and 70%, respectively, and upper percentile values (P_{75}–P_{90}) commonly exceeding 80%. Diorite and gabbro domains show similar but slightly lower quality, with weighted medians of approximately 48% and 52%. Given their comparable RQD distributions and geological affinity, these units are considered suitable for grouping within a single geotechnical domain for preliminary pit slope design.

Intrusive units—andesite and trachyte—show weighted median RQD values near 49% and 42%, respectively. These units form a distinct domain based on both geological considerations and RQD characteristics. Despite its comparatively high RQD (weighted median ~55%), tuff is maintained as a separate domain due to its known susceptibility to degradation and lower expected intact-rock strength based on analogous deposits in the region.

Faulted intervals represent the lowest-quality rock mass, with a weighted median RQD of 11.8% and a weighted P_{75} of only 27%, consistent with highly fractured and sheared conditions. As expected, overburden and non-rock materials exhibit near-zero RQD values.

These results form the basis for preliminary geotechnical domain definition and will be integrated with subsequent Modified Rock Mass Rating (MRMR) calculations. The RQD-derived domains support the development of representative inter-ramp angles for PEA-level slope design and will be validated against structural and geomechanical data during the next phases of the Project.

16.1.2 Geotechnical Rock Mass Classification and MRMR-Based Slope Recommendations

Given the early stage of the study and limited structural data, the approach used emphasizes the use of measured drill-core RQD, lithology-based UCS ranges derived from published literature, and conservative empirical adjustments. This method is consistent with accepted practice for conceptual studies and aligns with Read and Stacey's (2009) guidance, the Large Open Pit (LOP) Project, and empirical rock mass classification (Laubscher, 1990) and slope stability methods (Haines et al., 1991).

Data Availability and Limitations

Geotechnical drill holes contain RQD measurements at 3-m intervals, providing adequate information to establish preliminary indicators of rock mass discontinuity trends across the modelled area. Uniaxial compressive strength (UCS) data are not available from site-specific laboratory testing; therefore, UCS estimates were derived from peer-reviewed literature and analogous intrusive and volcanic rock types in northern British Columbia.

Structural data (bedding, foliation, joint sets, shear zones) are limited and insufficient to allow a rigorous kinematic analysis in this PEA. This limitation is explicitly considered in the conservative selection of classification parameters and slope design factors. Additional field characterization, oriented core, and joint set mapping are required to refine the design for future study stages.

Rock Mass Classification Method

Rock mass quality was evaluated using the Laubscher (1990) rock mass rating (RMR_L) system and subsequently adjusted to obtain the MRMR. The MRMR framework is widely used in preliminary slope design because it incorporates factors associated with mining: including weathering, blasting, structure orientation, and excavation-induced stress.

Given the data constraints, the following assumptions were applied to all lithological domains:

- Average intact rock strength (IRS) as % of strong rock IRS: 60%
- Joint spacing rating (J_s,n): 20 / 25
- Joint condition rating (J_c): 30 / 40
- Site-wide conservative MRMR factors:
  - Weathering= 0.78
  - Orientation= 1
  - Induced Stresses = 1
  - Blasting = 0.8

These assumptions reflect a conservative interpretation consistent with PEA-level confidence.

Determination of Base Rock Mass Rating

The base RMR (RMR_L) was calculated before applying orientation adjustments, as required for MRMR derivation. RQD values and UCS ranges were domain-averaged, and ratings were assigned using Laubscher's (1990) criteria.

RMR_L values range from 61 to 74, corresponding to a good rock mass quality at a conceptual level as shown in Table 16-2.

Table 16-2: RMR_L by Lithology Domain

Domain	Average RQD (%)	RQD Rating	IRS (MPa)	IRS Rating	Joint Spacing Rating[1]	Joint Condition Rating[1]	RMR_L
Diorite	47	8	126	14	20	30	72
Gabbro	50	10	147	14	20	30	74
Monzonite	53	8	124	12	20	30	70
Pyroxenite	66	10	97	10	20	30	70
Andesite	48	8	126	14	20	30	72
Trachyte	42	7	90	10	20	30	67
Tuff	53	8	18	3	20	30	61

Note: [1] No information is available about the Joint conditions or spacing, current assumption need to be review in the following studies.

MRMR Calculation

The MRMR values were computed using Laubscher's (1990) reduction factors, described above. These values indicate that after incorporating mining-related degradation, all domains fall within an MRMR range of 38–46.

Table 16-3: MRMR by Lithology Domain

Domain	RMR_L	MRMR
Diorite	72	45
Gabbro	74	46
Monzonite	70	44
Pyroxenite	70	44
Andesite	72	45
Trachyte	67	42
Tuff	61	38

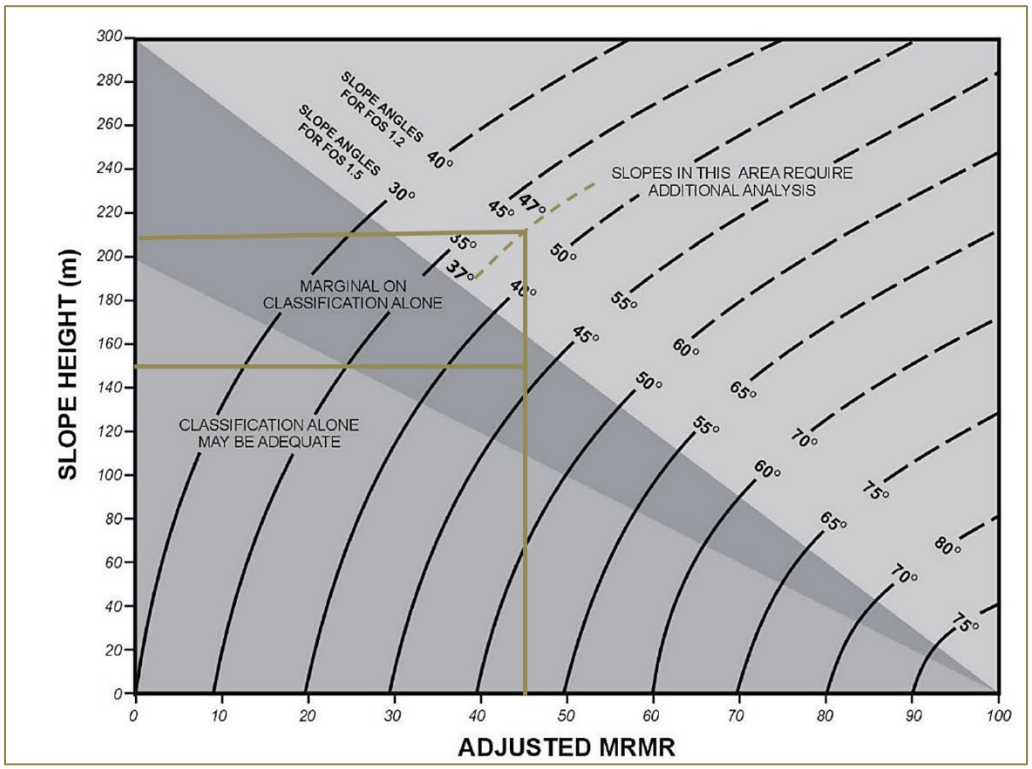
Preliminary Slope-Angle Estimation

Indicative bench stack angles were assessed using the Haines and Terbrugge (1991) empirical relationships, which correlate MRMR, slope height, and factor of safety (FOS) approximately 1.2–1.5 These empirical charts (Figure 16-4) are widely referenced in preliminary open pit design and are considered appropriate for PEA-level studies when supported by conservative assumptions and sensitivity analysis.

For MRMR values in the range of 38–46, and for typical bench stack heights of 150–210 m, the applicable Haines contours correspond to:

- Factor of Safety: 1.35
- Inter-ramp angle: approximately 42°–47°
- Classification indicates that additional kinematic and numerical analyses are required for later study stages.

The indicative angles are not intended as final design parameters; rather, they establish a defensible starting point for evaluating potential pit geometries and planning further data collection.



Source: Haines and Terbrugge (1991).

Figure 16-4: Empirical Slope Design Chart

16.2 Pit Optimization Parameters

The mine design is based on the 2026 Mineral Resource estimate block model. The pit optimization and design process used the following economic and operational parameters (Table 16-4).

Table 16-4: Pit Optimization Parameters

Parameter	Unit	Value
Mining Cost (Mill Feed and Waste)	$/t mined	3.014 (starting)
Incremental Haulage Cost	$/t/bench	0.015
Processing Cost	$/t milled	7.934
G&A Cost	$/t milled	2.94
Throughput Rate	t/d	120,000
Working Day	d/a	350
Copper Price	US$/lb	4.8
Gold Price	US$/oz	3,200
Cobalt Price	US$/lb	20
Silver Price	US$/oz	46
Copper Recovery	%	85

Parameter	Unit	Value
Gold Recovery	%	89
Cobalt Recovery	%	78
Silver Recovery	%	68
Refining Cost—Au	US$/oz	5
Refining Cost—Ag	US$/oz	0.35
Refining Cost—Cu	US$/lb	0.1
Refining Cost—Co	US$/lb	0.12
Transport Cost—Cu concentrate	US$/lb	0.2
Treatment Cost—Cu concentrate	US$/lb	0.2
Exchange Rate	$/US$	1.37
Payable Au	%	92.2
Payable Ag	%	56.7
Payable Cu	%	94.6
Payable Co	%	90.5
Royalty	% of NSR	2
Discount Rate	%	5
Dilution	%	5
Mine Recovery	%	95
Overall Slope Angle	°	42

16.2.1 Mineral Resource Model

The mineral resource model for the Hat Project is described in Section 14. For mine planning, the model was regularized to 15 × 15 × 15 m block size (Table 16-5) and includes estimated grades for silver, gold, copper, cobalt, and scandium, density, geological units, individual classification fields for each variable, and a composite copper-equivalent classification field. The model summary shown in this section differs from that presented in Section 14 because the block model was extended in the X and Y directions for pit optimization, solely to prevent the optimization from being artificially constrained by the original model limits. This modification was made for computational and mine planning purposes only and does not represent a change to the underlying MRE.

Table 16-5: Summary of Hat Resource Block Model

Coordinates	X	Y	Z
Model Origin (UTM Zone 9 North)	346255	6452700	0
Block Size (m)	15	15	15
Number of Blocks	205	200	75
Model Extents (m)	2,040	2,250	1,125
Rotation	0°	0°	0°

The Hat Mineral Resources have been classified as Measured, Indicated, and Inferred. The bulk density was estimated for each block (Section 14.12.4), with an average of 2.95 t/m^3 for waste and mineralization.

Figure 16-5 illustrates the spatial distribution of drill holes across the central area of the Hat Project (Section 10).

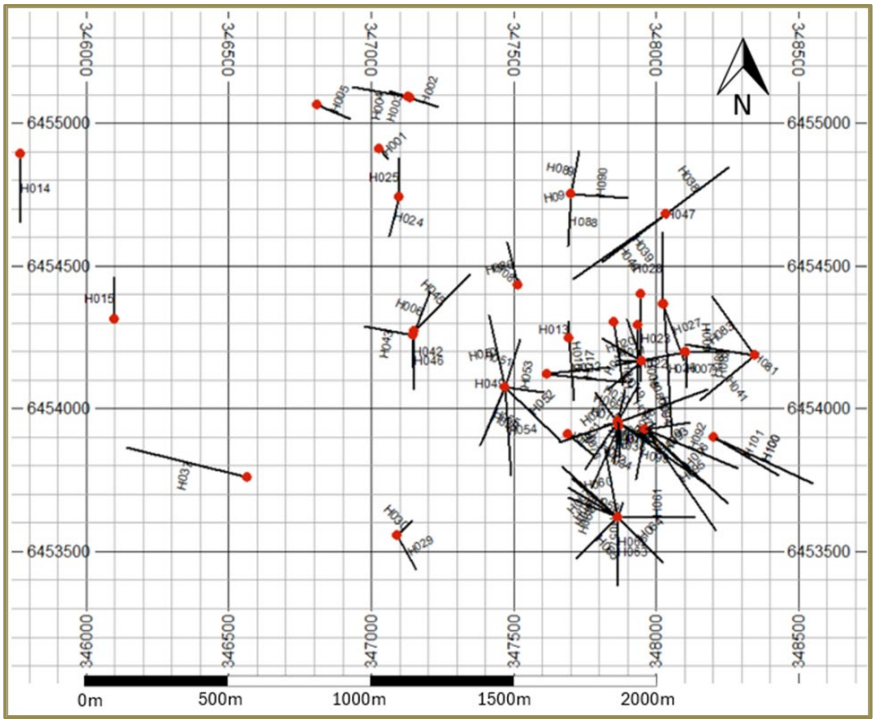


Figure 16-5: Plan View Showing Drill-Hole Locations Within the Hat Project (UTM Zone 9N, NAD 83)

16.2.2 Topography

The Hat Project area is characterized by rugged mountainous topography with strong local relief, steep valley walls, and incised drainage features. Elevations across the claim block range from approximately 650 m to over 1,600 m, with the lowest elevations occurring in the southern portion of the property near Hatchau Lake and Kennicott Lake. The central part of the Project area includes localized bench-like and moderately sloping ground, while the eastern and western portions are marked by steeper slopes and narrow drainages. This topographic setting influences pit layout, waste and low-grade stockpile placement, haul road design, and the location of site infrastructure.

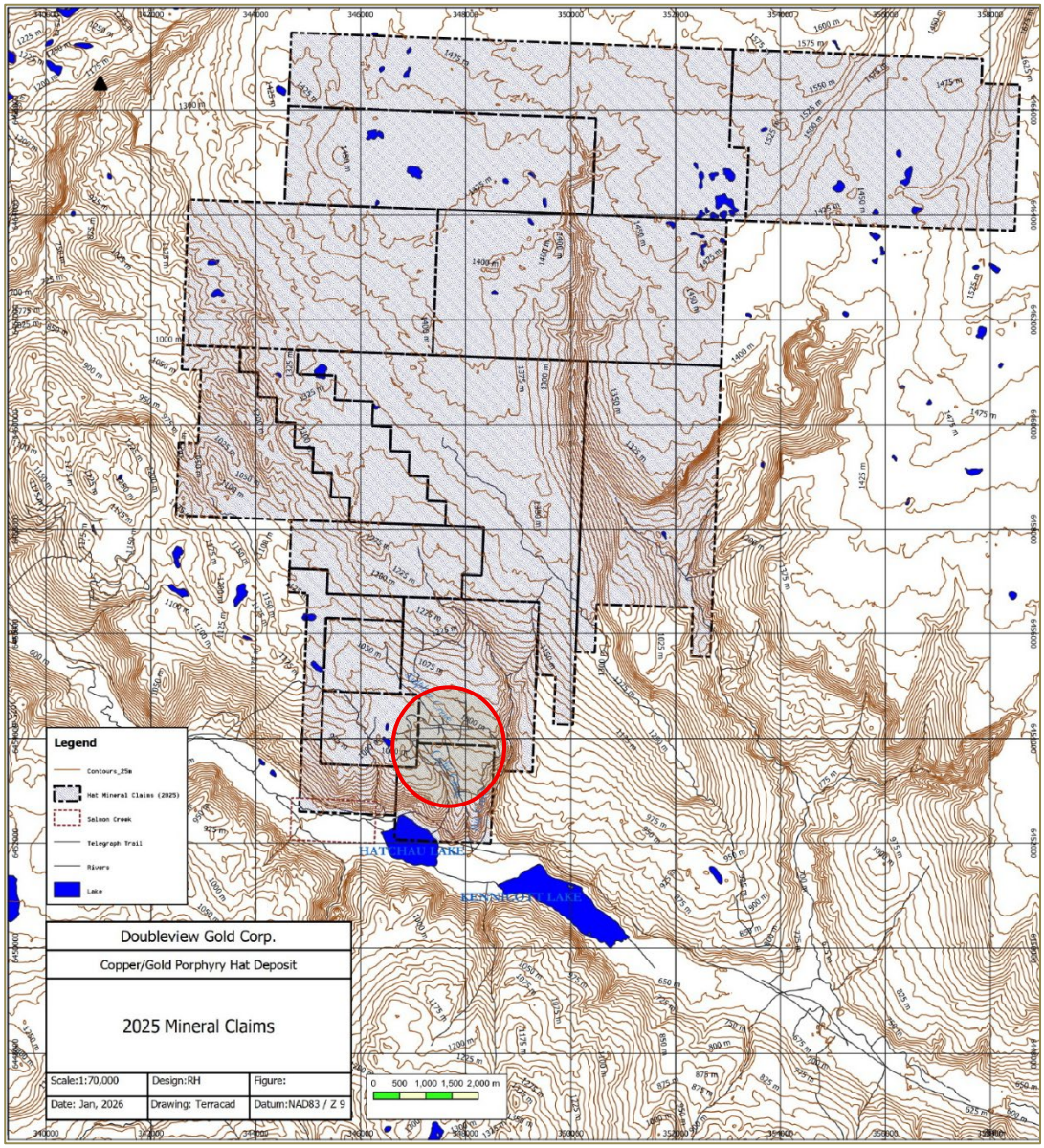


Figure 16-6: Topographic Overview of the Hat Project Area, Showing the Pit Location Highlighted by the Red Circle

16.2.3 Pit Slope

Refer to Section 16.1.

16.2.4 Processing Method and Recoveries

Based on the currently available metallurgical test data, the process plant is designed for a nominal throughput of 42 Mt/a (120,000 t/d) of copper–gold porphyry ore sourced from open pit mining. The flowsheet presently reflects a conventional concentrator configuration comprising crushing, grinding,

flotation, concentrate handling, and tailings management. While ongoing and future metallurgical test programs (Section 17) may refine specific design parameters and product streams, the overall process configuration is not expected to change materially.

Flotation is anticipated to produce two concentrate streams. The primary concentrate will be a marketable high-grade copper concentrate (approximately 25% Cu) containing payable gold. A secondary pyrite-rich concentrate, enriched in cobalt, is expected to undergo roasting to generate sulphuric acid and produce a calcine suitable for cobalt recovery, with additional gold and silver recovery via leaching. The sulphuric acid generated from roasting is intended for use in treating flotation tailings to enable scandium recovery.

The processing operating costs presented in Table 16-4 exclude costs associated with the downstream treatment of the secondary flotation concentrate and the proposed scandium recovery circuit. These costs have been excluded to allow pit optimization to be based solely on the conventional copper–gold flotation circuit (Cu–Au represent ~82%–94% of net revenue). This approach ensures that the open pit design demonstrates economic viability independent of the additional capital and operating costs related to the proposed cobalt, precious metals, and scandium recovery processes, which are considered optional and subject to further technical validation.

16.2.5 Cut-Off Grades

The pit optimization was conducted using a cut-off grade of 0.30% CuEq, reflecting the current level of metallurgical confidence and the limited recovery data available for lower-grade material, particularly below 0.20% CuEq. This threshold was selected to reduce uncertainty associated with processing performance while establishing a robust economic basis for mine design. To enhance early Project cash flow and defer processing of marginal material until later in the mine life, a stockpiling strategy is planned for ore in the 0.20%–0.25% CuEq and 0.25%–0.30% CuEq ranges. These materials are intended for treatment during the later years of operation.

16.3 Pit Optimization

Pit optimization was completed using Whittle 2022.3, applying the technical and economic parameters described in the preceding sections. The optimization generated a series of nested pit shells representing incremental variations in Project revenue assumptions. These shells were developed using revenue factors ranging from 0.2 to 1.0, at 0.01 increments, where the revenue factor is a scalar applied to the base-case metal prices summarized in Table 16-4.

16.3.1 Nested Pit results

The pit optimization results are presented in the nested pit shell analysis generated using Whittle (Figure 16-7), showing the variation in net present value (NPV) as a function of revenue factor (RF). The accompanying plot summarizes the economic response across the base case, worst case, and specified case scenarios, illustrating the relative sensitivity of Project value to pit-limit selection and sequencing

assumptions. This comparison provides the basis for identifying the design pit shell consistent with the Project's economic objectives and planning strategy.

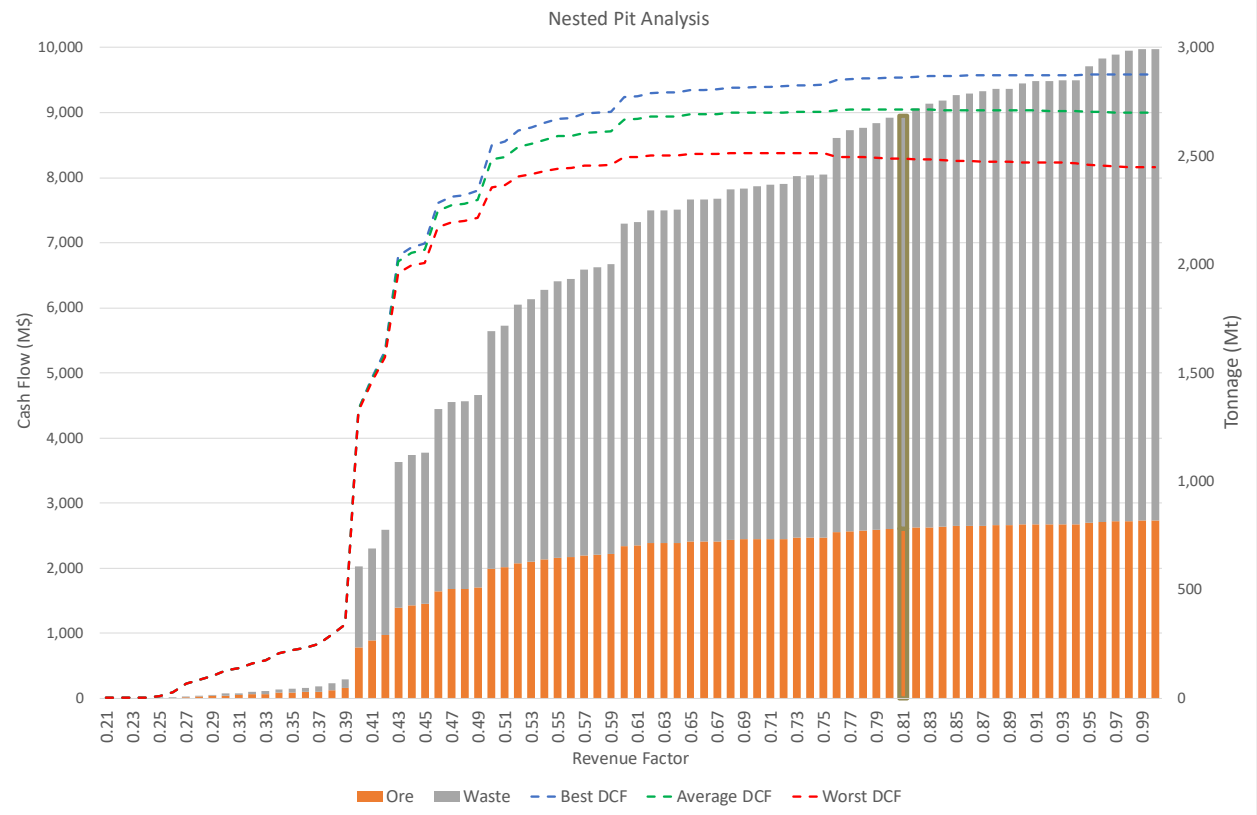


Figure 16-7: Nested Pit Results

16.3.2 Optimal Pit Selection

Pit optimization economics were evaluated using the metal price assumptions from the resource model and the cost parameters outlined on this section. NPV was calculated for each nested pit shell defined by its revenue factor (RF).

The RF 0.81 shell generated the highest average NPV, no significant design and operational challenges are expected for the selected shell. The optimal pit shell selected (Figure 16-8) serves as the basis for subsequent pit design and scheduling and does not represent a final engineered pit.

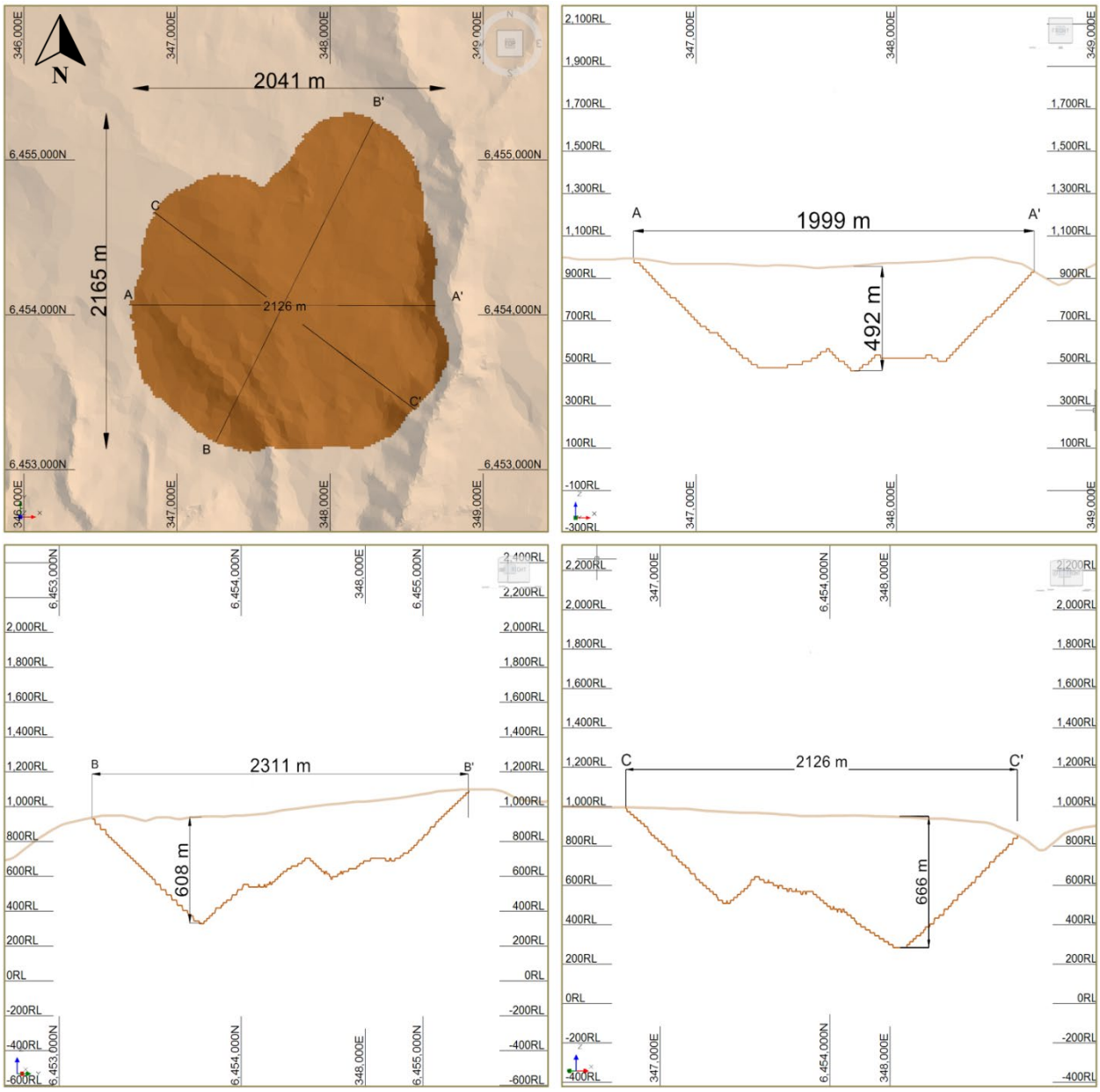


Figure 16-8: Dimension of the Selected Pit Shell for Pit Design at RF 0.81

The pit shell initially used to inform the MRE was revised during mine design to account for an additional topographic restriction along the eastern pit wall and an updated revenue factor used for ultimate pit selection. The topographic constraint was introduced to prevent encroachment into the ravine located east of the deposit. As a result, the ultimate pit selected for mine design differs from the shell referenced in the MRE and is locally more constrained on the eastern side of the deposit. Figure 16-9 and Figure 16-10 present plan views and representative cross sections comparing the two pit shells and illustrating the impact of these changes on the final pit configuration. The combined effect of the east ravine constraint

and revised revenue factor is to limit pit expansion and reduce the amount of mineralization captured within the mine design pit relative to the less constrained shell.

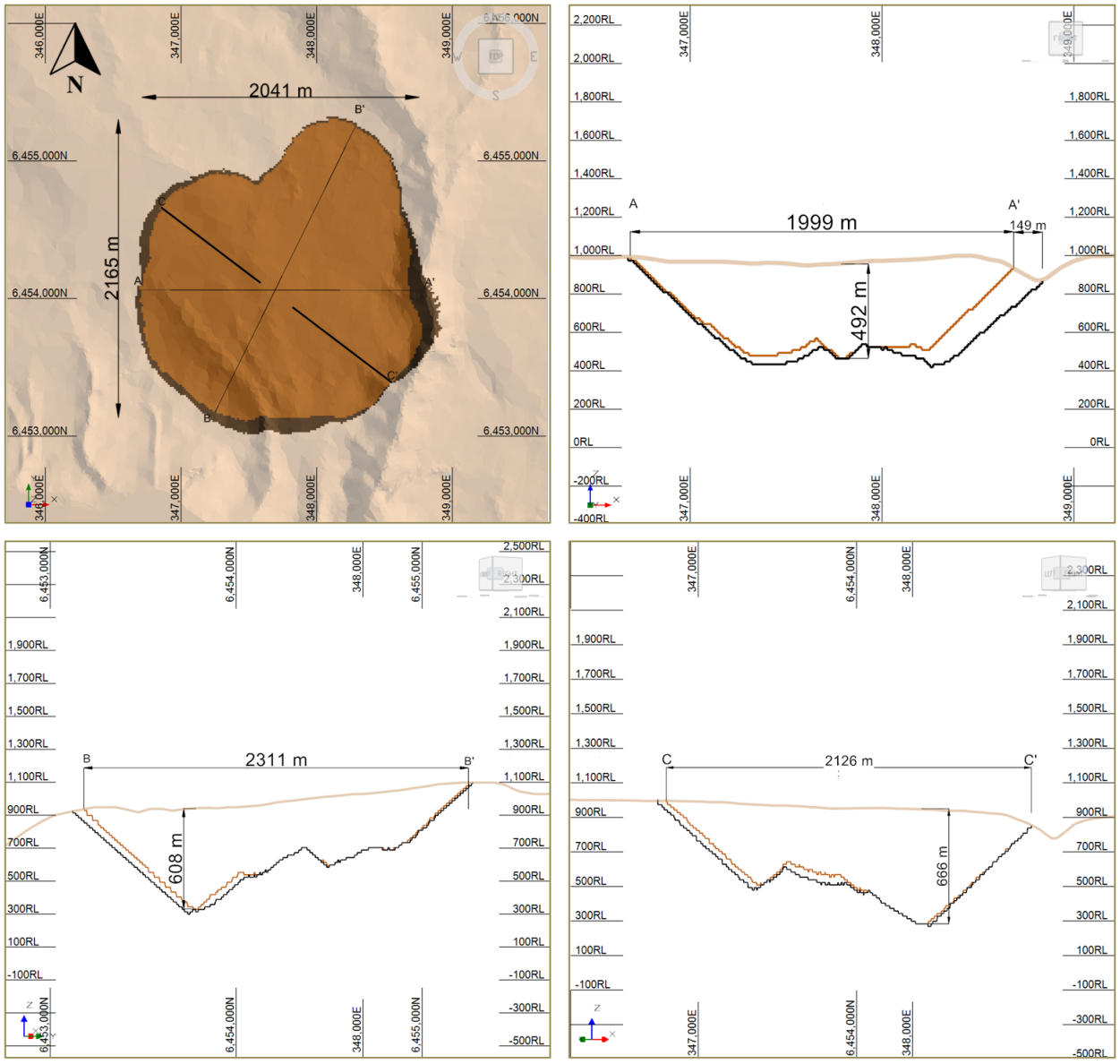


Figure 16-9: Comparison of Ultimate Pit Shells With and Without the East Ravine Constraint, Including Representative Cross Sections Showing the Impact on Pit Geometry

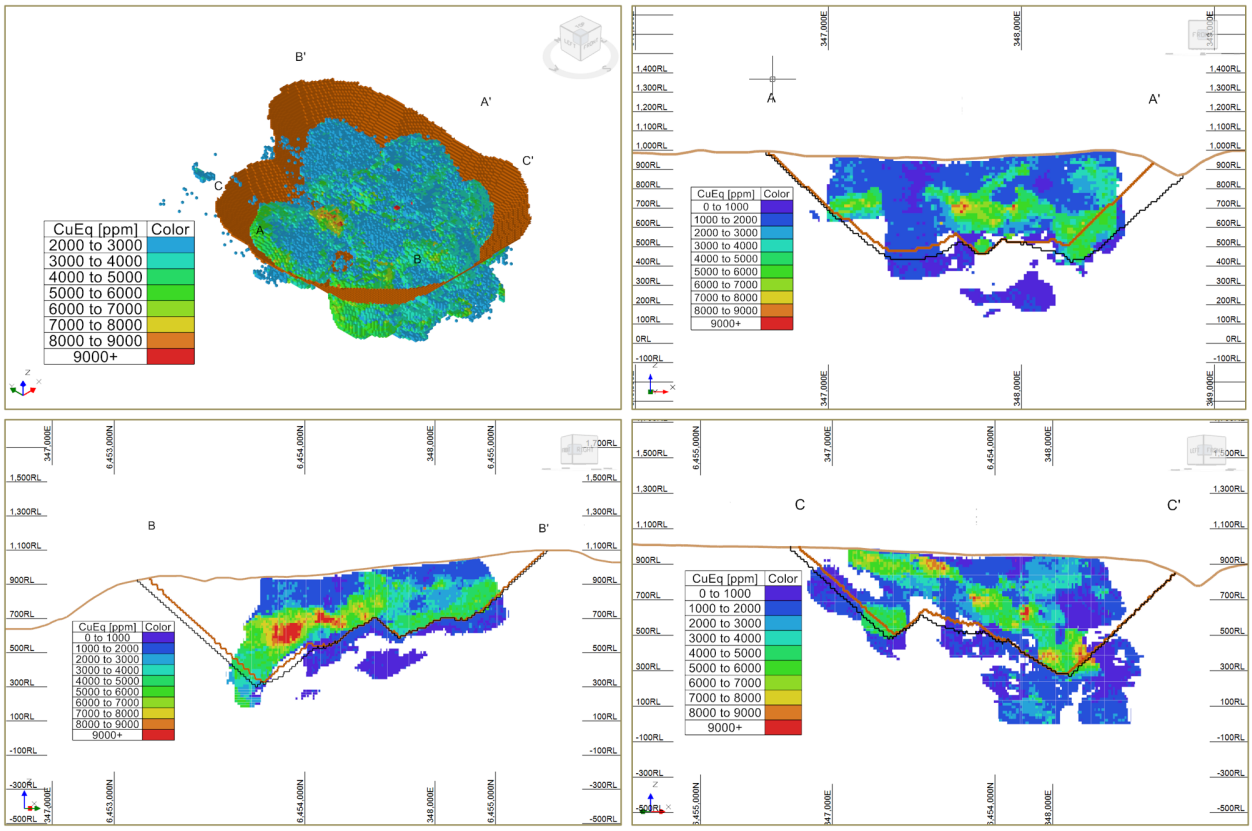


Figure 16-10: Comparison of Ultimate Pit Shells with and Without the East Ravine Constraint, Include Copper Equivalent Grades on the Block Model

Table 16-6 summarizes the material contained within the selected optimal pit shell (RF 0.81), classified according to the cut-off strategy described previously. The table distinguishes between direct mill feed, low-grade stockpile material scheduled for deferred processing, and waste rock. Table 16-7 presents the breakdown by Mineral Resource category. This classification is important in assessing the relative contribution of Measured, Indicated, and Inferred material to the mine plan, and highlights the need for additional drilling to support future resource upgrading in subsequent study stages.

Table 16-6: Summary of Material Inventory Within the Selected Pit Shell

Material Classification	Tonnage (Mt)	Avg CuEq (%)	Destination
ROM Ore (>0.30% CuEQ)	783	0.49	Direct to mill
Low-Grade Stockpile (0.25%–0.30% CuEQ)	128	0.28	End-of-Pit life processing
Low-Grade Stockpile (0.20%–0.25% CuEQ)	113	0.22	End-of-Pit life processing
Total Mill Feed	1,024	0.44	
Waste Rock	1,644	–	Waste storage facility

Table 16-7: Material Inventory Breakdown by Resource Category

	Resources Classification	Tonnage (Mt)	Average Grade				
			CuEq (%)	Cu (%)	Au (g/t)	Co (g/t)	Ag (g/t)
ROM Ore (>0.30% CuEq)	Measured	217	0.48	0.23	0.19	78.81	0.39
	Indicated	225	0.51	0.24	0.21	80.27	0.43
	Total M+I	**443**	**0.49**	**0.24**	**0.20**	**79.56**	**0.41**
	Inferred	340	0.49	0.20	0.22	78.98	0.39
Low-Grade Stockpile 1 (0.25%–0.30% CuEq)	Measured	13	0.27	0.10	0.13	69.34	0.25
	Indicated	55	0.28	0.10	0.13	73.63	0.35
	Total M+I	**68**	**0.28**	**0.10**	**0.13**	**72.81**	**0.33**
	Inferred	61	0.28	0.11	0.12	72.86	0.38
Low-Grade Stockpile 2 (0.20%–0.25% CuEq)	Measured	14	0.23	0.07	0.11	67.04	0.20
	Indicated	27	0.23	0.08	0.11	67.29	0.29
	Total M+I	**41**	**0.23**	**0.08**	**0.11**	**67.20**	**0.26**
	Inferred	72	0.22	0.09	0.08	75.65	0.36
Total Mill Feed	Measured	244	0.45	0.22	0.18	77.66	0.37
	Indicated	308	0.45	0.20	0.19	77.93	0.40
	Total M+I	**551**	**0.45**	**0.21**	**0.18**	**77.81**	**0.39**
	Inferred	472	0.42	0.18	0.19	77.69	0.39

The selected optimal pit shell contains approximately 1,024 Mt of potential mill feed and 1,644 Mt of waste rock, corresponding to an overall strip ratio of approximately 1.6:1 on a mass basis. Of the total mill feed, 783 Mt is classified as direct mill feed above the primary cut-off of 0.30% CuEq, while 242 Mt is assigned to low-grade stockpiles for deferred processing. The grade-based segregation of material reflects the mine design and production strategy described in Section 16.5, whereby material above the primary mill cut-off is processed directly, while lower-grade material above the stockpile cut-off is deferred for processing later in the mine life. The Mineral Resource category breakdown of the mill feed further illustrates the reliance on additional drilling to convert Inferred Mineral Resources to higher-confidence categories in support of future study stages.

16.4 Pit Design

16.4.1 Selective Mining Unit

For the purposes of mine planning and resource-to-mine conversion at the PEA level, a selective mining unit (SMU) consistent with the anticipated scale of open pit operations was assumed. Given a nominal bench height of 15 m, the use of large electric rope shovels, and the preliminary nature of drill-and-blast design inputs at this stage, the effective SMU was assumed to be 15 × 15 × 15 m, consistent with the parent block size of the mineral resource model. This SMU is considered representative of bulk mining

conditions and provides a conservative basis for pit optimization and production scheduling. Dilution and ore loss assumptions associated with this SMU were applied at a conceptual level, and will be reviewed and refined in subsequent study phases as additional drilling, blast design, and equipment selection information becomes available.

16.4.2 Mine Design Geometry Parameters

The open pit design geometry applied in pit optimization and subsequent mine design is based on the geotechnical recommendations presented in Section 16.1.2. Due to the limited geotechnical data set available at the PEA stage, slope design parameters were developed using conservative assumptions informed by rock mass characterization and industry benchmarks. A nominal bench height of 15 m was adopted for all production benches, consistent with the proposed drilling and loading fleet. Bench face angles in the range of 65° to 70° and minimum catch berm widths of 10 m were applied, resulting in inter-ramp slope angles of 41°- 44° consistent with the geotechnical design criteria.

Ramp design parameters were selected to ensure safe and efficient access for the planned haulage fleet. Only double-lane haul roads were considered within the pit. Main haul ramps were designed with a nominal running width of 35 m, allowing for two-way traffic, safety berms, and drainage, and are consistent with the operating requirements of 290-tonne haul trucks. Ramp gradients were limited to 12%, with localized flattening at switchbacks where required. Minimum curve radii of 30 m, based on the turning requirements of the selected haul-truck fleet, were applied. These parameters collectively define the practical mining envelope used for pit design and scheduling.

16.4.3 Optimal Pit Design

The optimal pit shell was generated using the Lerchs–Grossmann algorithm, with revenue factors of 0.81 to reflect Project uncertainty, capital intensity, and the inclusion of Inferred Mineral Resources typical of a PEA-level study.

Key characteristics of the resulting optimal pit are summarized below:

- Length—approximately 2.1 km (W–E)
- Width—approximately 2.2 km (N–S)
- Depth—approximately 675 m below surface
- Bench height—15 m
- Minimum berm width—10 m
- Ramp width—35 m, accommodating two-way traffic for 290-tonne haul trucks.

These dimensions are conceptual and reflect the current stage of mine design; they will be refined in future mine planning and engineering studies. Figure 16-11 and Figure 16-12 show the optimal pit design at the completion of open pit mining, including the low-grade stockpiles and waste rock storage facilities. The low-grade stockpiles are planned to be reclaimed and processed after pit extraction has concluded.

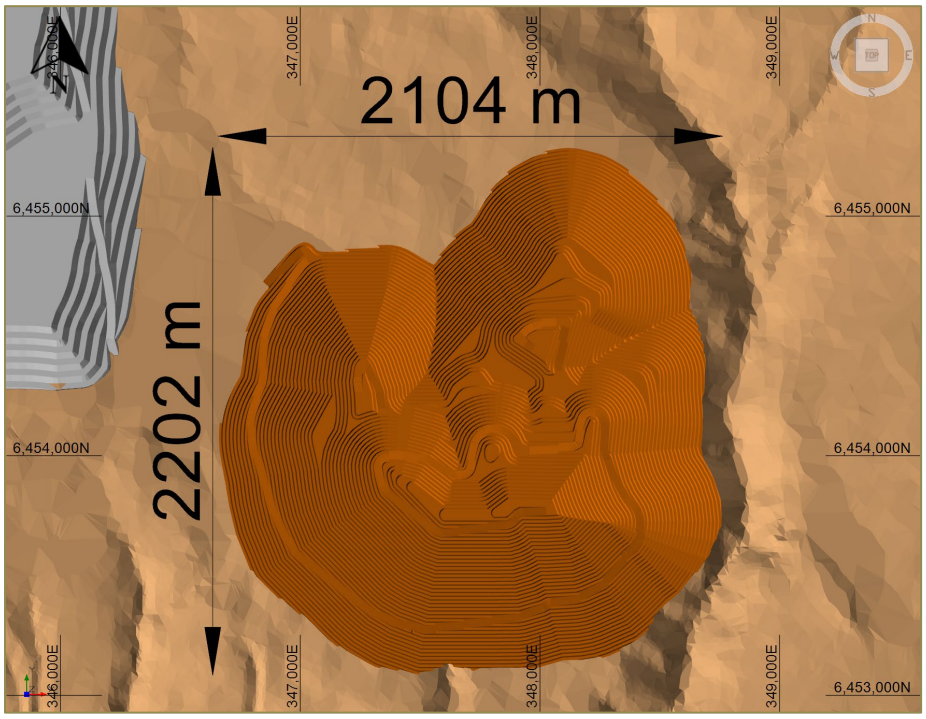


Figure 16-11: Conceptual Optimal Pit Design

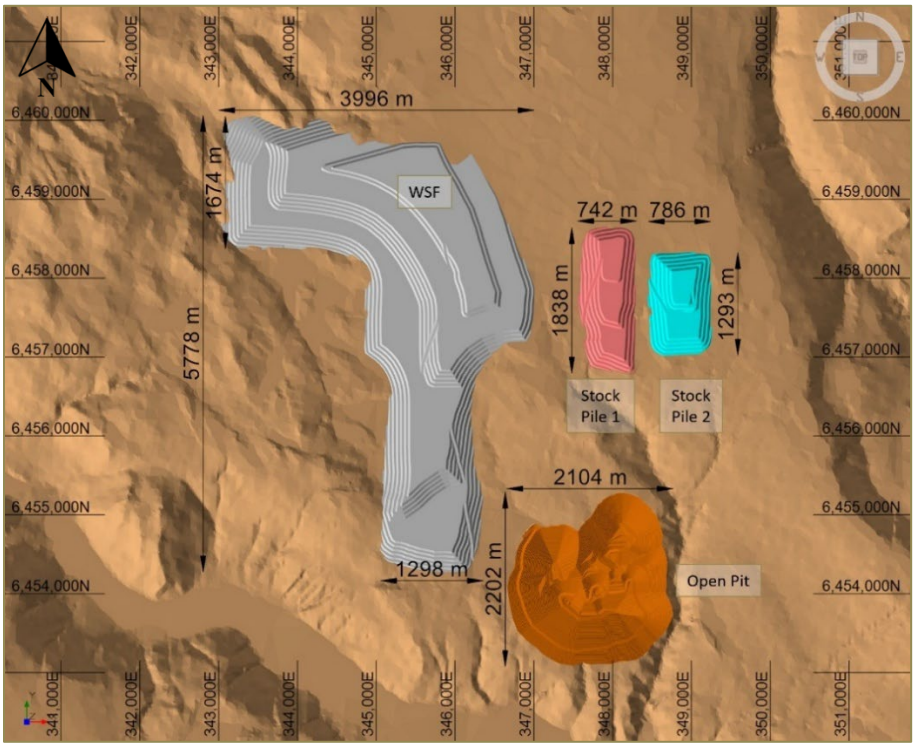


Figure 16-12: Conceptual End-of-Pit Mine Layout Showing the Optimal Pit Design, Low-Grade Stockpiles to be Processed Following Completion of Mining, and Waste Rock Storage Facilities (WSF)

16.5 Waste Storage Facility and Stockpiles

16.5.1 Waste Storage Facility

Waste rock from open pit mining will be placed in a dedicated WSF NW of the pit, selected to minimize haul distances while considering topography, drainage, geotechnical, and mineral claims constraints. The WSF is conceptually designed (Figure 16-13) assuming conventional end-dumping with dozer support, with overall slope angles of approximately 2.5H:1V , lift heights of 20 m with 37° face angles, maximum height of 100m, and an ultimate capacity of approximately 1,644 Mt, sufficient to accommodate life-of-mine waste volumes. Conceptual surface water management includes perimeter diversion ditches and internal drainage controls consistent with B.C. regulatory expectations. Final WSF geometry, stability criteria, material characterization, and closure concepts will be refined in subsequent geotechnical, geochemical, and environmental studies.

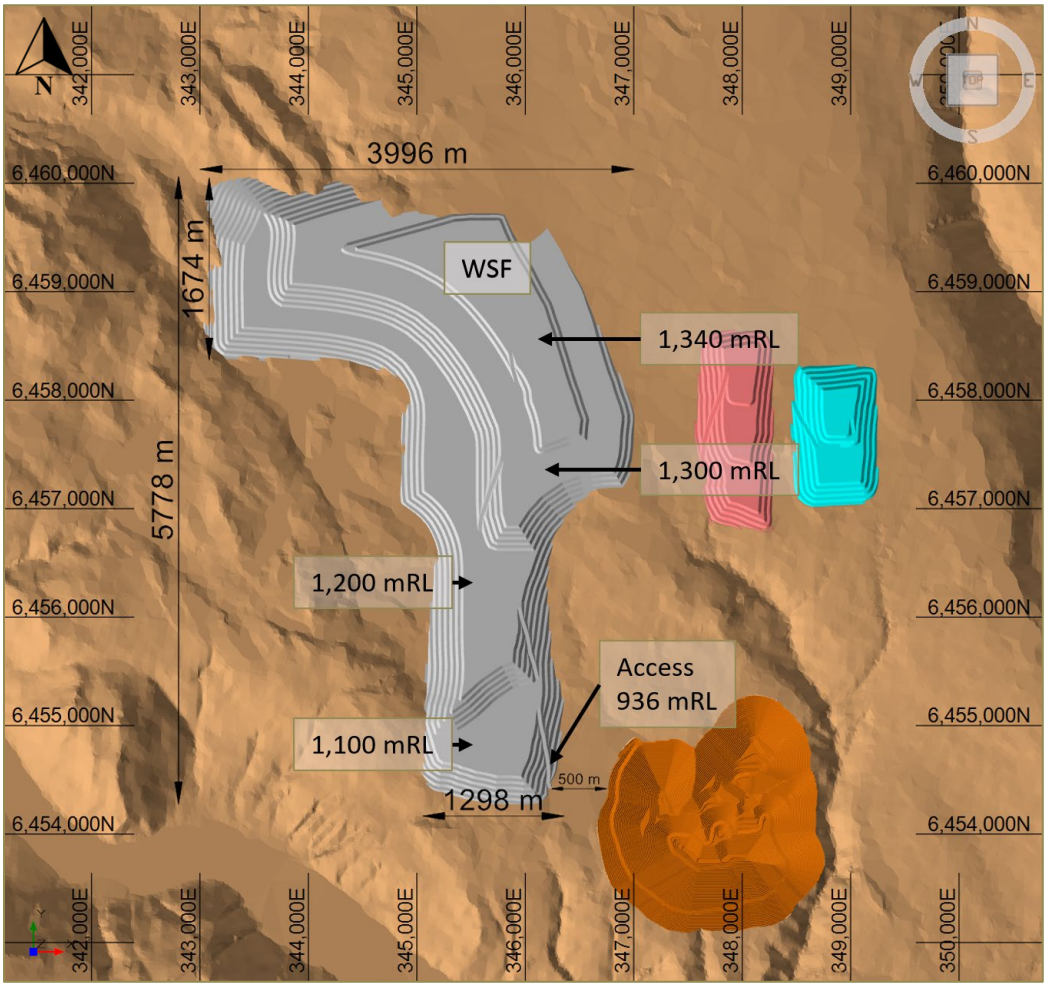


Figure 16-13: Conceptual Waste Storage Facility

16.5.2 Low-Grade Stockpiles (End-of-Life Processing)

Two low-grade stockpiles are planned to enable deferred processing toward the end of mine life and to provide operational flexibility subject to economic conditions. Material grading 0.20%–0.25% CuEq and 0.25%–0.30% CuEq will be segregated into separate stockpiles at the north of the pit, constructed using conventional end-dumping methods. The stockpiles are conceptually designed (Figure 16-14) with lift heights of 100 m, overall slope angles of 2.5H:1V, and estimated capacities of 113 Mt and 128 Mt, respectively. Conceptual surface water management includes perimeter diversion ditches and internal drainage controls consistent with B.C. regulatory expectations. Reclaim strategies, metallurgical performance, and processing schedules remain conceptual and will be refined in future engineering and metallurgical studies.

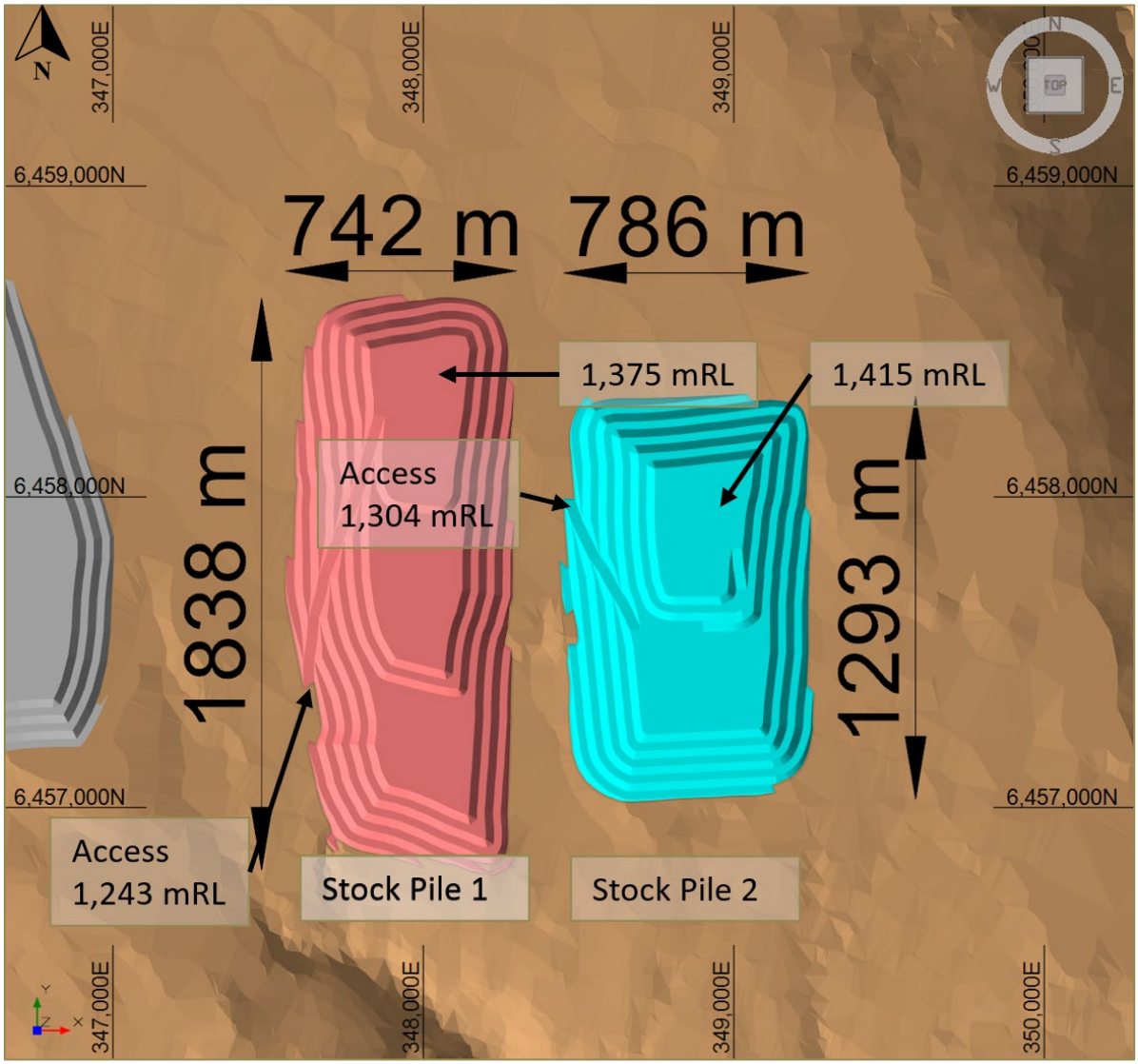


Figure 16-14: Conceptual Stock Pile Design

16.5.3 Run-of-Mine Handling

Material with a copper-equivalent grade greater than 0.30% will be classified as ROM ore and delivered directly to the primary crusher and processing plant without long-term stockpiling (Figure 18-2). Limited ROM surge capacity may be provided adjacent to the crusher to manage short-term variations in mine and plant throughput; however, no dedicated long-term ROM stockpile is assumed at the PEA stage.

16.6 Mine Schedule

A life-of-mine (LOM) production schedule was developed based on the selected optimal pit shell, and the grade-based material handling strategy outlined in Section 16.2.5. The schedule is intended to support the economic evaluation of the Project and reflects a preliminary level of mine planning appropriate for a PEA which includes Inferred resources.

The schedule includes an initial pre-stripping period to provide access to the ore body and to support constructing the initial tailings storage facility and commissioning the processing plant. Following pre-stripping, mining progresses maintaining a conceptual balanced extraction of ore and waste while delivering consistent mill feed. Material with a grade greater than 0.30% CuEq is scheduled for direct processing, while lower-grade material in the 0.20%–0.30% CuEq range is stockpiled for processing later in the mine life.

Annual ore and waste movement rates, mill feed tonnages, and stockpile accumulation are constrained by assumed mining and processing capacities (42 Mt/a), and are summarized in Table 16-4. The mine schedule is conceptual in nature and is based primarily on the optimized pit shell and the current final pit design. The schedule will be refined in future study phases as pit design, phase sequencing, equipment selection, and operating assumptions are further developed.

Figure 16-15 presents the annual mine material movement by category, and the corresponding average copper-equivalent grade over the LOM.

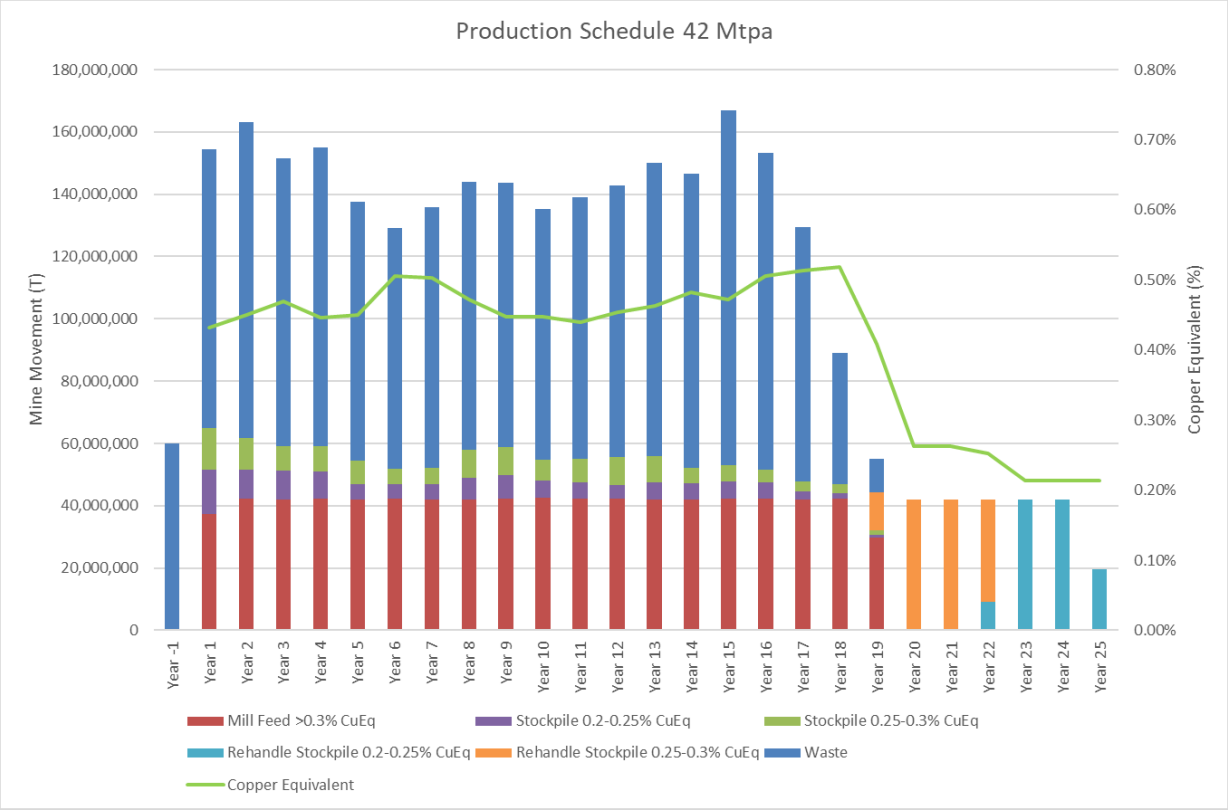


Figure 16-15: Production Schedule 42 Mt/a

16.6.1 Mill Feed

Mill feed is primarily sourced from ROM material with grades greater than 0.30% CuEq, with reclaimed low-grade stockpile material contributing to plant feed during the final years of the mine life. Figure 16-16 illustrates the annual plant feed tonnage and average copper-equivalent grade, with colour coding distinguishing direct ROM feed from stockpile reclaim.

The mill-feed schedule is intended to maintain the assumed plant throughput over the mine life while prioritizing higher-grade direct feed in earlier periods and deferring lower-grade material to later years. This approach is considered appropriate for a conceptual PEA-level schedule; however, the timing and extent of stockpile reclaim will depend on future refinement of the phased mine design, operating sequence, and processing strategy.

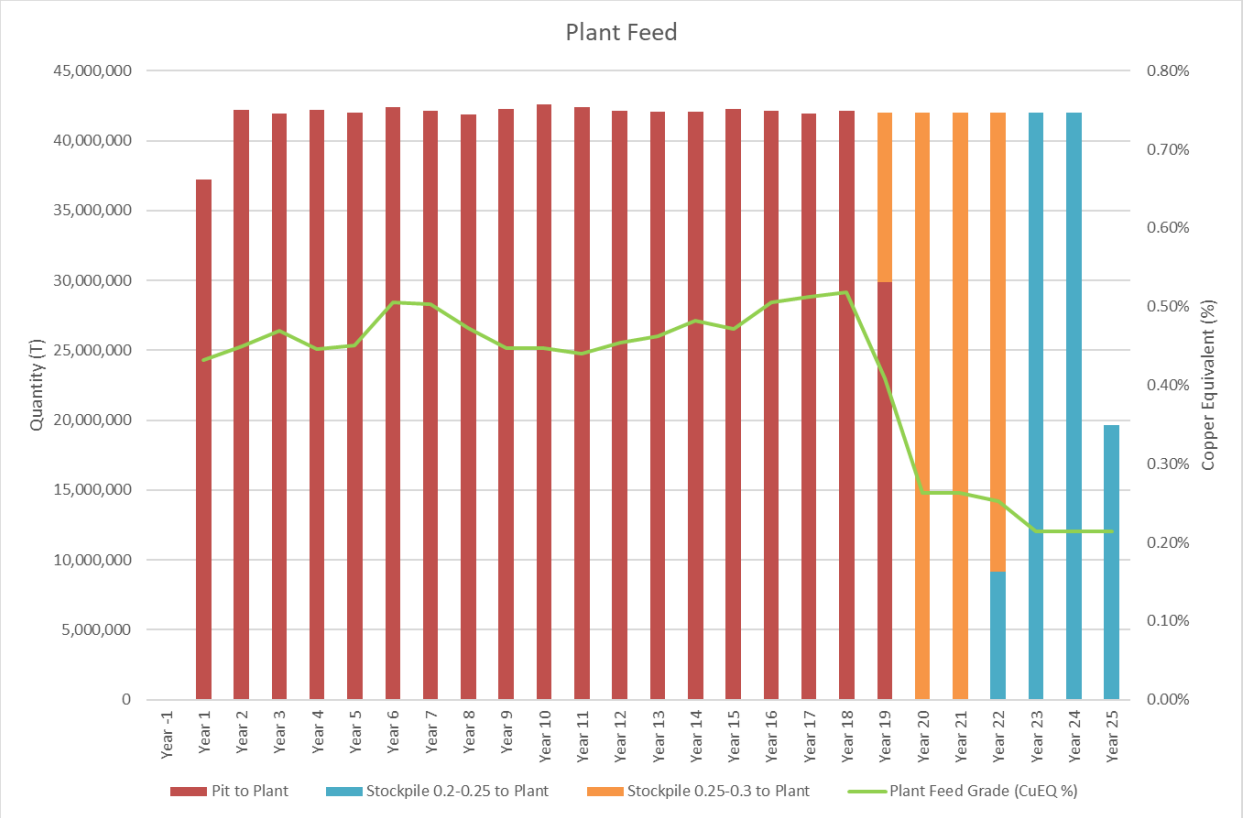


Figure 16-16: Mill Feed Schedule 42 Mt/a

16.6.2 Mine Period Plans

Figure 16-17 illustrates the end-of-period mine plans for selected years within the life-of-mine (LOM) production schedule, specifically Years 1, 5, 10, 15, 19, and 25. These mine period plans are intended to show the conceptual progression of pit development, waste placement, and stockpile evolution over the life of the Project.

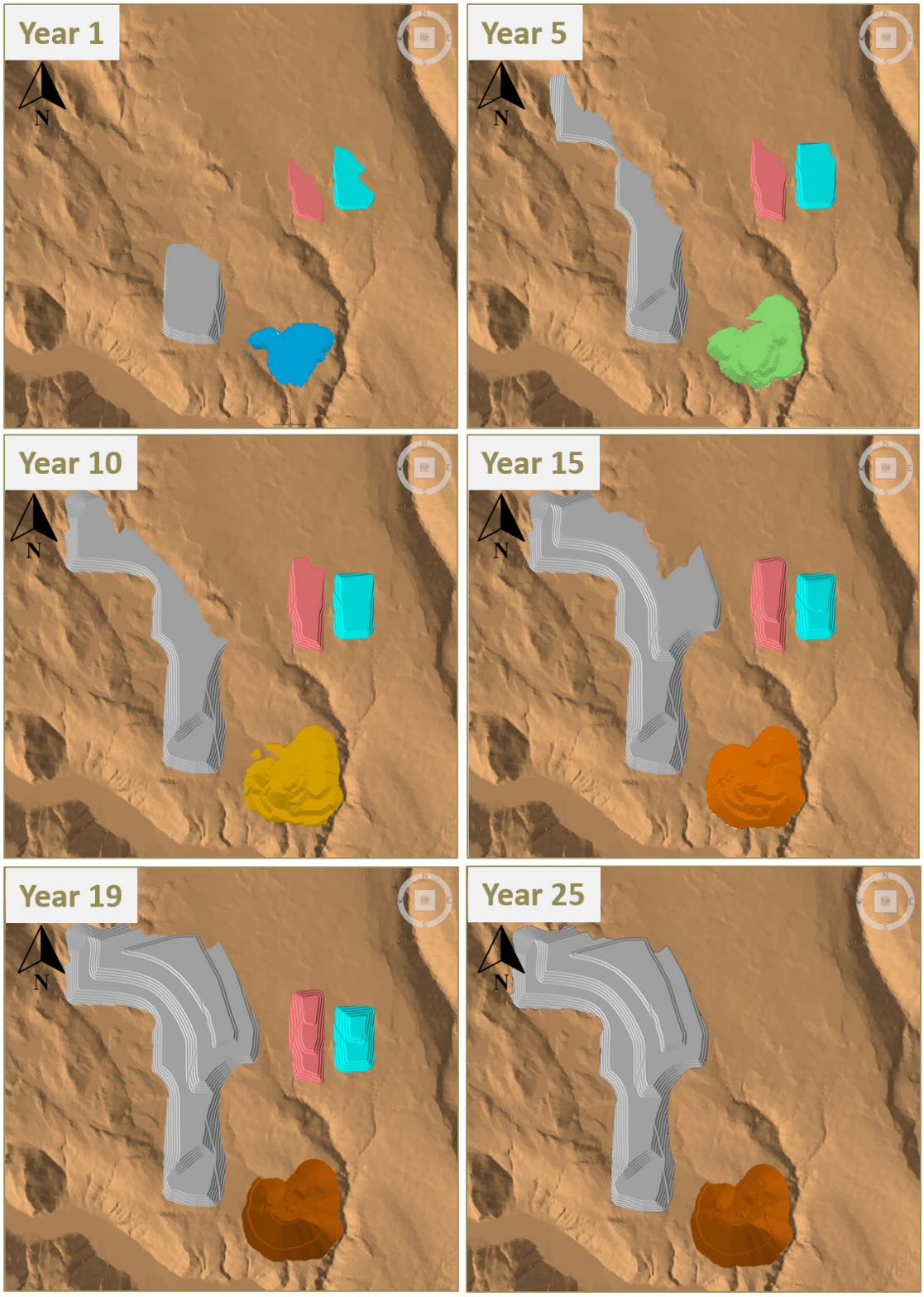


Figure 16-17: Mine Period Plans for Years 1, 5, 10, 15, 19, and 25 of the LOM3 Schedule

Figure 16-18 illustrates the Mineral Resource category distribution of the scheduled mill feed over the mine life. This reflects the current drill spacing and the limitations on spatial continuity associated with the fan-drilling geometry. Consistent with the geological risk assessment, additional drilling between existing drill pods is recommended to improve three-dimensional data distribution, reduce classification uncertainty, and support future resource conversion.

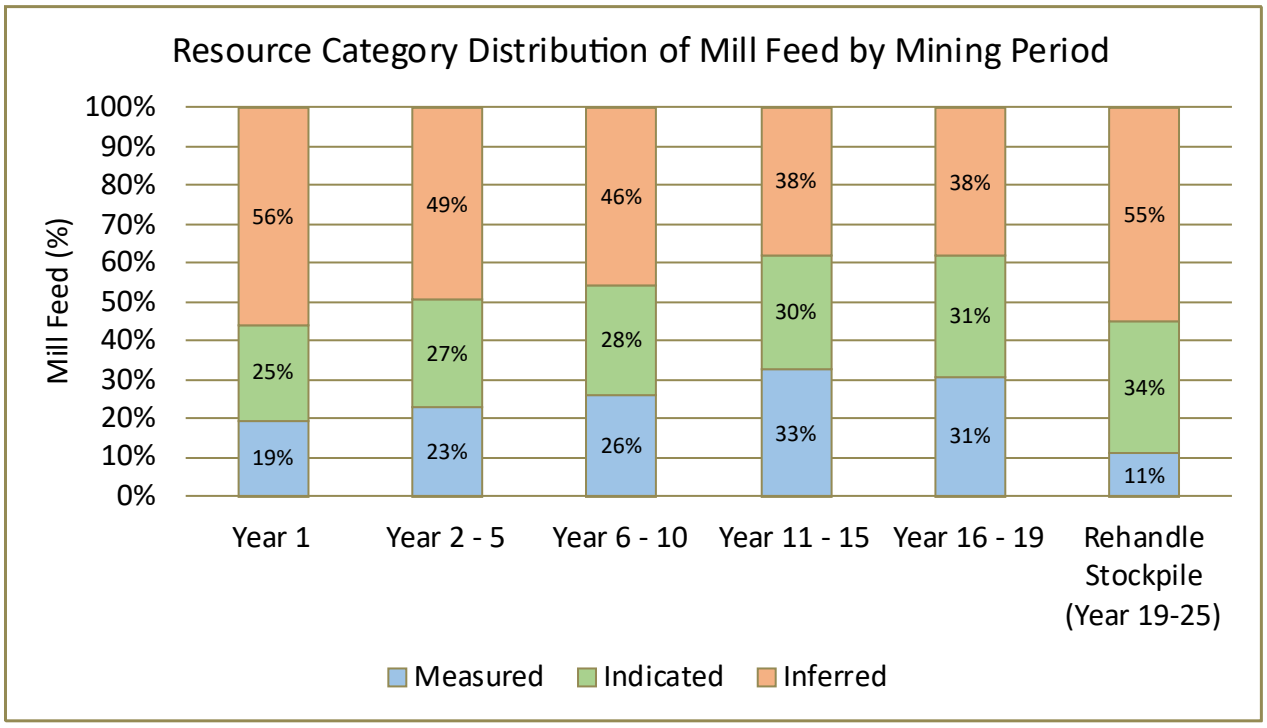


Figure 16-18: Resource Category Distribution of Scheduled Mill Feed by Mining Period

16.7 Underground Consideration

The geometry of the mineralization and the distribution of lower-grade material beneath the selected open pit shell suggest that, if considered in the future, the only potentially viable underground mining method would be a bulk method such as block caving or panel caving. However, the relatively low grades of the remaining material, combined with the depth below the ultimate pit and the capital intensity associated with caving operations, indicate limited economic attractiveness at this stage.

In addition, potential environmental and permitting constraints associated with surface subsidence resulting from caving methods represent a significant risk and may limit the practical applicability of such mining approaches. As a result, no underground mining scenario has been evaluated as part of this PEA, and the Project is considered an open pit-only development case at this stage.

16.8 Equipment Requirements

The mining equipment fleet has been defined at a conceptual level to support the material movement requirements of the selected mine plan and production schedule. Fleet sizing reflects large-scale open pit mining and has been grouped into approximately five-year operating periods to align with major equipment procurement and replacement cycles. As such, the operating periods presented do not correspond directly to the conceptual mining periods described in Section 16.6.

Table 16-8 summarizes the conceptual mining fleet by operating period, defined for equipment procurement and replacement planning. These reflect preliminary equipment selection and sizing developed to support the mine schedule and material movement requirements at the PEA level. The equipment quantities and timing shown in the table form the basis for the mining capital and operating cost estimates presented in Section 21.

Table 16-8: Operating Fleet Summary per period

Operating Fleet Summary	Pre-Production	Operating Period 1 (Year 1–5)	Operating Period 2 (Year 6–10)	Operating Period 3 (Year 11–15)	Operating Period 4 (Year 16–19)	Operating Period 5 (Year 20–25)
Trucks 930E	10	26	30	40	41	10
Shovels 4100XPC	1	2	2	2	2	1
Shovel 7000-11	1	1	1	1	1	0
Drill ZR77	3	7	7	7	7	0
Ancillary	67	69	73	81	78	51
Bulldozer D475A-8	4	4	5	6	6	2
Wheel Loader WE1850-2	1	1	1	1	1	1
Grader-GD955-7	2	2	2	2	2	1
Water Tanker HD785-8	1	1	1	1	1	1
Service Truck	5	5	5	5	5	5
Powder Buggy	2	2	2	2	2	2
Lighting Plant	4	4	5	8	8	8
Pump	2	4	6	10	7	7
Pickup Truck	46	46	46	46	46	24

16.8.1 Drill and Blast Requirements

Drill-and-blast operations are assumed to support a nominal bench height of 15 m, and conventional open pit blasting practices appropriate for bulk mining. Production drilling is planned to use rotary blasthole drills, with the Komatsu ZR77 considered representative of the required drilling capacity at the PEA level. Drill fleet quantities vary over the mine life in response to changes in total material movement. Explosives loading is supported by dedicated powder buggies. Detailed blast design parameters have not been finalized and will be developed in subsequent study phases.

16.8.2 Load and Haul

Loading operations are based on electric rope shovels operating in conjunction with ultra-class haul trucks. Loading equipment options evaluated included the Komatsu 4100XPC electric rope shovel (67.6 m³ bucket) and the PC7000-11 hydraulic shovel (38 m³ bucket). Two 4100XPCs were selected based on their suitability for large-scale bulk mining, compatibility with the assumed 15 m bench height, and alignment with haul truck productivity, and one PC7000-11 for the operational flexibility. Several haul truck sizes were assessed, with the Komatsu 930E-5 (290-tonne) selected as the base-case truck due to favourable payload capacity and compatibility with the selected shovel. Load-and-haul fleet sizes vary over time to accommodate changes in haul distances, pit depth, and production requirements.

16.8.3 Support Equipment

A fleet of ancillary support equipment is assumed to provide essential services for pit operations, including dozing, road maintenance, water distribution, equipment servicing, and general site support. This includes bulldozers, wheel loaders, graders, water trucks, service trucks, pumps, lighting plants, and light vehicles. Support equipment quantities scale with mine activity and are included in the mining capital and operating cost estimates presented in Section 21.

16.9 Uncertainty and Disclaimers

The mining methods, equipment selection, and production schedules presented in this section are conceptual in nature and have been developed to support the PEA–level financial evaluation described in Section 22. As such, they are subject to a number of technical, operational, and economic uncertainties that could materially affect the mine plan, production profile, capital costs, operating costs, and overall Project economics.

Key sources of uncertainty include, but are not limited to:

- Mine scheduling and phase development: The production schedule is based on the selected optimal pit shell and current conceptual pit design, and does not yet reflect a fully engineered sequence of operational pushbacks, haulage validation, or detailed phase interaction analysis. Future refinement of the mine sequence may affect the timing of waste stripping, stockpile development, equipment requirements, and annual feed grade profile.
- Geotechnical stability: Pit slope angles and bench configurations are based on preliminary geotechnical assumptions derived from limited data. Adverse structural conditions or weaker-than-assumed rock mass characteristics could require flatter slope designs, resulting in increased strip ratios, higher waste movement, and increased mining costs.
- Groundwater conditions: Groundwater inflows may require additional dewatering infrastructure or operational controls, which could increase costs and adversely affect pit floor conditions, slope performance, and mining productivity.

- Topography resolution: The mine design is based on the currently available topographic surface. Additional high-resolution topographic surveying may refine local terrain interpretation and could affect the design of pit limits, haul roads, stockpiles, and waste rock storage facilities.
- Waste rock storage facility and stockpile foundations: The layouts and capacities of the waste rock storage facilities and stockpiles are conceptual and have not yet been supported by detailed geotechnical or foundation investigations. Subsurface conditions may affect foundation preparation requirements, facility geometry, drainage provisions, and overall storage capacity.
- Seasonal water inflows and avalanche risk: Seasonal runoff, snowmelt, and localized extreme weather events may affect mine access, pit operations, waste placement, and water management performance. In addition, avalanche risk may influence the design and operability of mine infrastructure and require additional mitigation measures.
- Dilution and mining recovery: Dilution and mining recovery factors are estimated at a conceptual level based on typical open pit mining practices. Actual dilution and recovery will depend on ore body geometry, grade continuity, blast performance, and the effective selectivity of the mining equipment, which may differ from the assumptions used in the economic analysis.
- Resource classification: Portions of the mine plan and production schedule incorporate Inferred Mineral Resources, which have a lower level of geological confidence and cannot be converted to Mineral Reserves. There is no certainty that Inferred Mineral Resources will be upgraded to Indicated Mineral Resources, and the inclusion of such material introduces additional risk to the production schedule and economic outcomes presented in Section 22.

Accordingly, there is no certainty that the mining schedules, production rates, or cost assumptions presented in this section, or the economic results derived from them, will be realized. These assumptions will be reviewed and refined as part of future engineering, geotechnical, hydrogeological, and exploration programs.

17 RECOVERY METHODS

Based on the currently available metallurgical testwork, the proposed process plant has been designed to treat a nominal throughput of 42 Mt/a or 120,000 t/d of copper-gold porphyry ore from the open pit operation. At this stage, the process design reflects a conventional concentrator flowsheet comprising crushing, grinding, flotation, concentrate handling, and tailings management. As additional metallurgical data become available through future test programs, the overall treatment approach is expected to remain broadly unchanged, although product streams and downstream processing options may be refined.

For the purposes of this PEA, three processing cases are evaluated within two principal development scenarios:

- Scenario A1 represents the base concentrator flowsheet using the metallurgical recoveries derived from the testwork presented in Section 13.
- Scenario A2 represents the same base concentrator flowsheet but applies improved copper and gold recoveries reflecting expected performance following further metallurgical optimization and testwork.
- Scenario B represents an expanded flowsheet that retains the base concentrator and adds downstream hydrometallurgical processing for the cobalt-pyrite concentrate, together with a scandium recovery circuit.

Under Scenarios A1 and A2, the concentrator will produce a saleable copper concentrate grading approximately 25% Cu, with payable gold and by-product silver and cobalt credits. No pyrite-cobalt flotation circuit will be included in these two cases.

Under Scenario B, the base concentrator flowsheet will be expanded to include a pyrite-cobalt flotation circuit, shown in Figure 17-1 as “Scenario B only”. This additional circuit will produce a cobalt-enriched pyrite concentrate, which will be further processed through a downstream hydrometallurgical circuit illustrated in Figure 17-2. In this case, the pyrite concentrate will be roasted to produce sulphuric acid and a calcine. The calcine will be then processed for recovery of cobalt, gold, silver, and copper, while the sulphuric acid will be used to leach a portion of the flotation tailings for scandium recovery. The pyrite-cobalt flotation circuit, hydrometallurgical circuit, and scandium circuits are therefore unique to Scenario B and are not part of the A1 or A2 cases. Consistent with the staged development approach adopted for this study, commissioning of the hydrometallurgical circuit is assumed to commence in Year 3 of operations.

As described in Section 13, the available metallurgical testwork comprises early-stage characterization and bench-scale testing conducted under standard laboratory conditions to demonstrate metal recoverability for inclusion in the CuEq estimate. No flotation optimization program has yet been completed, and more advanced locked-cycle or pilot-scale flotation testing has not been undertaken. Accordingly, the metallurgical recoveries used in Scenario A1 are considered conservative. Scenario A2 has therefore been developed to reflect improved copper and gold recoveries considered achievable with

further process optimization and additional metallurgical testwork, consistent with performance typically achieved in comparable Cu-Au porphyry flotation circuits.

Table 17-1 summarizes the assumed recovery values applied in each scenario.

Table 17-1: Net Recovery for Each Scenario

Net Recovery	Scenario A1	Scenario A2	Scenario B
Cu Recovery	80%	89%	85%
Au Recovery	66%	75%	89%
Ag Recovery	53%	53%	68%
Co Recovery	30%	30%	78%
Sc Recovery	N/A	N/A	72%

17.1 Process Flowsheet

The unit processes selected for the proposed design of the process facilities are based on the results of a limited metallurgical test program performed at the Sepro Laboratories at Langley, B.C., and using industry-proven and conventional recovery processes for recovering other metals from similar deposits, except for the recovery of scandium, which is a newly developed chemical process. The metallurgical processing procedures selected for the design will produce a saleable high-grade copper concentrate containing by-product gold (Scenario A1/A2), while the pyritic concentrate will be roasted to produce sulphuric acid; the resulting calcine will be treated to recover cobalt, gold, and silver. The sulphuric acid produced will be used in the scandium-recovery process (Scenario B).

The envisaged treatment plant will consist of a primary crushing stage followed by a grinding and classification circuit, which in turn is followed by the first flotation circuit to recover and upgrade copper from the feed material (Scenario A1/A2), and a second flotation circuit to recover an upgraded cobalt content into a pyrite-base concentrate (Scenario B). A gravity concentration circuit has been considered for inclusion in the process to enhance gold recovery, but that requires further testwork for its inclusion in the process. As shown in the simplified flowsheet (Figure 17-1), both flotation concentrates will be thickened and filtered, and stored in respective stockpiles until the copper-gold concentrate is shipped to the smelter, and the cobalt-pyrite concentrate is processed for cobalt, gold, and silver recovery.

The final flotation tailings will be dewatered in the tailings thickener. Thickened slurry will be delivered to a distribution tank where the required amount of tailings will be diverted to the scandium recovery plant, while the excess tailings will be deposited in the tailings collection pond. This deposition design will allow for control of water management including the reuse of return water from the tailings storage facility which will be pumped to the process-water tank together with the water from the tailings thickener overflow. The overflow water from the concentrate thickeners will be collected and reused in the respective flotation circuit. Fresh water will be used for gland service for the pumps, for reagent preparation purposes, and for the fire-water circuit.

The process plant will consist of two identical processing circuits each with the following main unit operations and facilities:

- Run-of-mine (ROM) ore-receiving bin
- Gyratory crusher
- Crushed-ore stockpile
- Crushed-ore stockpile ore reclaim to feed the grinding circuit
- Semi-autogenous grinding (SAG) and ball mill grinding circuit incorporating cyclones for classification
- Copper-gold flotation circuit conditioning tanks and rougher flotation stage
- Copper-gold rougher concentrate regrinding and classification circuit
- Copper-gold cleaner flotation circuit
- Copper-gold concentrate thickening, filtration, storage, and dispatch
- Scenario B additions:
  - Cobalt-pyrite flotation circuit including conditioning tanks and rougher flotation stage
  - Cobalt-pyrite rougher concentrate regrinding
  - Cobalt-pyrite cleaner flotation circuit
  - Cobalt-pyrite concentrate thickening, filtration, and dispatch to roaster facility
- Flotation tailings thickening
- Distribution of tailings to scandium recovery facility, and disposal of excess tailings to the tailings storage facility
- Reagent preparation facility.

The simplified flowsheet is shown in Figure 17-1.

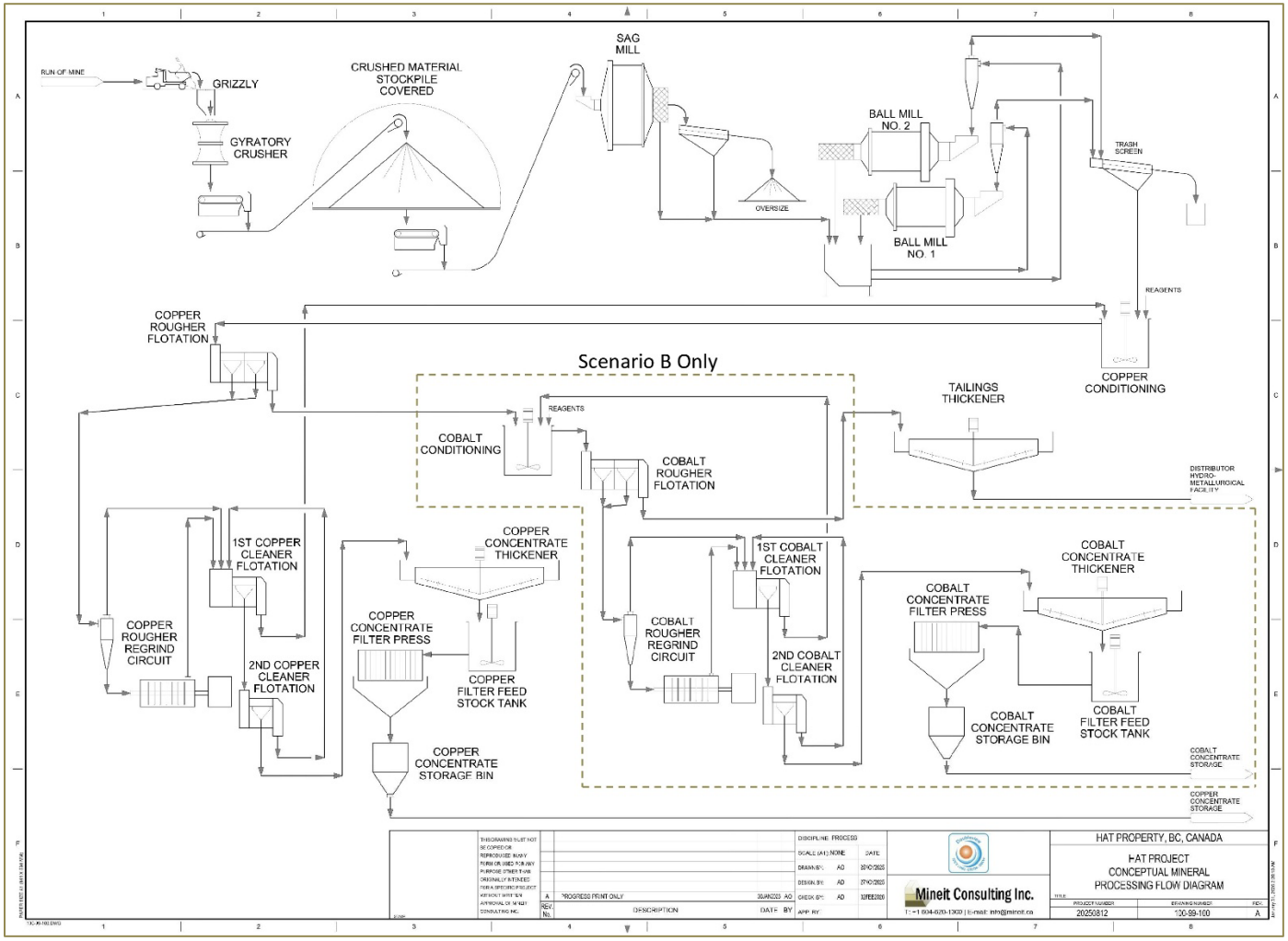


Figure 17-1: Conceptual Process Flowsheet for the Hat Project Concentrator. The Cobalt-Pyrite Flotation Circuit Enclosed by Dashed Line is Included in Scenario B Only.

17.2 Major Design Criteria

The concentrator has provisionally been designed to process 120,000 t/d, equivalent to 42,000,000 t/a, and consists of two identical processing lines each designed for 60,000 t/d. The major preliminary design criteria used in the anticipated design are outlined in Table 17-2.

Table 17-2: Major Design Criteria

Criteria	Unit	
Number of Days per Year	d	365
Processing Days per Year	d	350
Total Processing Throughput per Day -Two Processing Lines	t/d	120,000
Processing Throughput per Day for One Processing Line	t/d	60,000
Crusher Availability	%	70.0
Grinding and Flotation Availability	%	92.0
Concentrate Filtration Availability	%	80.0
Total Crushing Rate	t/h	7,143
Crushing Rate for One Processing Line	t/h	3,571
Total Milling and Flotation Process Rate	t/h	5,435
Milling and Flotation Process Rate for One Processing Line	t/h	2,717
SAG Mil Feed Size, 80% Passing	mm	150
SAG Mill Transfer Size, 80% Passing	µm	1,400
Ball Mill Feed Size, 80% Passing	µm	1,400
Ball Mill Product Size, 80% Passing	µm	75
Ball Mill Circulating Load	%	300
Bond Ball Mill Work Index	kWh/t	19.0
Bond Abrasion Index	g	0.260
Copper-Gold Rougher Concentrate Regrind Size, 80% Passing	µm	30
Cobalt-Pyrite Rougher Concentrate Regrind Size, 80% Passing	µm	30

The process plant design parameters discussed in this section are based on indicated but limited test work results, and general good industry practices and experience with similar copper-gold porphyry process plant designs. The flotation results from the testwork Sepro undertook have been used as a guide for the flotation circuit design to treat the Hat Deposit material. The crushing circuit will be typical of similar applications. The SAG and ball mill grinding circuit configuration and mill sizes are conceptual and will be confirmed once grinding test data become available. The flotation cells were sized based on the flotation residence times as determined during the laboratory testwork and adjusted by scale-up factors typically applied in the design of flotation plants. Typical flotation-cell design parameters have been used in the design of the two flotation circuits—copper-gold and cobalt-pyrite. Similarly, typical industry-standard concentrate dewatering units have been incorporated into the design, and more accurate sizes of

equipment will become available with further test programs, which will enable the equipment to be appropriately sized.

17.3 Process Plant Design and Description

17.3.1 Operating Schedule and Availability

The two proposed crushing and processing plants will be designed to operate on the basis of two 12-hour shifts per day, for 350 mining and processing days per year, each treating 60,000 t/d.

Typically, the equipment availability for the crusher circuit will be 70%; the SAG and ball mill grinding circuits, and the flotation circuit availabilities will be 92%; and the concentrate filtration availability will be 80%. This will allow for sufficient downtime for the scheduled and unscheduled maintenance, and unplanned operational stoppages of the crushing circuit and the rest of the process plant equipment.

17.3.2 Crushing

The primary crushing operation will use a gyratory crusher for the mined material. The ROM rock will be discharged directly onto a grizzly to prevent oversize material from entering the crusher feed chamber. A rock breaker will be available to reduce the size of any oversize material and prevent blocking of the ore flow into the crusher. The major items of equipment in crushing area will include the following:

- Static grizzly with anticipated 1.4 m x 1.4 m openings
- Rock breaker
- Gyratory crusher, model KM60-89 or similar.

The crusher will reduce the ROM material to P_{80} 140 mm. The crushed material will be conveyed to the respective crushed-ore stockpile. Each ore stockpile that feeds a grinding circuit will be a covered facility.

17.3.3 Ore Stockpile and Reclaim

The crushed-ore stockpile is a production surge facility that will allow for a steady feed to the grinding circuit.

The major equipment and facilities in this area includes:

- Ore stockpile with a live capacity of 60,000 tonnes
- Each stockpile will be covered
- Reclaim apron feeders (3 operational, 1 standby)
- Mill-feed conveyor belts complete with metal detectors, self-cleaning magnets, and belt-tear detectors
- Belt scale for each conveyor
- Dust collection system.

Each ore stockpile will have a live capacity of 60,000 tonnes, equivalent to the throughput rate of one operating day. The material will be reclaimed from the stockpile using apron feeders at a nominal rate of 1,087 t/h. The apron feeders will load the crushed rock onto the conveyor belt that will feed the SAG mill. Each SAG mill feed conveyor belt will be equipped with a belt scale.

The crushed-ore stockpile and reclaim area will be equipped with a dust collection system to control fugitive dust that will be generated during conveyor loading, discharge onto the stockpile, and the transportation of the feed material to the SAG mill.

17.3.4 Grinding and Classification

The anticipated grinding circuit will consist of two lines each of one SAG mill and two ball mills in closed circuit with the classifying cyclones. Grinding will be conducted as a wet process at a total nominal rate of 5,435 t/h. The major items of equipment in the grinding and classification circuit will include:

- Two SAG mills, each with estimated dimensions of 12.2 m diameter x 6.7 m long (40.0 ft x 22.0 ft)
- Two SAG mill discharge screens
- Four ball mills, each estimated dimensions of 8.6 m diameter x 12.2 m long (28.0 ft x 40.0 ft)
- Two cyclone feed pump-boxes
- Two sets of cyclone feed slurry-pumps
- Four cyclone clusters with manifolds
- SAG mill liner handler
- Ball mill liner handler
- Two mass flow meters
- Two particle-size analyzers
- Two on-line sampler system on the flotation feed line
- Two vibrating trash screens.

For each line, the product from each SAG mill will be discharged into a common cyclone feed pump-box. The slurry will be pumped to two cyclone clusters for classification, one for each ball mill. The cut size for the cyclones will be P_{80} 75 µm, and the circulating load will be 300%, with the cyclone underflow returning to the respective ball mills as feed material.

The fresh-ore feed to the grinding circuit will be 2,717 t/h, and this will also constitute the feed rate to the copper–gold flotation circuit. All the SAG mill and the ball mills are anticipated to operate at a critical speed of 75%. Dilution water will be added to the grinding circuit mills as required.

The cyclone overflows from the classification circuit will pass over a vibrating screen to remove trash material prior to being discharged into the copper-gold flotation conditioning tanks ahead of the first rougher flotation stage. The pulp density of the cyclone overflow slurry will be approximately 34% solids.

Provision will be made for the addition of lime to the cyclone feed pump-box for the initial adjustment of the pH of the slurry in the grinding circuit ahead of the flotation process, together with other flotation reagents as required.

Grinding media will be added to the mills to maintain grinding efficiency. Steel balls will be periodically added to each mill using a ball charging kibble. The mill motors will likely be air-cooled. but this would be subject to vendor recommendation.

Provision will be made in each grinding circuit for a gravity gold concentrator circuit should future testwork indicate that gravity-recoverable gold is present in the mined material.

17.3.5 Flotation and Regrind Circuits

The milled pulp will be subjected to the flotation process to recover the targeted minerals, namely to produce a high-grade copper-gold concentrate, followed by a cobalt-bearing pyrite-based concentrate. Each rougher stage concentrate will feed one of two regrind mills prior to upgrading the product in the respective cleaner flotation circuit. Tank-style flotation cells will be used throughout the flotation circuit for the rougher stages and the initial cleaner flotation stages. A column cell may be installed as a third cleaner stage to ensure that a marketable grade of copper-gold concentrate will be attained, but this is subject to confirmation by future testwork, as is the detailed sizing and number of flotation cells in both circuits.

Each processing line for the proposed copper-gold flotation circuit and the cobalt-pyrite flotation circuit will each include the following equipment:

- Two conditioning tanks with estimated dimensions of 9.0 m diameter x 10.0 m high (including freeboard)
- Flotation reagent addition facilities
- Rougher flotation-tank cells, anticipated to be one line of four cells, 600 m^3 volume for each cell, for each circuit
- Rougher concentrate regrind mills, anticipated to be an IsaMill or similar
- Classification cyclone cluster (one for each rougher concentrate regrind mill)
- First cleaner flotation circuit—four cells, each of with 100 m^3 volume for the copper-gold circuit, and two cells each of 100 m^3 volume each for the cobalt-pyrite circuit
- Second cleaner flotation circuit—three cells, each of with 100 m^3 volume for the copper-gold circuit, and two cells each of 100 m^3 volume for the cobalt-pyrite circuit
- Third cleaner flotation circuit column cells, possible future allowance
- Pump-boxes and standpipes
- Slurry and concentrate pumps as required
- Particle-size analyzers, one for each concentrate regrind circuit
- Froth cameras
- On-line sampling system in each circuit.

The cyclone overflow from each grinding circuit will feed the copper-gold flotation circuit conditioning tanks sequentially by gravity flow from the grinding circuit classification cyclone cluster, after passing over the trash screen. The slurry will be monitored for P_{80} size, and flotation feed samples will be taken periodically for process control and metallurgical accounting purposes.

The copper-gold rougher flotation circuit will consist of a train of four cells, each with 600 m^3 cells, which will operate in the manner described below.

The cyclone overflow will discharge over the trash screen and into the first of two conditioning tanks in series. Each conditioning tank will be equipped with an agitator and has been sized for a retention time, or conditioning period, of about 5 minutes, which will ensure that the required slurry pH value is at the correct level, and that the required reagents are mixed into the slurry. The slurry will be conditioned in the conditioning tanks at the design feed rate of 2,717 t/h. The conditioning tanks will also receive the Cleaner 1 tailings for reprocessing. The flotation reagents added will be lime for the pH control value, set at between 10.5 to 11.0; the pyrite depressant sodium cyanide (NaCN); the collector reagents sodium isopropyl xanthate (SIPX) and sodium dialkyl dithiophosphate (A3477); and the frother reagent methyl-isobutyl carbinol (MIBC). Provision will be made for the staged addition of the reagents in the rougher circuit.

The conditioned slurry will overflow from the second conditioning tank into the feed end of the rougher flotation-tank cell line. Air injection will facilitate the flotation process. The copper minerals (mainly chalcopyrite) will be selectively floated into a copper-gold rougher concentrate away from the other minerals and the gangue present in the slurry. The rougher concentrate will constitute approximately 10% of the plant feed mass, which would be a typical mass recovery value (although the Sepro laboratory scoping tests indicated a 20% mass recovery for this stage). The rougher tailings will be sampled automatically for process control and metallurgical accounting purposes prior to discharge into the pump-box for delivery to the first of two conditioning tanks in the cobalt-pyrite flotation circuit.

To completely liberate the fine-sized grains of the copper minerals from the pyrite and other gangue constituents, and to enhance the upgrading of the copper in the copper-gold concentrate, a regrind step of the rougher concentrate will be incorporated. The reground concentrate will feed the cleaner circuit, and two stages of flotation cleaning have been chosen so that a final product of acceptable copper grade and recovery will be achieved. Allowance will be made for a third cleaner stage should future testwork show this to be necessary.

The copper-gold rougher concentrate will be delivered to one of two regrind mill cyclone-feed pump-boxes and the slurry will be pumped to the regrind densification cyclone cluster. The regrind circuit cyclone will separate the coarser cyclone underflow that will be the feed for the regrind mill and the cyclone overflow product. The cyclone overflow material will be pumped to the first cleaner-stage feed pump-box where it will be recombined with the regrind mill product, together with the tailings from the second cleaner stage. The regrind mills will be appropriately sized when locked-cycle test data and mass flowrates become available. Certain concentrators also have a first cleaner-scavenger cell option, but that would require confirmation by testing. The concentrate from the first cleaner-scavenger flotation stage would then report to the rougher stage conditioning tank, while the cleaner-scavenger tailings would be

directed to the final tailings pump-box. In this case, the first cleaner tailings will be returned to the rougher flotation stage via the conditioning tanks.

The flotation reagents added in the cleaner flotation circuit will be lime for pH control, the collector reagents SIPX and A3477, and the frother reagent MIBC. The first cleaner concentrate will be combined with the tailings from the third cleaner stage, if this is incorporated into the circuit, to feed the second cleaner stage. At this stage of the project, it is anticipated that the concentrate from the second cleaner stage will constitute the final-product copper-gold concentrate with an anticipated copper grade of 25%. The copper-gold concentrate will feed directly to the copper-gold concentrate thickener for dewatering and filtering. This stream will be sampled with the on-line analyzer and will be used for process control. As mentioned earlier, the second cleaner tailings will be recycled to the first cleaner flotation stage

Depending on the outcome of the future flotation-test program, it is possible that the kinetics of the rougher flotation stage may indicate the requirement for an additional extension to the rougher circuit, the so-named rougher-scavenger stage. If this is determined necessary, then the concentrate from the rougher-scavenger tanks will require recirculation to the rougher conditioning tanks, with the installation of the necessary piping and pumps.

Provision will be made for the copper-gold concentrate thickener overflow water to be recycled for use in the copper-gold flotation circuit as process water.

The tailings from the copper-gold rougher flotation circuit will be the feed material for the cobalt-pyrite flotation circuit (Scenario B). The slurry will be pumped to the cobalt-pyrite conditioning tanks, and the general flow of this circuit will be the same as described earlier for the copper-gold flotation circuit; at this stage of the project these will have the same equipment sizes.

The cobalt-pyrite rougher-flotation circuit will operate in the following manner. The conditioning tanks will ensure that the slurry pH value is at the required level, 6.7, and that the required reagents will be thoroughly mixed into the slurry. The flotation reagents added will be sulphuric acid for pH control; the pyrite activator copper sulphate; the collector reagent PAX, and the frother reagent MIBC.

The cobalt-pyrite flotation circuit will be the same as for the copper-gold, with two cleaning stages, and a regrind-mill circuit consisting of two regrind mills for regrinding the rougher cobalt-pyrite concentrate. The cobalt-pyrite flotation rougher tailings will be sampled automatically for process control and metallurgical accounting purposes prior to discharge into the pump-box for delivery to the distribution box where an amount of the tailings material will be redirected to the scandium recovery process plant, while the balance of the tailings will be discharged to the tailings thickener.

The cobalt-pyrite rougher concentrate regrind circuit, and the cleaner flotation circuit, will operate in the same way as described for the copper-gold circuit.

The flotation reagents added in the cleaner flotation circuit will be sulphuric acid for the pH control, the pyrite activator copper sulphate, the collector reagent PAX, and the frother reagent MIBC.

At this stage, the concentrate from the second cleaner stage will constitute the final-product cobalt-pyrite concentrate with an upgraded cobalt content suitable for the subsequent cobalt recovery process. The cobalt-pyrite concentrate will feed directly to the cobalt-pyrite concentrate thickener for dewatering. This concentrate stream will also be sampled with the on-line analyzer and will be used for process control.

The cobalt-pyrite concentrate thickener overflow water will be reused in the cobalt-pyrite flotation circuit as process water, and will be kept separate from the water from the copper-gold flotation circuit.

17.3.6 Concentrate Handling

Both second (final) cleaner flotation concentrates will be thickened, filtered, and stored. The copper-gold concentrate will be shipped off site to a smelter (Scenario A1/A2/B), while the cobalt-pyrite concentrate will be stored and serve as feed material to the roasters for cobalt recovery and sulphuric acid production for the subsequent scandium recovery process (Scenario B).

The two final copper-gold flotation concentrates will be combined into one concentrate handling circuit. Similarly, the two cobalt-pyrite flotation concentrates will also be combined. Each respective concentrate handling circuit will conceptually have the following equipment:

- Concentrate thickener, estimated to have a diameter of 23 m
- Concentrate thickener overflow standpipe
- Flocculant preparation and addition system
- Concentrate thickener underflow slurry-pumps
- Process water tank and pumps
- Concentrate stock tank, 8 hours storage
- Concentrate filter-press feed-pumps
- Concentrate filter press
- Concentrate moisture content, 10%
- Filter-press washing and filtrate handling equipment
- Dewatered concentrate storage, 7 days storage
- Copper-gold concentrate dispatch facility including bagging, weighing, and sampling.

Each concentrate produced will be pumped from the final cleaner flotation stage to the concentrate thickener where flocculant will be added to aid the settling process. The thickened concentrate will be pumped to the concentrate stock tank using thickener underflow slurry-pumps. The underflow density will be at about 60% solids. Each concentrate stock tank will be an agitated tank that will serve as the feed tank for the respective concentrate filter, which will be a pressure filter-unit. Since filtration with a filter-press unit will be a batch process, the concentrate stock tank will also act as a surge tank for the filtration operation. The filter press will dewater the concentrate to produce a final concentrate with a moisture content of about 10%. The filtrate will in each case be returned to the respective concentrate thickener. The filter press solids, namely the dewatered concentrate, will be discharged directly onto the respective

concentrate stockpile. The dewatered concentrate will be stored in a designated storage facility providing storage for about 7 days production. The copper-gold concentrate will be bagged and periodically loaded into trucks for dispatch off the property. The cobalt-pyrite concentrate will be stored and will serve to feed the roaster operation to produce the sulphuric acid required for use in the cobalt and scandium recovery circuit (and in the cobalt-pyrite flotation circuit where it is used for pH control). Both concentrates will be sampled and assayed as required for metallurgical accounting purposes.

The filtrate and the thickener overflow solution from each concentrate thickener will be collected in the respective process water tank for recycling within the respective flotation circuit mill circuit. A portion of the overflow solution will be used within the respective filtration area as filter wash-water.

17.3.7 Tailings Handling

The anticipated flotation tailings handling circuit will constitute the following. The rougher flotation tailings from the cobalt-pyrite flotation circuit will be pumped to a distribution tank where the required amount of tailings will be diverted from the main stream of the flotation tailings and directed to the scandium recovery processing facility, while the balance of the flotation tailings will be discharged to the tailings storage facility for final deposition. This balance of the remaining flotation plant tailings material will initially be dewatered in the tailings thickener, with flocculant added to facilitate the solids settling. The slurry will be thickened to about 50% solid during tailings dam construction (cyclone sands) and 65% solid during winter deposition. The tailings thickener overflow-solution will be collected, together with the excess water from the tailings storage facility, and these will be recycled to the process plant for reuse as general process water.

The flotation tailings handling circuit will have the following equipment in each process line:

- Distribution tank
- Tailings thickeners, each estimated to have a diameter of 110 m
- Tailings-thickener overflow standpipe
- Flocculant preparation and addition system
- Tailings-thickener underflow slurry pumps
- Slurry transfer-pumps to scandium plant from the distribution tank
- Tailings pond return-water system
- Process-water tank and water distribution pumps.

17.3.8 Reagent Handling and Storage

Various chemical reagents will be added to the process slurry streams to facilitate recovery of the copper-gold and a cobalt-pyrite concentrates during the flotation process. Although two reagent preparation systems are presently considered, it is possible that a single section serving both processing lines may be installed, but this would be dependent on the process plant layout and reagent delivery distances involved. Each preparation section of the various reagents will require:

- A bulk-bag handling system
- A bulk-container handling system
- Mixing and holding tanks, as required
- Reagent-delivery metering pumps
- A lime silo and pneumatic off-loading system
- A lime slaking and distribution system
- Fresh-water supply
- Sump pumps
- Eye-wash and safety showers
- Applicable safety equipment.

Different chemical reagents will be added to the grinding and flotation circuit to modify the mineral particle surfaces and enhance the floatability of the valuable mineral particles respectively into the copper-gold and cobalt-pyrite concentrate products. Fresh water will be used in the making up or the dilution of the various reagents that will be supplied in powder or solid form, or which require dilution prior to addition to the slurry. These solutions will be added to the various addition points of the flotation circuits and streams using metering pumps.

The PAX and SIPX are water-soluble collector reagents that will arrive at the plant in 1-ton bulk bags. Each reagent will be dumped into suitable-sized hoppers for withdrawal of predetermined quantities for mixing with water to the required solution strength. The reagent will be made up in a mixing tank, then transferred to the holding tank, from where the solution will be pumped to the addition points in the circuit using metering pumps.

Flocculant will be prepared in the standard manner as a dilute solution with about 0.30% solution strength. The flocculant solution will be further diluted in the thickener feed-well.

Lime, as quicklime, will be delivered in bulk and will be off-loaded pneumatically into a silo. The lime slurry will then be prepared in a lime slaking system as a 20% concentration slurry. This lime slurry will be pumped to the addition points using a closed-loop system. The valves will be controlled by pH monitors that will control the amount of lime added.

Sulphuric acid reagent will be delivered in 2-ton carboys. To control the slurry pH the acid will be transferred into a holding tank for dilution with fresh water prior to pumping to the addition points ahead of the cobalt-pyrite flotation circuit.

The A3477 reagent will be used in the copper-gold flotation circuit as the collector reagent for the flotation of copper and gold minerals. The A3477 will arrive in drums and be transferred to a holding tank prior to distribution within the plant. The reagent will be pumped directly to the flotation circuit using metering pumps.

Sodium cyanide is used as the depressant for pyrite in the copper-gold flotation circuit. It is a solid and will typically arrive in 1-ton bags where the solids will be drawn out into a mixing hopper and collected in

a receiving tank, typically as a 2% solution strength, prior to delivery to the addition points by metering pumps. The hazardous nature of this reagent will result in the preparation area being fenced off, and only trained operating staff will be allowed access to the area.

Copper sulphate is used as the activator for pyrite in the cobalt-pyrite flotation circuit. It is a solid, and will typically arrive in 500 kg bags where the crystals will be drawn out into a mixing hopper and collected in a receiving tank as a 20% solution prior to delivery to the addition points by metering pumps.

The reagent MIBC will be used in both flotation circuits to provide the froth phase of the flotation process. MIBC will arrive in bulk containers and will be transferred to a holding tank prior to distribution within the plant. MIBC will be pumped directly to the flotation circuit as a 100% solution using metering pumps.

To ensure a managed reagent spill containment, the reagent preparation and storage facility will be located within a containment area designed to accommodate 110% of the content of the largest solution tank. In addition, each reagent will be prepared in its own bunded area to constrain spillage and facilitate its return to its respective mixing tank. The storage tanks will be equipped with level indicators and instrumentation to ensure that reagent does not overflow during normal operation. Appropriate ventilation, fire and safety protection, and Material Safety Data Sheet stations will be provided at the facility. Each reagent line and addition point will be labelled in accordance with Workplace Hazardous Materials Information Systems (WHMIS) standards. All operational personnel will receive WHMIS training, along with additional training for the safe handling and use of the reagents, particularly sodium cyanide.

17.3.9 Assay and Metallurgical Laboratory

The assay laboratory will be equipped with the necessary analytical instruments to provide all the routine assays for the mine, concentrator, and environmental department. It will be equipped with the required equipment to prepare samples for analysis, and this will include, among others, crushers, pulverizers, drying ovens, rifflers, balances, equipment tools, and glassware, together with the required safety equipment. The most important analytical instruments required for the laboratory include:

- Fire assay equipment
- Atomic absorption spectrophotometer (AAS)
- X-ray fluorescence spectrometer (XRF)
- Leco furnace.

The metallurgical laboratory will undertake all the necessary testwork to monitor metallurgical performance and, more importantly, to improve process-flowsheet unit operations and efficiencies. The laboratory will be equipped with laboratory crushers, ball and stirred mills, particle-size analysis sieves, flotation cells, filtering devices, balances, and pH meters.

17.4 Water Supply

Two separate water-supply systems will exist, one for fresh water and the other for process water that will be provided to support the operation.

17.4.1 Fresh-Water Supply System

Fresh and potable water will be supplied to a fresh water and fire-water storage tank from a fresh-water source nearby, but yet to be identified. Fresh water will be used primarily for the following:

- Fire water for emergency use
- Cooling water for any motors or systems, if required
- Gland service for the slurry pumps
- Reagent makeup
- Potable water supply including the offices and camp.

The fresh-water and fire-water tank will be equipped with a standpipe to ensure that the tank is always holding at least 40 m^3 of water, equivalent to a two-hour supply of fresh fire water, or as required by the regulations.

The potable water from the fresh-water source will be treated and stored in the potable water-storage tank prior to delivery to various service points.

17.4.2 Process Water Supply System

Some process water generated in the flotation circuit as concentrate thickener overflow solution will be reused in the respective process circuit via the process-water tank in that circuit. Reclaimed water will also be pumped from the tailings thickener overflow to the process-water tank for distribution to the points of usage.

17.5 Air Supply

Separate air service systems will supply air to the following areas:

- Low-pressure air for flotation cells will be provided by air blowers.
- High-pressure air for the concentrate filter press operation will be provided by dedicated air compressors.
- Instrument air will come from the plant air compressors and will be dried and stored in a dedicated air receiver.

17.6 On-Line Sample Analysis

The plant will rely on an on-stream analyzer for process control. The analyzer will analyze the various flotation streams in both flotation circuits. A sufficient number of samples will be taken, probably as shift-samples, so that the circuit can be balanced by analytical results and calculations made as required. Specific samples that will also be taken for metallurgical accounting purposes will be the flotation feed to each circuit, the flotation circuit tailings and final tailings, and the final concentrate samples. These samples will be collected on a shift-basis and will be assayed in the assay laboratory. On-stream particle size monitors will determine the P_{80} particle size of the primary cyclone overflow product feeding the flotation circuit, and the two regrind circuit products.

17.7 Hydrometallurgical Recovery Circuit

17.7.1 Summary Description

The hydrometallurgical process for Scenario B, will treat the pyrite concentrate for recovering cobalt, gold, and silver, and the production of sulphuric acid, which will be used to extract cobalt from the roaster calcine and scandium from flotation tailings. The process is in the early stages of development, is at a low level of maturity, and is regarded as conceptual. Only the recovery of scandium has been demonstrated in the laboratory with scandium in flotation tailings being successfully recovered to a di-scandium trioxide product. However, recovering cobalt, gold, and silver from the roaster calcine will employ established methods such as SX/EW and carbon in pulp (CIP) or carbon in leach (CIL) and is not expected to present any undue difficulty. All stages of the proposed hydrometallurgical process will be tested during the planned metallurgical testwork program. Currently, treatment of tailings to recover scandium is only limited by internally produced acid; higher levels of scandium production would require external supplies of pyrite, sulphur, or bulk acid.

17.7.2 Hydrometallurgical Plant Block Flow Diagram

The Hydrometallurgical Plant Block flow diagram is presented on Figure 17-2.

17.7.3 Concentrate Drying

Pyrite concentrate will be filtered using a conventional plate-and-frame filter press. Filtrate will be recycled as process water. A conveyor will deliver filtered pyrite to the dryer feed bin. Concentrate will be fed to a conventional rotary dryer by either a screw feeder of belt feeder; the dryer will either use waste heat from the roaster or be gas or electrically heated. Dryer off-gas will be cleaned via a baghouse for the recovery of fine pyrite dust before being vented to atmosphere. Dry pyrite concentrate will be fed to the roaster feed bin by conveyor belt, screw feeder, bucket elevator, or pneumatically, depending on the arrangement. Direct-spray drying of pyrite concentrate slurry is also a consideration, as it may be more easily integrated into roaster operations. Dryer options will be further investigated in subsequent trade-off studies.

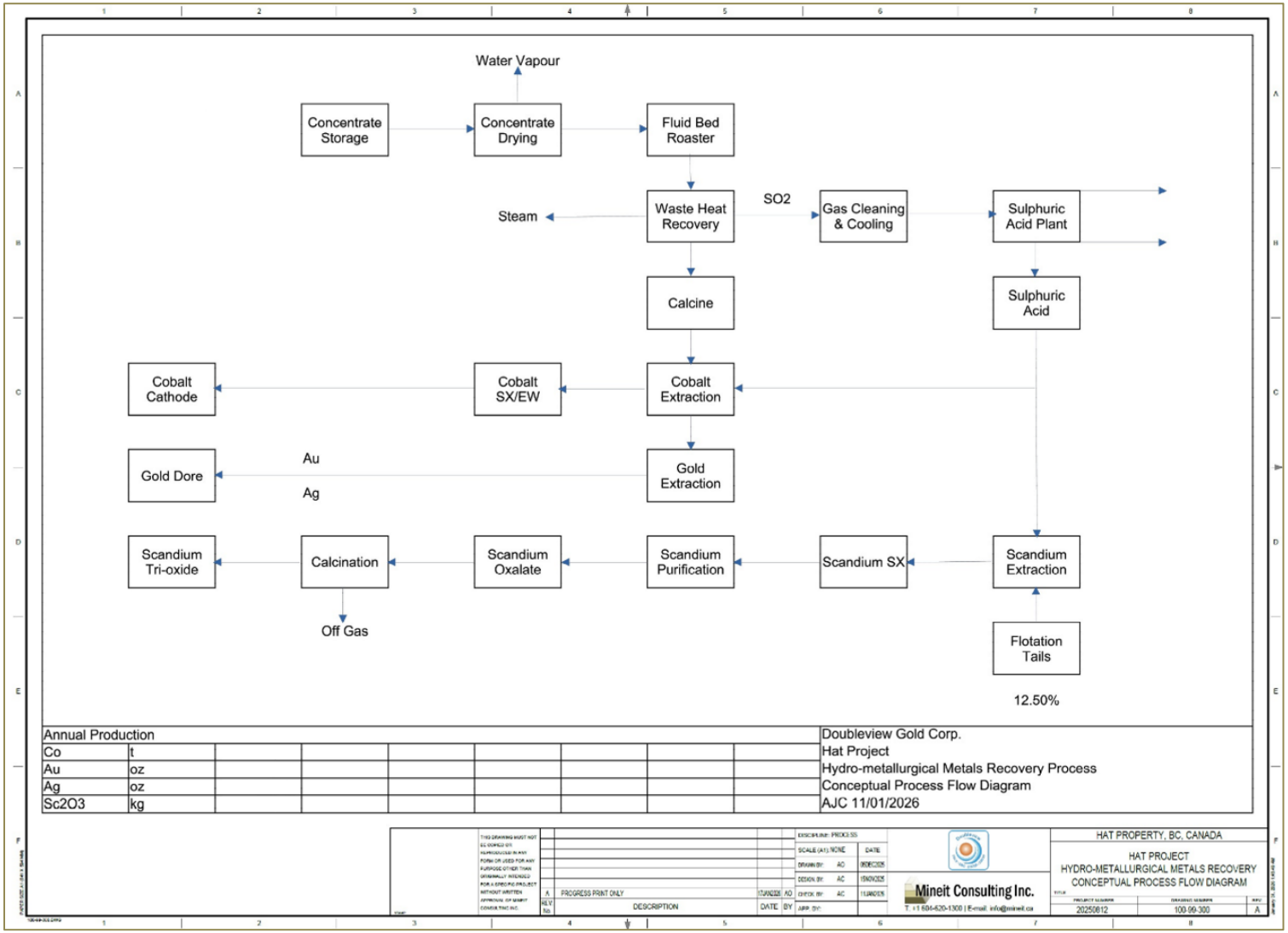


Figure 17-2: Hydrometallurgical Plant Block Flow Diagram

17.7.4 Roasting and Sulphuric Acid Plant Operations.

It is anticipated that the roaster and associated waste-heat recovery, gas cleaning, and calcine cooling plant, as well as the sulphuric acid plant and acid storage, will be procured as a complete package from a suitably qualified vendor. A standard two-stage fluo-solids roaster and conventional double-contact sulphuric acid plants are envisaged. Ideally, roasting will be carried out as a sulphating roast to form cobalt and copper sulphates, with the iron forming the oxide, as is commonly practiced on the Zambian copper-belt at Chambeshi for example. This permits cobalt and any residual copper to be leached in water, with the iron separated as an insoluble residue. Pyrite production will allow for up to 2,000 t/d of acid production. Owing to the project location and carbon footprint considerations, effective waste-heat recovery and energy use will be an important consideration. Up to 129 MW of waste heat will be produced, of which 116 MW will likely be recoverable and used to generate electrical power. This will be more than sufficient to power the entire hydrometallurgical plant; excess power can be used elsewhere in the operation to offset power costs.

17.7.5 Cobalt Solvent Extraction and Electrowinning

Roaster calcine containing cobalt and residual copper as sulphates will be leached in water to extract cobalt and copper. The cobalt will be recovered by conventional solvent extraction (SX) and electrowinning (EW); similarly, copper, but after stripping it will be precipitated as a sulphide to be admixed to the copper concentrate to improve grade. As the scandium recovery process employs oxalate as part of the purification process, consideration can also be given to production of cobalt as cobalt oxalate. Cobalt oxalate undergoes thermal decomposition under controlled conditions to form cobalt metal powder. Oxalate versus EW for cobalt recovery will be evaluated during future testwork and trade-off studies.

17.7.6 Gold and Silver Recovery

Residual precious metals that do not report to the primary copper concentrate will report to the calcine leach residue, which will comprise primarily iron oxides, silica, and silicates. The gold and silver will be leached by conventional cyanidation, recovered by CIL or CIP and produced as a doré.

17.7.7 Scandium Extraction and Recovery

Scandium is a rock-forming mineral and is primarily associated with clinopyroxene, a gangue mineral present in flotation tails. Scandium production is limited by internally generated acid that is sufficient to treat about 12.5% of the tailings produced by primary flotation or produce 180 t/a of di-scandium trioxide (Sc_2O_3).

Scandium is extracted in acidic sulphate media under atmospheric conditions, at low pH, elevated temperature and up to 8 hours using a leach aid that promotes scandium extraction; primary leach extraction is typically 82%. The leach residues are then filtered and washed. The pregnant leach solution then reports to SX where the scandium is extracted in multiple stages using di (2-ethylhexyl) phosphoric acid (DEHPA) and 1-tridecanol in Exxsol D80. Conditions are such that the selectivity of DEHPA for

scandium over iron, titanium, aluminium, and magnesium are maximized. The loaded organic is then scrubbed with a mixture of sulphuric acid and hydrogen peroxide for removing iron, titanium, aluminum, and manganese. After scrubbing, the scandium is stripped from the loaded organic using sodium carbonate, with strip efficiency up to 100%. The scandium is then precipitated using sodium hydroxide, and the precipitate is filtered and washed. The hydroxide precipitate is then redissolved in sulphuric acid, and scandium precipitated as scandium oxalate using oxalic acid, which is filtered, washed, and dried. The dried oxalate is then calcined to produce a high purity di scandium tri-oxide product. Scandium recovery across the recovery circuit is typically 72%.

18 PROJECT INFRASTRUCTURE

18.1 Off-Site Project Infrastructure

18.1.1 Access Road and Bridge

The Hat Project will be accessed by a permanent, all-season industrial road connecting the site to the existing regional transportation network. The route will originate from Telegraph Creek Road via Golden Bear Road and extend approximately 54 km to the mine site.

The road will be designed as a two-lane access road intended for year-round use. It will support transportation of personnel, equipment, fuel, reagents, consumables, and other materials required for both construction and operational phases of the Project. The road will also facilitate the shipment of copper concentrate from the mine site. It will not be used for production haulage from the pit.

Preliminary alignment studies indicate localized grades of up to approximately 13%. The majority of the route will maintain lower sustained gradients. Road design will incorporate appropriate drainage, slope stabilization, avalanche path consideration, and snow clearing measures, and provisions for winter maintenance to ensure safe and reliable year-round operation.

A permanent bridge crossing is proposed at approximately Km 12 over Beatty Creek. The structure will be designed to accommodate year-round traffic associated with mining operations. Final design parameters, including span configuration, foundation conditions, and hydraulic capacity, will be established following detailed hydrological and geotechnical investigations during subsequent engineering phases.

The access route will make use of portions of existing roads combined with newly constructed segments. The majority of the infrastructure is expected to be constructed and maintained by the Project owner.

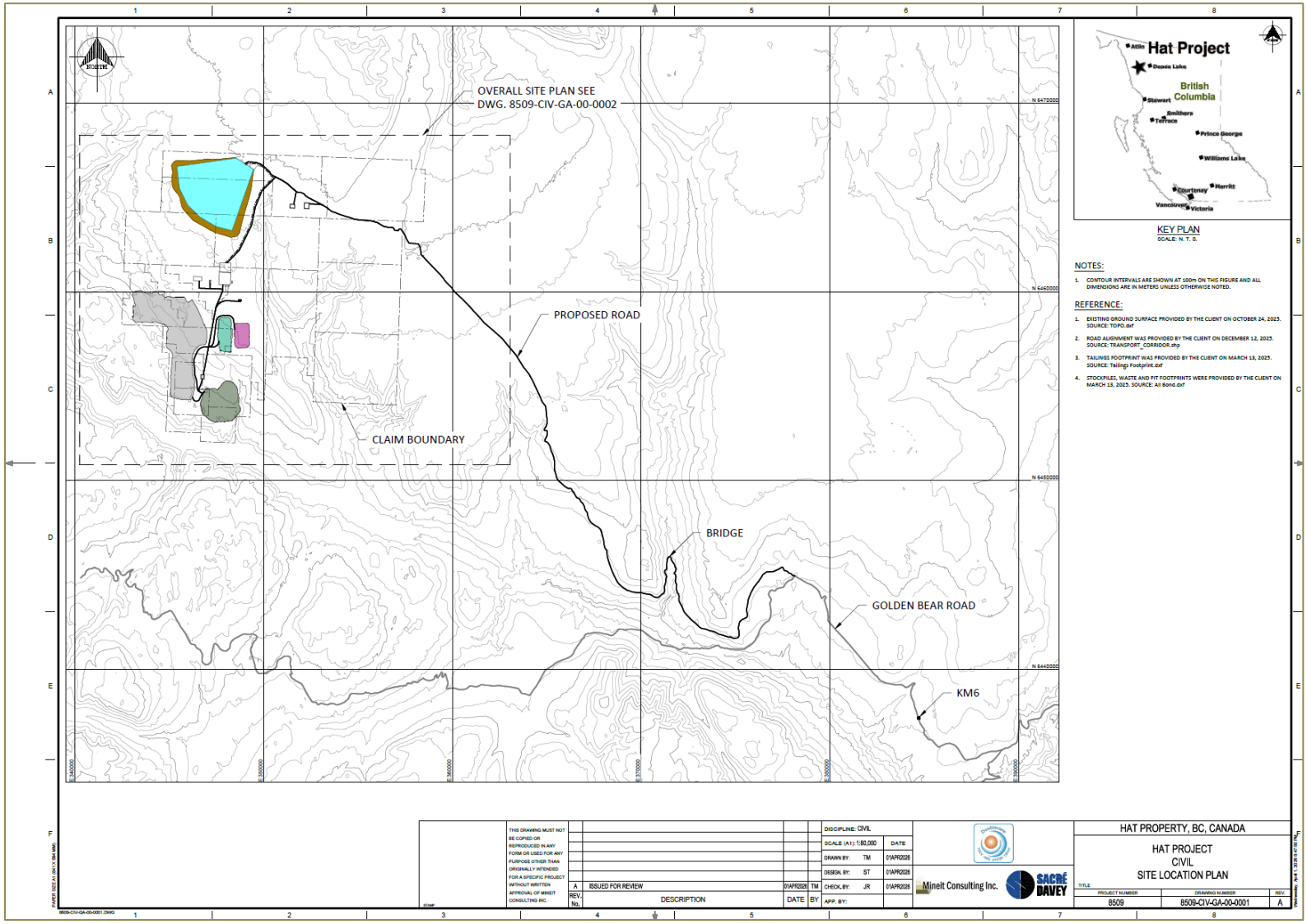


Figure 18-1: Mine Access and Site Layout

18.1.2 Power

Electrical power for the Project is expected to be sourced from the BC Hydro grid system. A preliminary review of supply options indicates that a new transmission connection will be required to link the Project site to the regional transmission network.

The current development concept assumes approximately 100 km of new overhead transmission infrastructure extending from a future point of interconnection with the BC Hydro system to the mine site. The specific interconnection location has not yet been defined and will be determined through future power supply and routing studies conducted in consultation with BC Hydro.

Electrical power delivered to site would be received at a main site substation and distributed throughout the Project to supply the primary crusher, conveying systems, process plant, pumping facilities, camp infrastructure, and other operational installations. Preliminary evaluations indicate that the Project may require approximately 200 MW of peak electrical demand with an estimated continuous operating load of approximately 175 MW.

Emergency diesel generation will be installed on site to maintain operation of critical systems during grid interruptions.

In addition to a Project-specific transmission connection, there is potential for the Hat Project to access electrical power through planned transmission expansion in northwestern British Columbia. BC Hydro has identified the need for additional transmission capacity in the region to support future industrial load growth, including mining developments. Given the Project's location within this region, the current development concept assumes that grid connection could potentially be established through this emerging North Coast Transmission Line. The exact point of interconnection, transmission routing, and available system capacity have not yet been defined and will require further electrical and transmission studies in consultation with BC Hydro. At the PEA stage, this potential connection provides optionality for future grid access but remains subject to confirmation through subsequent engineering, system planning, and regulatory review.

18.2 On-Site Project Infrastructure

18.2.1 General Site Layout

The site layout has been developed to support a large-scale open pit mining operation with a processing capacity of 120 kt/d.

The layout incorporates the open pit, primary crushing facilities, concentrator plant, hydrometallurgical plant (Scenario B), tailings storage facility, waste rock storage areas, camp facilities, and supporting infrastructure. The arrangement of facilities has been developed to reduce internal material transport distances while maintaining appropriate separation between industrial operations and accommodation areas.

ROM ore from the open pit will be delivered to a primary crushing facility located near the pit margin. Crushed material will then be conveyed to the process plant area where it will be deposited onto a coarse ore stockpile prior to processing.

The central operational area will contain the concentrator plant, stockpile facilities, electrical infrastructure, maintenance shops, warehouses, and supporting operational buildings. Grouping these facilities within a central area allows efficient distribution of utilities and minimizes the overall disturbance footprint.

The waste rock storage facility and tailings storage facility have been positioned to accommodate long-term mine development while maintaining suitable separation from major infrastructure.

The operations camp will be located in a designated area separate from the primary industrial zone but connected through the internal site road network.

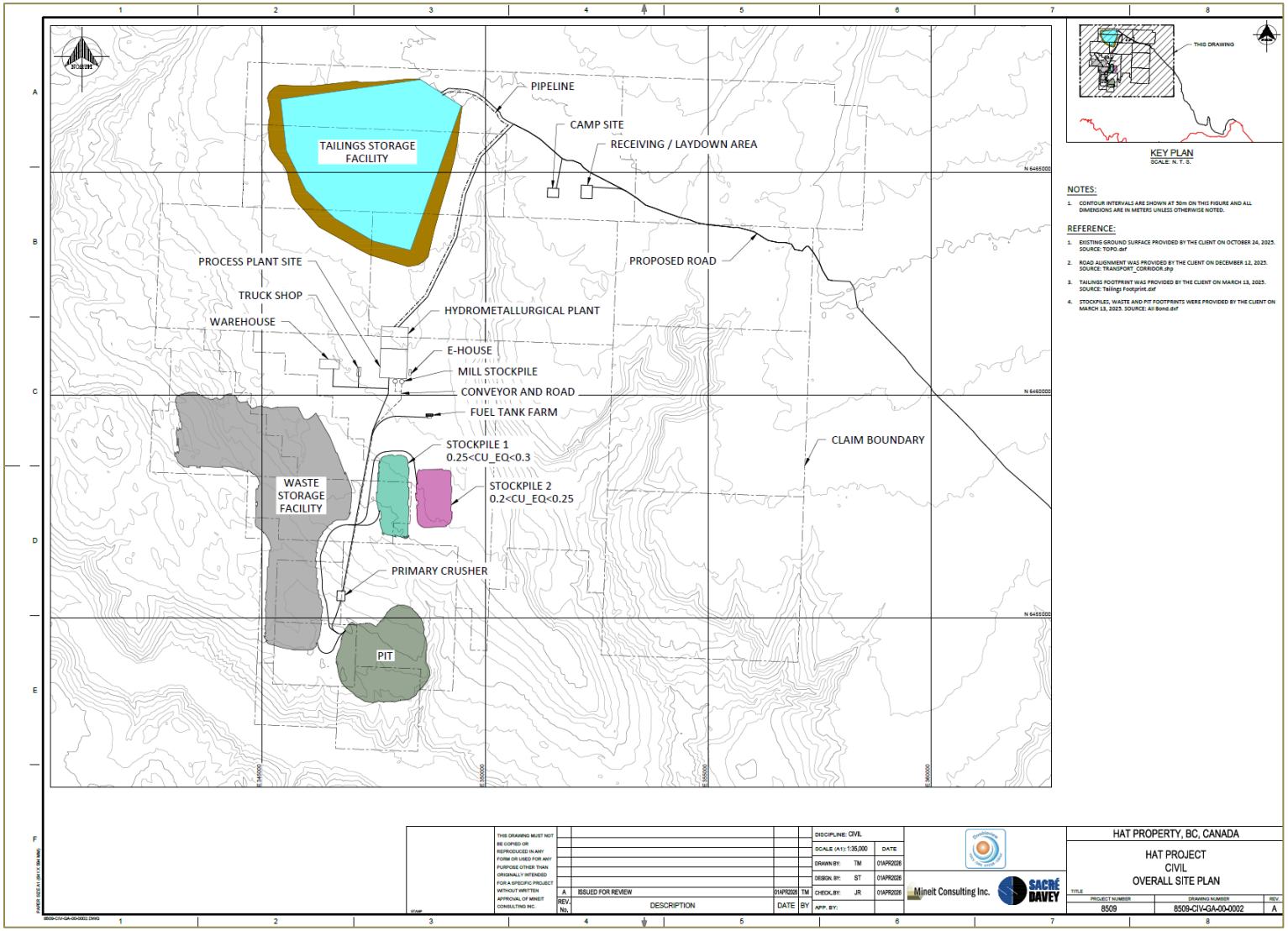


Figure 18-2: Mine Site Layout

18.2.2 Logistics

Project logistics will primarily rely on the all-season access road connecting the mine site to the regional highway network.

Inbound transport will include personnel, construction materials, equipment, reagents, fuel, and consumables required during construction and operations. Major plant and mine equipment will also be transported to site using the regional road network.

Copper and gold concentrate produced at the Project will be transported by truck from the mine site to an established bulk concentrate export facility.

Potential export terminals include the Port of Stewart or other regional ports capable of handling copper concentrate shipments. The final transportation route and export facility will be determined during future studies.

Air transport will play a limited supporting role, primarily for emergency response, light cargo transport, and occasional helicopter flights.

18.2.3 On-Site Power Distribution

Electrical power delivered from the off-site transmission system will be received at the main site substation and distributed throughout the Project area via an internal electrical network.

The system will supply electrical power to the primary crusher, conveyor systems, process plant, pumping stations, camp facilities, workshops, and other supporting infrastructure.

Emergency diesel or LNG generators will be installed to maintain operation of essential systems during power interruptions.

18.2.4 Water Supply

Water required for mining and processing operations will be sourced through a combination of site water collection and reclaimed process water.

Surface water within the Project area will provide initial supply requirements and will be supplemented by reclaimed water recovered from the tailings storage facility. Recycling of process water will be prioritized to minimize fresh-water withdrawals.

The water management approach emphasizes conservation, recycling, and responsible water use. Detailed water balance studies and infrastructure design will be completed during future engineering stages.

18.2.5 Fuel Storage

A central fuel storage facility will be constructed to supply diesel fuel for the mobile mining fleet, auxiliary equipment, site service vehicles, and standby power systems. The storage installation will consist of bulk

diesel storage tanks located within a dedicated tank farm equipped with secondary containment, transfer pumps, and associated fuel handling infrastructure designed in accordance with applicable regulatory requirements and industry practice.

To provide operational reliability and accommodate fuel delivery logistics to the remote site, the facility will be designed to maintain approximately 14 days of diesel storage capacity. On this basis, the total bulk storage capacity for the Project is expected to be on the order of approximately 5 ML. The fuel storage facility will be located within the central site infrastructure area to allow efficient distribution of fuel to the mining fleet and other operational equipment.

During mine closure, all fuel storage tanks, piping systems, pumps, and associated infrastructure will be removed from the site. Remaining fuel inventories will be removed prior to decommissioning, and containment structures will be dismantled. Disturbed areas associated with the fuel storage facility will be reclaimed in accordance with the Project closure and reclamation plan.

18.2.6 Site Buildings

A number of operational and service buildings will be developed to support mining, processing, maintenance, and administrative functions. These facilities will generally be located within the central infrastructure area of the site, in proximity to the process plant and mine service facilities, to provide efficient access to operating areas and utilities.

At the PEA level, the site building infrastructure is expected to include the following principal facilities:

- Administration and operations office
- Mine dry and change house
- Truck shop and mobile equipment maintenance facility
- Warehouse and materials storage building
- Laydown and equipment staging area
- Electrical equipment building (e-house)
- Laboratory and sample preparation facility
- Medical and first aid facility
- Security and site access control building
- Communications and control room facilities
- Utility buildings supporting water and wastewater systems
- Maintenance workshop for auxiliary and service equipment
- Solid waste management and environmental monitoring facilities.

These facilities will support daily site operations, equipment maintenance, and site management functions. Building footprints, construction specifications, and internal layouts will be refined during subsequent stages of engineering as the overall site infrastructure design progresses.

18.2.7 Camp Facilities

A permanent operations camp will be developed to accommodate the workforce required to support mining and processing operations. The facility is planned with capacity for approximately 450 personnel during steady-state operations, with the ability to expand through temporary modular accommodation during peak construction and commissioning periods.

The camp will be located within a designated area of the site that is separated from the primary mining and processing infrastructure while remaining connected through the internal site road network. This arrangement allows appropriate separation from industrial activities while maintaining efficient access to operational areas.

Camp facilities will consist primarily of modular accommodation buildings and associated service infrastructure designed for year-round operation under northern climatic conditions. The complex will include sleeping quarters, dining and kitchen facilities, recreation space, administrative offices, medical and first aid services, laundry facilities, and communications systems required to support a remote operations workforce.

Camp utilities will include potable water supply, wastewater treatment, heating systems appropriate for northern operating conditions, and electrical distribution connected to the site power network. Solid waste generated from camp operations will be managed through the site waste management systems.

Additional modular accommodation may be installed during the construction phase to accommodate the larger temporary workforce. These facilities can be removed or relocated once the Project transitions to normal operations.

At mine closure, camp buildings and associated infrastructure will be decommissioned and removed unless a beneficial post-mining use is identified. Disturbed areas will be reclaimed in accordance with the Project closure and reclamation plan.

18.2.8 Waste Storage Facility

The layout of the waste storage facility (WSF) is described in Section 16.5.

The majority of the waste rock will be stored in the WSF located adjacent to the open pit. This location was selected to minimize haul distances. The facility will be designed with an overall slope angle of 2:5H:1V and will be constructed in 20 m lifts with 37° face angles using a bottom-up construction method. It will have a maximum height of 100 m.

No geochemical studies, including static or kinetic testing, have been conducted to estimate release rates for parameters of concern. Based on exploration data, the deposit mineralization is dominated by chalcopyrite and pyrite, with minor cobaltian pyrite and magnetite. Pyrite is the principal acid-generating mineral, while chalcopyrite represents the main copper sulphide and a potential source of copper leaching. Multi-element assays and mineralogical studies indicate that cobalt is structurally bound within pyrite lattices, and silver occurs within chalcopyrite and pyrite. The Hat deposit also contains elevated

scandium relative to average crustal abundances, representing an additional potential leaching risk from disturbed materials. Further acid-base accounting and kinetic testing are required to delineate the proportions of potentially acid-generating (PAG) and non-acid generating (NAG) material and to determine its suitability for construction, particularly where sulphur content is low.

Reclamation of the WSF will consist of re-sloping to a final overall slope of approximately 2.5H:1V, placing a soil cover, vegetating and long-term seepage collection and treatment. The soil for the cover is assumed to come from the overburden stockpile, and vegetation is assumed to consist of hydroseeding, broadcast seeding or planting of seedlings. As the WSF will be built from the bottom up, this will allow for progressive reclamation of the bottom lifts. It is assumed that all but the top lift of the facility and the access ramp can be reclaimed prior to the active closure period.

Progressive reclamation of the WSF is assumed to occur in the last six years of mine operations. Active closure will occur over five years directly following the end of mining, including reclamation of the topmost lift and access road. Post-closure monitoring, maintenance and water treatment costs were then estimated for an additional 95 years.

Future work should evaluate waste management practices and remediation options that eliminate the need for indefinite water treatment.

18.2.9 Tailings Storage Facility

This section presents the conceptual design for the tailings storage facility (TSF).

Alternatives Assessment

An alternatives assessment was conducted to determine the preferred tailings siting and management technology, evaluating several TSF locations for conventional slurry, filtered tailings, and cycloned sand options. The preferred option is the cycloned sand tailings facility which is situated on a gently sloping plateau between Hackett River and Little Tahltan River valley approximately 10 km north of the proposed open pit. The site was selected because it performed the best overall when considering the critical factors as a whole. These factors include fish habitat impact, embankment height and fill volume, and cost.

Key Design Criteria

Storage requirements:

- The mine life is expected to be 25 years with a total ore production of approximately 1.026 Mt and a total waste rock production of approximately 1.644 Mt of waste rock.

Material properties:

- The dry density of the consolidated impounded tailings was estimated to be 1.5 t/m^3 based on Mineit's estimate of a rock density of 2.9 g/cm^3, and an estimated void ratio of 0.85 for the tailings.
- The dry density of compacted cyclone sand is estimated to be 1.5 t/m^3.
- The dry density of the placed waste rock is estimated to be 2.0 t/m^3.

Risk classification:

- Based on the Project parameters and the regulatory framework in British Columbia, this facility aligns with a 'High' to 'Extreme' consequence classification. This assessment is consistent with the Canadian Dam Association (CDA) guidelines, the B.C. Ministry of Energy, Mines and Low Carbon Innovation requirements, and the Global Industry Standard on Tailings Management. Consequently, the TSF will be engineered to withstand both the maximum credible earthquake and the probable maximum flood.

Geochemistry:

- The mine processing plan will be processing the pyrite, which may result in non-acid generating (NAG) tailings. The acid-rock drainage (ARD) potential for the tailings needs to be confirmed through laboratory testing.

Seismicity:

- Seismicity level: the Project is located in the Northern Cordilleran Volcanic Province, which is a seismically active zone. However, regional seismicity is primarily associated with large major structures like the Denali and Duke River Fault zones rather than localized volcanic activity.
- The seismic hazard is defined by the 6th Generation Seismic Hazard Model of Canada (CanadaSHM6), which forms the basis for the 2020 National Building Code of Canada (NBC 2020). NBC 2020 Seismic Hazard Values (Dease Lake Region)
- For a probability of 2% in 50 years (1 in 2,475-year return period), typical hazard values for a standard reference site (Site Class C—soft rock/very dense soil) in this region are:
  - Peak ground velocity: approximately 0.15 m/s to 0.25 m/s.
  - Peak ground acceleration: approximately 0.14 g to 0.18 g.

Climate:

- As no meteorological stations were located at the proposed mine site, an analysis was done using nearby Environment Canada stations. Included in the analysis were three stations from Telegraph Creek and two from Dease Lake. A summary of the processed precipitation data is presented in Figure 18-3.
- The inverse distance elevation weighting method is a specialized spatial interpolation technique employed by the Hat Project to estimate precipitation in regions with significant topographic relief. Applying this approach, the Hat Project's estimated annual precipitation is 384 mm.
- The annual evaporation was estimated using data from the Water Surface Evaporation over Canada's Landmass open government portal, yielding an estimated annual evaporation of 400 mm.

Table 18-1: Regional Environment Canada Stations

Name	Station Number	Network	Latitude	Longitude	Date Range	Elevation	Distance (south)
Telegraph Creek	42860	Environment Canada	58	−131.167	1942–1979	169	45
Telegraph Creek	38750	Environment Canada	58	−131.333	1979–2000	250	45
Telegraph Creek	16390	BC FLNRORD - Wild Fire Management Branch	58	−131.197	1988–2025	349	45
Dease Lake	14226	Environment Canada	58	−130.011	1944–2011	805	92
Dease Lake AUT	16026	Environment Canada	58	−130.026	1993–2025	804	92

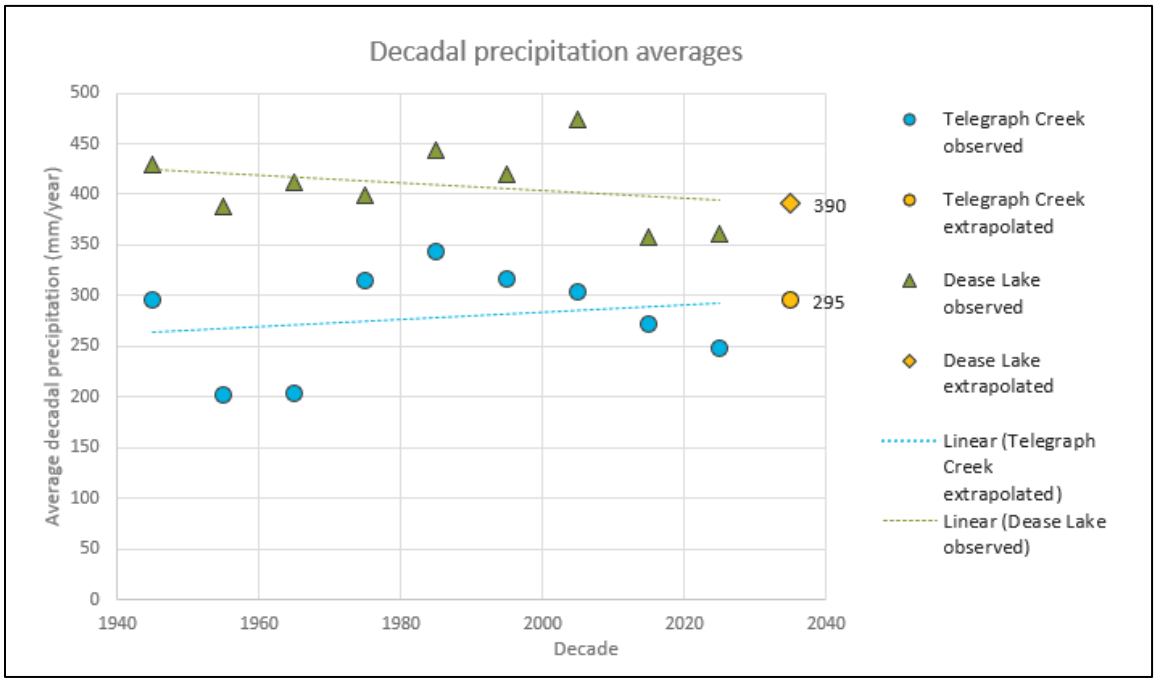


Figure 18-3: Precipitation Data

Tailings Storage Facility Design

The layout of the TSF is shown in Figure 18-2. It consists of a perimeter dyke primarily constructed from compacted cycloned sand. This material will be sourced from the coarse underflow of tailings processed through an on-site cyclone plant. Based on Mineit's estimate of the particle distribution, an estimated 25% portion of the whole may be used for the cyclone sand dam.

Assuming each annual construction period will be approximately seven months, the total amount of cycloned sand available for the life-of-mine is 150 Mt. Subtracting this from the total amount, the tailings needing to be contained in the TSF becomes 876 Mt.

By integrating the natural topography into the design, the northeast end of the TSF will be keyed directly into natural ground to form a secure perimeter dyke. The design cross-section for the cyclone sand dam

is shown in Figure 18-4. An initial starter dam, constructed from non-acid generating (NAG) waste rock or local borrow material, will store the first year of tailings. Subsequent annual raises will be completed using compacted cycloned sand, with a seven-month construction window dictated by the cold climate. By utilizing the centreline raise method, the dam is designed to be free-draining, which lowers the phreatic surface to facilitate geotechnical stability. This includes a drainage blanket of NAG waste rock or local borrow, protected by a granular filter layer positioned between the blanket and the cycloned sand to prevent internal erosion and piping. Seepage that reports from TSF will be managed by a collection system and pumped to the mill for as reuse water.

The dam's key design specifications include:

- Crest: The dam crest has a width of 20 m to allow for heavy equipment traffic.
- Freeboard: A 5 m freeboard has been assumed between the top of the tailings and the embankment crest. The dam crest will need to be raised annually to maintain this freeboard and flood storage during operations. The final freeboard requirement will be determined in later design stages based on detailed analysis of the design flood and necessary flood storage capacity.
- Beach Length: A minimum 200 m operational beach length will be maintained along the cyclone sand dam to protect against wave action and overtopping.
- Slopes: Downstream overall slopes are a minimum of 4H:1V.
- Drainage Blanket: 3 m thickness plus a transition layer for filter compatibility with the tailings.

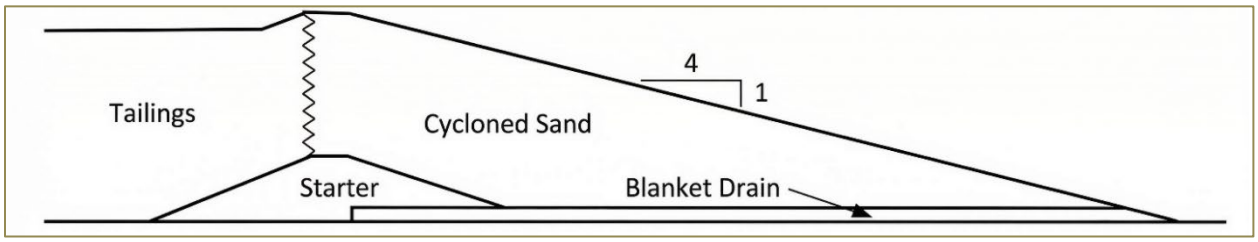


Figure 18-4: Conceptual Cross-Section of the Cycloned Sand Dam

Deposition Strategy and Dam Construction

Tailings will be transported from the process plant via a 7 km-long pipeline. The pipe corridor will be built adjacent to an access road to the TSF. The tailings pipeline will be insulated to facilitate continued operation during the winter months. Tailings spigots will be placed mainly along the dam crest to allow the reclaim pond to develop in the northwest portion of the facility, away from the dam embankment. The strategy for depositing the whole tailings is designed to optimize the consolidation process, facilitate water reclaim, and maintain the required minimum beach length during operations.

The heights of the starter dam and ultimate dam are estimated to be 29 and 94 m, respectively, to meet the design criteria and specifications. Total volumes for the main materials are as follows:

- NAG waste rock or local rockfill for the starter dam: 5.5 Mm^3
- Compacted cyclone sand for the ultimate dam: 100 Mm^3
- NAG waste rock or local rockfill for drainage blanket: 15 Mm^3.

Filter layers are also needed between to prevent piping into the starter dam and the drainage blanket.

Dam Construction with Cycloned Sand

Cycloned sand, the coarse underflow from the cyclone plant, is delivered via pipeline and discharged through a series of spigots along the active crest in a series of cells to build up the downstream shell. Once the sand is deposited in a cell and the excess water has drained, heavy earthmoving equipment—typically low-ground-pressure dozers—spreads the material into uniform, thin lifts. These lifts are then conditioned for moisture and subjected to multiple passes with vibratory smooth-drum or padfoot rollers to achieve the specified target density. This systematic layering and compaction process allows the cyclone sand to achieve the necessary shear strength to remain stable and resistant to liquefaction. Site testing is conducted to assess the density of the compacted material.

Starter Dam

A portion of the waste rock, if available as NAG, approximately 2.8 Mt, will be allocated for the Starter dam of the TSF. Construction using waste rock is done by hauling it from the mining pit and depositing it in horizontal layers, typically up to 1.0 m thick. Each lift is spread by dozers and compacted using heavy vibratory rollers to increase shear strength and minimize post-construction settlement.

Tailings Storage Facility Water Management

The TSF water management system is designed to minimize the facility's water footprint by diverting clean water away and recycling contact water, as well as protecting the facility from extreme storm events.

- Diversion Channel: A diversion channel with a length of 2.5 km will divert non-contact surface runoff away from the TSF to a natural gulley located east of the TSF.
- Emergency Spillway: It will be placed in natural ground to allow for discharge during extreme flood events to prevent the tailings dam from being overtopped during operations. The emergency spillway will need to be raised on an intermittent basis during operations.
- Seepage Recovery: A seepage recovery system will be installed downstream of the tailings facility to collect potential seepage and return to the process plant.
- Water Reclaim System: A water reclaim system will pump the supernatant water via a 3 km-long pipeline from the TSF back to the process plant for reuse.

TSF Closure Plan

The TSF closure plan aims for long-term physical and chemical stability. The tailings dam will remain in place at the end of operation. Prior to reclamation activities, the supernatant pond water will be treated and safely discharged. The tailings will be covered with granular trafficability layer overlaid with a growth

medium and seedling planting. Currently, no other cover material or liner is assumed for the final cover. The emergency spillway will be converted to a closure spillway.

18.3 Site Water Management

Given the elevated metal content associated with the source rock, the possibility that acidification could occur, and the relatively low effluent concentration limits anticipated, it has been assumed that treatment of contact water will be necessary.

The overall site water management strategy is for minimizing environmental impact by separating non-contact (clean) water from contact (impacted) water, collecting all contact sources, and treating surplus water prior to discharge.

As site-specific hydrological and geotechnical data are currently unavailable, several design parameters for the water management design were developed using the QP's engineering judgment. Consequently, the limited understanding of the foundation and climate conditions poses a design risk that mandates comprehensive meteorological and hydrological data collection, as well as geotechnical and site investigations during future design phases.

18.3.1 Contact Water Ponds

Contact water ponds (CWP) are strategically placed at downslope locations to capture runoff and seepage from the waste rock storage areas. Key design features include an HDPE liner to minimize seepage of contact water, and an emergency spillway to discharge excess water during extreme events. Throughout the year, the CWPs will supply make-up water for the plant site operations. However, excess water generated during positive water balance months will require treatment prior to discharge.

18.3.2 Contact Water Collection Ditches

The waste rock areas include a perimeter ditch that collects both seepage and contact surface water and directs them to the CWPs.

18.3.3 Pit Dewatering

Pit water will consist of contributions from direct precipitation, surface runoff, and groundwater seepage. The water is collected in sumps within the pit and pumped to one of the CWPs via a pipeline, which then will be pumped to the process plant as make-up water.

18.3.4 Groundwater Interception Wells

In the absence of hydrogeological field investigations, groundwater interception wells are included downgradient of the open pit and waste rock storage areas to intercept any impacted seepage and pump it to the CWPs. From there, it will be pumped to the process plant as make-up water.

18.3.5 Water Treatment

The water management strategy prioritizes the reuse of site-impacted water, directing TSF water, contact water from the waste rock storage areas, and open pit dewatering to the process plant for use as make-up water. Any surplus process water from the TSF that is not reclaimed for processing will require treatment prior to release.

18.3.6 Closure

Following the 25-year mine life, the open pit will be decommissioned and allowed to naturally flood to create a pit lake. This saturation strategy is intended to submerge exposed sulphide minerals, thereby limiting oxygen exposure and slowing the rate of ARD. During the filling period, all contact water within the pit will continue to be collected and directed to the treatment system to ensure any surplus meets environmental standards before release. Once the water level reaches a stable equilibrium and quality is verified, the pit will be allowed to decant at a designated discharge point, transitioning into a long-term passive management phase.

For closure, long-term stability and water quality are paramount. Water treatment is currently assumed to continue long-term (95 years) into the post-closure period.

The possibility of constructing low-permeability covers over the waste rock storage and TSF areas has been identified as a potential opportunity to significantly reduce or eliminate the need for ongoing, long-term water treatment.

19 MARKET STUDIES AND CONTRACTS

19.1 Products and Marketing Overview

The Hat Project is expected to generate different saleable products depending on the processing scenario evaluated in the PEA.

Under Scenarios A1 and A2, the Project is assumed to produce a marketable copper concentrate containing payable copper, gold, silver, and cobalt, consistent with the commercial assumptions presented in Section 22.

Under Scenario B, the Project is assumed to produce on-site saleable products comprising gold-silver doré, cobalt cathode, and scandium oxide (Sc_2O_3). In this scenario, sulphuric acid generated within the roasting circuit is assumed to be consumed internally as a reagent in the downstream scandium recovery process rather than sold as a separate product.

Copper concentrate is expected to be marketed to third-party smelters and refiners, either directly or through concentrate traders or marketers. The cobalt and scandium products contemplated under Scenario B would require product qualification, confirmation of specifications and impurity limits, and identification of acceptable commercial sales pathways.

As of the effective date of this Technical Report, the Company has no marketing agency agreements, offtake agreements, or formal sales arrangements in place for any Project product. These product and marketing assumptions are preliminary and are intended solely to support the PEA-level economic analysis.

The scandium product assumptions applied in Scenario B are subject to a higher degree of uncertainty than those associated with the copper concentrate and cobalt product streams, reflecting the comparatively early stage of product qualification, the limited scale of test work completed to date, and the additional requirements for confirming product specifications, impurity limits, and commercial acceptance. Notwithstanding this elevated uncertainty, the scandium-related assumptions adopted for this PEA are informed by the metallurgical test work summarized in Section 13 and are considered reasonable for a PEA-level study. These assumptions should be regarded as preliminary and may be refined through further metallurgical testing, product qualification, and commercial validation in future study stages.

19.2 Market Studies, Pricing Basis, and Commercial Assumptions

No project-specific market studies, product valuation studies, or binding offtake assessments have been completed for the Hat Project. Commercial assumptions used in the PEA are based on industry-normal terms and publicly available information and are considered appropriate for a PEA.

Commodity prices applied in the economic analysis are presented in Section 22.3.1 and include assumptions for copper, gold, silver, cobalt, and scandium oxide. These prices are expressed in US dollars, with financial results reported in Canadian dollars using the exchange rate assumption described in Section 22.

Commercial terms applied in the economic analysis are summarized in Section 22.3.2 and include, as applicable:

- Concentrate payables
- Treatment and refining charges
- Transport and selling costs
- Refining charges for payable metals
- Product payability assumptions for on-site saleable products.

For Scenarios A1 and A2, these assumptions relate primarily to the sale of copper concentrate containing payable copper, gold, silver, and cobalt. For Scenario B, the assumptions relate to the sale of gold-silver doré, cobalt cathode, and scandium oxide. All such commercial terms remain conceptual and are subject to future negotiation based on final product specifications, market conditions, logistics arrangements, and counterparties.

19.3 Contracts and Status

As of the effective date of this Technical Report, the Doubleview has no executed contracts that are material to Project development relating to:

- Copper concentrate sales/offtake, marketing, smelting, or refining
- Gold and silver sales/offtake
- Cobalt product sales/offtake
- Scandium product sales/offtake
- Transportation, port handling, shipping
- Sales hedging or forward sales.

Future commercial arrangements are expected to be advanced in subsequent study stages as product specifications are further defined, metallurgical flowsheets are optimized, and marketing discussions with potential counterparties progress.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Baseline Studies

20.1.1 Water and Fish Species

The Project area includes two drainage systems. Hackett River drains from the south and western portion of the property into the Sheslay River to the northwest. The Little Tahltan River flows southeast to the Tahltan River. Both river systems ultimately drain though the Alaskan Panhandle into the Pacific Ocean.

From the list of fish species identified in Wildwood (2025), EMM Consulting Pty. completed a review of the conservation status for each using BC Species and Ecosystems Explorer (B.C. Conservation Data Centre. 2025. Available: http://a100.gov.bc.ca/pub/eswp/). Table 20-1 lists fish species known to be present within or near the Project and classified with a status other than "Not at Risk" (or the equivalent as applicable to each agency) by one or more of: the Government of British Columbia Conservation Status Ranks, Government of Canada Species at Risk Act (SARA) status, or the Committee on the Status of Endangered Species in Canada (COSEWIC).

Table 20-1: Fish Species of Concern in Project Area

English Name (Scientific Name)	B.C. Conservation Status Rank	Federal SARA Status	COSEWIC Classification
Cutthroat trout (*Oncorhynchus clarkii clarkii*)	Blue (Special Concern)	-	-
Eulachon *(Thaleichthys pacificus)*	Blue (Special Concern)	-	E/T
Green sturgeon (*Acipenser medirostris*)	Blue (Special Concern)	1-SC (Special Concern)	SC (Special Concern)
Longfin smelt (Spirinchus *thaleichthys)*	Blue (Special Concern)	-	-
Giant threespine ttickleback[1] (Gasterosteus sp. 1)	Red (are or candidate for Extirpated, Endangered, or Threatened status)	1-SC (Special Concern)	SC (Special Concern)

Note: [1] Unable to confirm threespine stickleback species or subspecies; species with highest sensitivity and potentially in the Project area was selected.
E/T = endangered or threatened

Current Environmental Baseline Studies

In 2025 a desktop review was completed to delineate existing environmental baseline information for the area in and around the Hat Project claim block (Wildwood Engineering & Environmental Ltd. [Wildwood], 2025). A summary of the available environmental baseline data from this Technical Report is provided below.

Wildlife

The Project area includes the Atlin (6-26) and Stikine (6-22) wildlife management units. The Project is within the Ungulate Winter Range (UWR) for Stone's (thinhorn) sheep (*Ovis dalli stonei*), as well as the range for the Level-Kawdy herd of northern caribou that overlaps the northeastern corner of the claim area.

Wildlife species of management concern potentially present in the Project vicinity are given in Table 20-2.

Table 20-2: Wildlife Species of Management Concern Potentially Present in the Project Area

Taxonomic Class	Species
Mammals	• Caribou • Grizzly bear • Moose • Mountain goat • Stone's sheep
Amphibians	• Columbia spotted frog • Long-toed salamander • Western toad • Wood frog
Birds	• Bank swallow

Climate Zone and Vegetation

The Project lies on the Stikine Plateau and comprises mainly Spruce-Willow-Birch, Boreal Black and White Spruce, and Boreal Altai Fescue Alpine biogeoclimatic zones. There are scattered at-risk ecosystems or species around the Project area, and two classified polygons for secured endangered species or ecosystems fall within the claim boundary.

There are no old-growth management areas in the vicinity.

Proposed Environmental Baseline Studies

Additional baseline studies are anticipated for further characterization of the existing biological and physical conditions in and around the Project area, which are summarized in Table 20-3.

Table 20-3: Anticipated Baseline Environmental Studies

Disciplines	Study Information
Meteorology, Climate, and Air Quality	• Install a meteorological station to collect on-site meteorological baseline data selected in consultation with B.C. Ministry of Environment and Parks (ENV). • Air-quality baseline monitoring program for key air-quality parameters, selected in consultation with ENV. • Dispersion modelling.
Surface Water Quality and Hydrology	• Surface water-quality sampling at four locations selected in consultation with ENV. • Surface hydrology baseline measurements at four locations selected in consultation with ENV.
Hydrogeology	• Monitoring groundwater quality and quantity to support development of a conceptual site model.
Aquatic Life and Habitat	• Biophysical survey program for aquatic species to document presence, distribution, habitat availability, and food supply. Study design to be informed by known species presence, feedback from regulatory agencies, and through consultation with participating First Nations to identify valued species or areas important to local communities.
Soil and Vegetation	• Vegetation and ecosystem baseline surveys of the disturbance footprint of the Project.
Wildlife and wildlife Habitat	• Biophysical wildlife and habitat surveys of the disturbance footprint of the Project, potentially impacted habitat, and in consultation with participating First Nations to identify valued species of local communities.
Geochemistry	• Geotechnical site investigations will obtain material samples within the disturbance footprint to undergo geochemical characterization including material composition, and both static and kinetic testing for ML/ARD potential • Materials are expected to include waste rock, ore, and overburden within the disturbance footprint. Tailings will also be characterized based on creation of a representative sample from the planned processing.

20.2 Social and Community Impact

20.2.1 Land Use and Cultural Heritage

The Hat property is within the Traditional Territories of the Tahltan Nation and the Taku River Tlingit First Nation. The closest communities are Dease Lake 95 km northeast and Telegraph Creek 40 km south of the Project.

Socioeconomic and cultural baseline studies have not yet been completed for the Hat Project. An AIA was completed (Taiga Heritage Consulting Ltd., 2024); it was accepted by the British Columbia Ministry of Forests—Archaeology Branch. The AIA field investigation was conducted under HCA permit 2022-470 (Taiga Heritage Consulting Ltd., 2024), which identified 4 sites that were registered with the B.C. Archaeological Site Inventory (IbTv-04-07). Excavation of IbTv-04 was completed under 2022-0470, and subsequently capped July 2025 under HCA permit 2025-0139 with the final report submitted January 2026 (Taiga Heritage Consulting Ltd., 2026).

The Bastion Group (2021) completed an earlier investigation of the Hat claims under Permit 2018-219; it was considered in the AIA. As a result of both programs, four precontact archaeological sites were recorded within the Project area and resulted in the following recommendations:

- Avoid identified sites
- Alteration permits and site monitoring as necessary
- Additional assessments if planned developments are sited in certain areas
- Consultation with First Nations for preferred management approaches for sites and features that are not protected by the B.C. *Heritage Conservation Act* of 1996.

20.2.2 First Nations and Community Engagement

The B.C. *Environmental Assessment Act* (2018) and the Government of Canada *Impact Assessment Act* (IAA) of 2019 (as amended) require Doubleview to formalize discussions with participating First Nations through the prescribed consultation procedures in a respectful manner, with the goal of gaining consent for the Project.

Engagement is critical to identifying valued components and important traditional land use areas for local First Nations, which may be discerned as part of a Traditional Knowledge study if required. Doubleview will continue to engage with First Nations to inform baseline study design, Project design and associated mitigation measures, and land and resource management.

Local community and stakeholder engagement will also be conducted to comply with assessment and permitting standards, and to properly consider and mitigate potential concerns local community members may have in relation to Project development.

Land and Resource Use Plans

The Project lies within the Traditional Territories of the Taku River Tlingit First Nation (TRTFN) and Tahltan First Nation (Tahltan).

A shared decision-making framework is in place respecting land use and wildlife management between the Taku River Tlingit First Nation and the B.C. Government (TRTFN & British Columbia, 2011), which contains guidance on management objectives and consultation requirements to be considered by the Project.

The Tahltan are also co-developing a land use plan with the Province of British Columbia (Tahltan Central Government, 2025), which may be enacted into law and is anticipated to provide further guidance on conservation and development objectives relevant to Project development.

Trapline Areas

The Project overlaps three trapline tenures (TR0622T017, TR0626T001, TR0626T003), the owners of which will be formally engaged with the goal of developing mutual agreements or letters of understanding, and the tenures will be considered in terms of socioeconomic or traditional use effects where applicable.

20.2.3 Health, Social, and Economic Effects

A human health and socioeconomic study will likely be required to forecast and describe community health and socioeconomic changes that may take place because of Project development. Should this be required, identified issues will be considered and addressed through proposed mitigative measures.

20.3 Environmental and Impact Assessment and Permitting

This section summarizes existing authorizations for the Project, as well as potential federal and provincial permits, licenses, and approvals that may be required for future construction and operation of the Project.

20.3.1 Existing Approvals

Doubleview operates the Hat Project under Multi-Year Area-Based BC *Mines Act* of 1996 mineral exploration permit MX-1-872, which is valid until March 31, 2037. The permit authorizes use of the Project area for an existing temporary camp (core tent, office, dry tent, generator shed), geophysical surveys, surface drilling, and an exploration trail.

Pursuant to the conditions of MX-1-872, Doubleview maintains a security with the Minister of Finance for the Project.

20.3.2 Anticipated Approvals

The Hat Project will be authorized under a series of provincial and federal regulatory approvals. Table 20-4 provides a list of potential federal and provincial authorizations, licenses, and permits that may be required to develop the Hat Project.

Table 20-4: Possible Federal and Provincial Authorizations or Permits Required for the Hat Project

Authorization	Legislation	Issuing Agency	Purpose
Federal			
Impact Assessment Act Decision Statement	*Impact Assessment Act*, 2019	IAAC	Decision statement on whether the Hat Project is likely to cause non-negligible adverse effects within federal jurisdiction. It includes conditions, consisting of mitigation measures, and a follow-up program that the proponent must fulfill to proceed with the Project.
Fisheries Act Authorization	*Fisheries Act*, 1985	Fisheries and Oceans Canada	Authorization under Section 35(2) of the Fisheries Act for harmful alteration, disruption, or destruction (HADD) of fish habitat. Authorization under the exceptions described in Section 34.4(2), which includes work authorized under Section 35(2) of the *Fisheries Act.*
Scientific License	*Fisheries Act*, 1985	Fisheries and Oceans Canada	Required to harvest fish for experimental, scientific, educational, or public display purposes.
Factory License	*Explosives Act*, 1985	Natural Resources Canada	Required for on-site explosives manufacturing.

Authorization	Legislation	Issuing Agency	Purpose
Magazine License	*Explosives Act,* 1985	Natural Resources Canada	Required for on-site explosives manufacturing and storage.
MDMER Authorization	*Metal and Diamond Mining Effluent Regulations* (MDMER), 2002	Environmental and Climate Change Canada	May be required to deposit effluent that contains a deleterious substance, as defined by MDMER.
International River Improvements Act Notification	*International River Improvements Act*, 1985	Environmental and Climate Change Canada	A notification that may be required if the Project may have an effect on an international waterway on water levels or volume beyond specified thresholds at the Canadian Border.
Migratory Bird Permit	*Migratory Birds Convention Act*, 1994	Environmental and Climate Change Canada	May be required if the Project is shown to affect nesting habitats used by migratory birds or if activities occur during the nesting season.
Species at Risk Act Permit	*Species at Risk Act*, 2002	Environmental and Climate Change Canada	May be required if the Project has the potential to affect a species listed on Schedule 1 of the Act, including any part of its critical habitat, or the residences of its individuals.
Transportation of Dangerous Goods Act and Regulations Permits	*Transportation of Dangerous Goods Act*, 1992	Transport Canada	Required for the transportation and handling of dangerous goods.
Radio-isotope License	*Nuclear Safety and Control Act*, 1997	Natural Resources Canada	Authorization for nuclear devices.
Radio Licenses	*Radiocommunications Act*, 1985	Industry Canada	License for on-site radio communication.
Provincial			
Environmental Assessment Certificate	*Environmental Assessment Act*, 2018	Environmental Assessment Office (EAO)	Certificate including conditions the proponent must fulfill to proceed with Project.
Mines Act Permit	*Mines Act*, 1996	MCM	Permit to construct, operate, close, decommission, and reclaim a mine.
Explosives Storage and Use Permit	*Mines Act*, 1996	MCM	Approval for storage and use of explosives.
Liquid Effluent Discharge Permit	*Environmental Assessment Act*, 2018	ENV	Authorization to discharge mine-affected effluent to the receiving environment.
Permit to Discharge Waste	*Environmental Assessment Act*, 2018	ENV	Permit to discharge solid waste into an on-site landfill.
Air Emissions Discharge Permit	*Environmental Assessment Act*, 2018	ENV	Permit for air emissions, including sources of dust and emissions from the processing plant.
Hazardous Waste Registration	*Environmental Management Act*, 2003 Hazardous Waste Regulations	ENV	Authorization to temporarily storage hazardous wastes on site.
Registration of Petroleum Storage and Distribution Facilities	*Environmental Management Act*, 2003 Petroleum Storage and Distribution Facilities Storm Water Regulation	ENV	Required registration of fuel storage area.
Pesticide Use Permit	*Integrated Pest Management Act*, 2003	ENV	Required to control noxious weed and invasive plants in disturbed areas within the Project footprint.

Authorization	Legislation	Issuing Agency	Purpose
Pesticide User License	*Integrated Pest Management Act*, 2003	ENV	Required for control of noxious weeds.
Record of Sewerage System and Sewerage System Letter of Certification	*Public Health Act*, 2008 Sewerage System Regulation	Interior Health Authority	Authorization for septic tank and disposal field.
Construction Permits for Water Supply Systems	*Drinking Water Protection Act*, 2001	Regional Health Authority	Permits for potable water wells, water system construction, and water system operations.
Occupant License to Cut – Mine Site and Special Use Permit	*Forest Act*, 1996	Ministry of Forests	Licenses to harvest timber for site clearing and for use of Crown Land within provincial forest.
Occupant License to Cut	*Forest Act*, 1996	Ministry of Forests	License for vegetation clearing on Crown land.
Road Use Permit	*Forest Act*, 1996	Ministry of Forests	Approval for use of forest service roads.
Industrial Access Permit	*Industrial Roads Act*, 1996	Ministry of Forests	Required to make improvements to access roads on Crown land.
Highway Use Permit – Resources and Industrial Access	*Transportation Act*, 2004	Ministry of Transportation and Transit	Permit for new roads joining onto public road.
License of Occupation and Statutory Right of Way	*Land Act*, 1996	Ministry of Water, Land, and Resource Stewardship (WLRS)	Authorization to occupy Crown land for power line right-of-way.
Mining Right of Way Permit	*Mining Right of Way Act*, 1996	WLRS	Required for right-of-way access within Crown or private lands.
Change Approval for Works in and About a Stream	*Water Sustainability Act*, 2014	WLRS	Approval for complex changes in and around a stream.
Water Use Approval, Water Use License	*Water Sustainability Act*, 2014	WLRS	License to divert, store, and use water and make changes in or around a stream.
Wildlife Permit	*Wildlife Act*, 1996	WLRS	Permit for wildlife salvages and bird nest removal or relocations.
Fish Collection Permit	*Wildlife Act*, 1996	WLRS	Authorization to capture or collect fish.
Heritage Conservation Act Concurrence Letter	*Heritage Conservation Act*, 1996	Ministry of Tourism, Arts, Culture, and Sport	Letter stating that archaeological assessment is complete.
Section 14 Heritage Inspection Permit	*Heritage Conservation Act*, 1996	Ministry of Tourism, Arts, Culture, and Sport	Permit identifying areas of archaeological potential within the Project footprint.
Section 12 Site Alteration Permit	*Heritage Conservation Act*, 1996	Ministry of Tourism, Arts, Culture, and Sport	Permit for alteration of an archaeological site.
Permit to Connect a Powerline	*Safety Standards Act*, 2003	BC Hydro	Required for connection of private powerline to the BC Hydro grid.

20.3.3 Major Authorizations

Federal Impact Assessment and Provincial Environmental Assessment

Project development is expected to proceed through the federal IAA process, based on conceptual design and estimated production volumes exceeding the thresholds within the Canadian Physical Activities Regulations. The Impact Assessment Agency of Canada (IAAC) will review the initial project description and determine whether to designate the Project as reviewable, necessitating an Impact Assessment (IA).

When a project undergoes an Impact Assessment, the B.C. and Canadian governments have a cooperation agreement (Canada & British Columbia, 2019) that allows either:

- A coordinated process that harmonizes the timelines and requirements of the assessments, but will result in independent decisions by provincial and federal regulators on issuance of a provincial Environmental Assessment Certificate and Decision Statement, respectively.
- A substituted process that delegates authority of the assessment to the provincial EAO.

Alternatively, if the IAA does not designate the Project as reviewable, a provincial Environmental Assessment will be completed, since conceptual designs will also exceed provincial thresholds that necessitate acquiring an Environmental Assessment Certificate under the *Environmental Assessment Act*.

Upon completion of the impact assessment(s), the Project will be approved federally by issuance of a Decision Statement or provincially by issuance of an Environmental Assessment Certificate, as applicable.

Joint Mines Act and Environmental Management Act Application

Doubleview has not yet filed the necessary provincial permit applications with the B.C. Government, which will take place following completion of the impact assessment process (Section 20.4.3i). Doubleview will complete a Joint *Mines Act* and *Environment Management Act* Application (JA) under the coordinated process administered by the B.C. Major Mines Office (MMO) (British Columbia, 2023), and apply for other applicable provincial permits outside of the JA process (Table 20-4).

Provincial permits under the *Mines Act*, *Environmental Management Act*, or other applicable acts and regulations will contain the conditions for construction, operation, monitoring, and closure of the Project based on submitted Project designs, plans, or other commitments made during permitting.

Fisheries Act and other Federal Authorizations

Designated projects undergoing a federal impact assessment under IAA may be subject to other federal authorization frameworks (Table 20-4), which can be completed concurrently with the federal process to consolidate information, studies, and consultation requirements where possible. Should the Project not be designated under IAA, applicable federal authorizations will nonetheless be obtained in alignment with their individual frameworks.

Fisheries Act of 1985 authorizations from the Department of Fisheries and Ocean Canada (DFO) are anticipated under Section 34, works that contribute a deleterious material to fish or fish habitat, and Section 35, works that may cause the harmful alteration, disruption, or destruction of fish habitat. As the Project is a metal mine, it will be subject to the Metal and Diamond Mines Effluent Regulations (MDMER), which, under authority of the *Fisheries* Act, regulates the deposition of deleterious substances by metal mines in Canada.

21 CAPITAL AND OPERATING COSTS

The capital cost estimate has been prepared at an Association for the Advancement of Cost Engineering (AACE) Class 5 level, consistent with the preliminary nature of the PEA. The estimate was developed using a combination of preliminary engineering, process design criteria, equipment sizing, factored cost development, benchmark comparisons, and available vendor, consultant, and industry-reference information. As such, the estimate is conceptual in nature and is intended to support an early-stage economic assessment of the Project rather than detailed budgeting or execution planning.

All values presented in this section are reported in Canadian dollars, unless otherwise stated. Totals may not sum due to rounding.

21.1 Capital Cost

The PEA evaluates two processing scenarios: (A) a conventional Cu–Au–Ag–Co flotation concentrator at 120,000 t/d (42 Mt/a) with two recovery cases—A1 based on Sepro Laboratories' metallurgical testwork, and A2 reflecting target and expected performance; and (B) a full circuit that retains the base flowsheet and adds a downstream hydrometallurgical scandium-recovery circuit (described in Section 17). These scenarios will be presented on the respective tables where significant differences were evaluated.

21.1.1 Capital Cost Estimate Summary

Table 21-1 summarizes initial and total capital cost (CAPEX) for Scenarios A and B. Totals may not sum due to rounding. The breakdown of each component will be presented in the following sections.

Table 21-1: Initial Capital Cost Summary ($ millions)

Capital Cost Summary	Scenario A1	Scenario A2	Scenario B
Initial CAPEX			
Processing Plant (Excluding Hydrometallurgical Plant)	1,609	1,645	1,810
Mining CAPEX	394	394	394
Mining Pre-Stripping	97	97	97
Infrastructure (Power, Water, Roads, Camp)[1]	326	326	326
Tailings and Water Management	157	157	157
Indirects + Engineering, Procurement, and Construction Management	258	262	278
Contingency (25%)	710	720	766
Total Initial CAPEX	**3,552**	**3,601**	**3,828**
Sustaining CAPEX			
Processing Plant (Including Hydrometallurgical Plant)	285	285	1,194
Mining CAPEX	811	811	811
Infrastructure (Power, Water, Roads, Camp)	63	63	63
Tailings and Water Management	1,065	1,065	1,065

Capital Cost Summary	Scenario A1	Scenario A2	Scenario B
Indirects + Engineering, Procurement, & Construction Management	142	142	233
Contingency (25%)	390	390	640
Total Sustaining CAPEX	**2,755**	**2,755**	**4,006**
Closure and Reclamation	503	503	503

Notes: [1] Capital cost estimate Infrastructure includes the required power infrastructure include the extension of the transmission line (~150 km), switching stations and mine main substations (~$140 million).
Totals may not sum due to rounding.

Total initial CAPEX is estimated at $3,552 million for A1, $3,601 million (A2), and $3,828 million (B), reflecting the higher processing plant scope and associated indirect costs and contingencies in Scenario B. Total sustaining CAPEX is estimated at $2,755 million for A1 or A2, and $4,006 million for B, with the increase in B driven primarily by the inclusion of the hydrometallurgical plant and scandium recovery circuit within sustaining capital, while mining, infrastructure, and tailings sustaining components remain broadly consistent across scenarios.

21.1.2 Mining Capital Costs Estimate

Mining CAPEX includes mobile equipment, support equipment, and mine facilities required for operations. Cost related to site preparation and pre-stripping during the pre-production phase have been capitalized. Primary and ancillary equipment had been listed in Table 21-2.

Table 21-2: Mining Capital Cost ($ millions)

Mining Capital	Initial	Sustaining	Total
Primary Equipment Capital	348	258	606
Ancillary Equipment Capital	46	31	77
Equipment Direct Capital Cost	394	289	683
Pre-Stripping	97	0	97
Indirect + EPCM (10%)	49	29	78
Contingency (25%)	135	80	215
Spares, Maintenance and Life cycle cost	0	523	523
Total	**675**	**921**	**1,596**

Notes: The pricing basis for the mining fleet capital estimate, including purchase cost and life-cycle cost assumptions, was provided by the equipment suppliers, including Komatsu and SMS Equipment.

Figure 21-1 shows the mining cost schedule based on the mining fleet requirement, including fleet replacement and Life Cycle Cost (indirect and contingency not included). As mentioned in Section 16, the last five years of operation are focused on reprocessing of low-grade stockpiles and mine closure. No additional mining equipment purchase, or replacement is planned during that end period. Leasing and other equipment financing strategies should be assessed in future study phases to reduce initial mining fleet capital requirements.

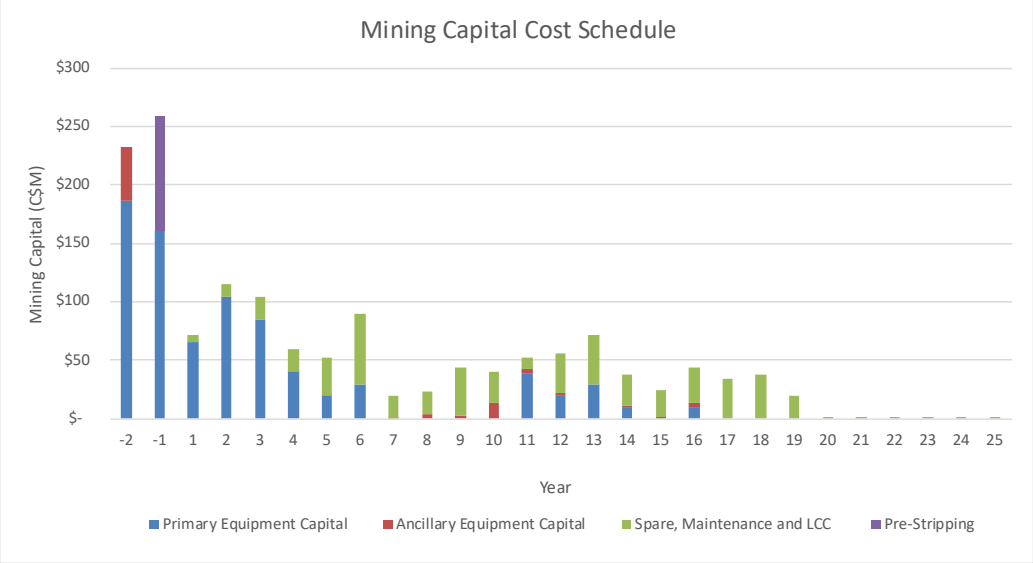


Figure 21-1: Mining Capital Schedule (Year 20–25 Mill Feed from Low-Grade Stockpiles)

21.1.3 Processing Capital Cost Estimate

As mentioned previously, three scenarios were evaluated in Section 17. Table 21-3 summarizes the initial CAPEX associated with each scenario.

Table 21-3: Processing Initial Capital Cost Summary ($ millions)

Processing Capital	Pricing Basis	Scenario A1	Scenario A2	Scenario B
Mechanical Equipment Supply				
Crushing	Database pricing and historical based on equipment List. Includes additional 10% for minor miscellaneous equipment	92.6	92.6	92.6
Stockpile		14.6	14.6	14.6
Grinding Circuit		182.1	182.1	182.1
Copper–Gold Flotation		52.8	60.7	52.8
Cobalt–Pyrite Flotation		0.0	0.0	47.0
Copper Concentrate Dewatering		4.9	5.1	4.9
Cobalt Concentrate Dewatering		0.0	0.0	4.9
Tailings Disposal		17.3	17.3	17.3
General		4.1	4.1	4.1
Reagents		7.6	8.8	7.6
Subtotal		**376.1**	**385.4**	**428.0**
Direct Cost				
Mechanical Install	25% of mechanical equipment supply	94.0	96.4	107.0
Earthworks	20% of equipment supply	41.1	41.1	41.1
Concrete	30% of equipment supply	112.8	115.6	128.4
Steel	20% of equipment supply	75.2	77.1	85.6
Platework	10% of equipment supply	37.6	38.5	42.8

Processing Capital	Pricing Basis	Scenario A1	Scenario A2	Scenario B
Piping	40% of equipment supply	150.5	154.2	171.2
Electrical	35% of equipment supply	131.6	134.9	149.8
Instrumentation	12% of equipment supply	45.1	46.3	51.4
Building and HVAC	Allowance for northern-climate buildings	68.5	68.5	68.5
Subtotal		**1,132.7**	**1,158.0**	**1,273.9**
Indirect Cost				
Construction—Temporary Facilities	10% of direct cost	113.3	115.8	127.4
Freight (Not Heavy Haulage)	20% of equipment	75.2	77.1	85.6
Vendor Representatives	3% of equipment	11.3	11.6	12.8
Spares	5%[1] of equipment	9.4	9.6	10.7
First Fills	Allowance	6.9	6.9	6.9
EPCM services	20%[1] of direct cost	158.6	162.1	178.3
Vendor Construction Supervision	8% of direct cost	90.6	92.6	101.9
Commissioning Cost	3% of equipment	11.3	11.6	12.8
Subtotal Indirect Cost		**476.5**	**487.3**	**536.5**
Subtotal		**1,609.2**	**1,645.2**	**1,810.4**
Indirect Cost (10%)	Allowance for additional Costs	160.9	164.5	181.0
Contingency (25%)[2]		442.5	452.4	497.8
Total Processing Capital Cost Estimate		**2,212.6**	**2,262.2**	**2,489.2**

Notes: [1] Due to the two grinding lines using the same equipment, savings on spares and engineering, procurement, and construction management are considered—reducing them to 2.5% and 14% respectively.
[2] Contingency of 25% applied across the Project to all direct and indirect costs.
Process capital cost prepared by Sacré-Davey using process equipment selection and design inputs provided by Mineit's in-house process team.

Processing sustaining CAPEX for Scenario A1 and A2 is estimated at $285 million over the LOM. For Scenario B the sustaining CAPEX increases to $1,194 million, since it includes constructing the hydrometallurgical and scandium-recovery processing circuit, starting at Year 3 Table 21-4 and Table 21-5 show the total CAPEX of $909 million associated with the hydrometallurgical plant.

The total direct Hydrometallurgical plant CAPEX includes:

- Purchased equipment delivered (including fabricated equipment and process machinery)
- Purchased equipment installation
- Instrumentation and controls (installed)
- Piping (installed)
- Electrical (installed)
- Building (including services)
- Site improvements
- Services facilities (installed)
- Permitting.

Table 21-4: Hydrometallurgical Plant Direct Capital Cost Summary ($ millions)

Hydrometallurgical Plant	Capacity	Unit	Estimate CAPEX	Base Date	Escalation	CAPEX
Concentrate Drying	64	t/h pyrite	19	2023	1.07	20.5
Roasting, Waste Heat Recovery, and Acid Plant	2,000	t/d acid	253	2025	1.00	253.5
Cobalt Extraction and Recovery	2,578	t/a cathode	44	2020	1.26	55.4
Gold Extraction and Recovery	46	t/h calcine	21	2025	1.00	20.6
Scandium Extraction and Recovery	180	t/a Sc_2O_3	160	2016	1.35	216.4
Total Direct Hydrometallurgical Plant Costs						**566.3**

Table 21-5: Hydrometallurgical Plant Total Capital Cost ($ millions)

Hydrometallurgical Plant Capital Cost	Estimation Factor[1]	CAPEX
Total Direct Hydrometallurgical Plant Costs	Base	566
Indirect Costs—Engineering and Supervision	0.10	54
Indirect Costs—Construction	0.12	67
Contractor Fee	0.06	34
Contingency	0.12	69
Subtotal		**791**
Working Capital as a Factor of Total Capital Investment	0.15	119
Total Capital		**909**

Notes: 1 The Estimation Factor represents the multiplier applied to the relevant base cost to estimate each capital cost item. Indirect costs, contractor fee, and contingency are estimated as a proportion of direct hydrometallurgical plant cost, whereas working capital is estimated as a proportion of total capital investment.

21.1.4 On-Site Infrastructure Capital Cost

On-site infrastructure includes site facilities and services required for construction and operations (camp, power, buildings, site roads, communications, fuel storage, workshops, etc.). Table 21-6 summarizes the infrastructure CAPEX estimate. Sustaining capital for some components was considered not to be significant at this level of estimation, and is assumed to be accounted for with the indirect and contingency allowances across the Project CAPEX.

Table 21-6: On-Site Infrastructure Initial Capital Cost ($ millions)

On-Site Infrastructure	Pricing Basis	Initial CAPEX	Sustaining CAPEX
Road and Bridge	Estimated based on first principles	44	Included on indirect and contingency
Camp Facility During Construction and Permanent	Based on $100,000 per permanent staff	48	Included on indirect and contingency
Power Transmission Line Expansion	Order of Magnitude estimate	143	63
Auxiliary and Support Buildings	Allowance	92	Included on indirect and contingency
Subtotal		**326**	**63**
Indirect Cost (10%)	Allowance	33	6
Contingency (25%)	Allowance	90	17
Total		**448**	**87**

A preliminary capital estimate of approximately $140 million has been developed for the infrastructure required to deliver grid power to the Project. The estimate has been prepared at a high level and includes the major components expected to be required for grid connection, namely the transmission line, switching station, main mine substation, and associated allowances for environmental and permitting activities, legal requirements, and supporting technical studies. This estimate is conceptual in nature and subject to refinement as Project definition advances.

21.1.5 Tailings Storage Facility and Water Management Capital Cost

CAPEX for tailings storage facilities and associated pipelines, reclaim systems, and starter embankments is shown in Table 21-7.

Table 21-7: Tailings Storage Facility Direct Capital Cost ($ millions)

Tailings Storage Facility	Initial CAPEX	Sustaining CAPEX	Total
Tailings Dam	71.2	857.5	928.8
Tailings Dual Pipeline	13.0	13.5	26.5
Spigot System	18.5	28.4	46.9
Reclaim Water Line	8.0	13.5	21.5
TSF Spillway	2.0	6.0	8.0
Diversion Pipeline	1.5	1.5	3.0
Seepage Collection System	3.5	3.5	7.0
Total	**117.7**	**923.9**	**1,041.6**

Water Management Capital Cost

Water management CAPEX includes diversion channels, collection ponds, dewatering systems, interception wells, and water treatment plant infrastructure (Table 21-8).

Table 21-8: Water Management Direct Capital Cost ($ millions)

Site Water Management	Initial CAPEX	Sustaining CAPEX	Total
Contact-Water Pond	10.0	10.0	20.0
Collection Channels for Contact Water	6.0	7.3	13.3
Pit Dewatering	15.0	20.0	35.0
Groundwater Interception Wells	8.0	8.0	16.0
Water Treatment Plant	-	96.0	96.0
Total	**39.0**	**141.3**	**180.3**

Table 21-9 includes the estimated closure and post-closure cost associated with the TSF and contact-water treatment.

Table 21-9: Total Capital Cost of TSF and Water Management, Including TSF Closure and Post-Closure ($ millions)

Capital Cost	Initial CAPEX	Sustaining CAPEX	Total
Total TSF Direct Cost	117.7	923.9	1,041.6
Total Site Water Management Direct Cost	39.0	141.3	180.3
Indirect (10%)	15.7	106.5	122.2
Contingency (25%)	43.1	292.9	336.0
Total TSF and Site Water Management Cost	**215.5**	**1,464.6**	**1,680.1**
Closure	-	303.8	303.8
Post-Closure	-	199.5	199.5

21.2 Operating Costs

21.2.1 Mining Operating Cost Estimate

Based on the final fleet selection described in Section 16, mining OPEX (OPEX) were estimated using supplier-provided hourly equipment OPEX, adjusted to reflect Project-specific assumptions for labour, consumables, and supplies. Costs were developed for the fleet required in each operational phase, then scheduled by year to reflect the planned increase in mining activity over the LOM. Table 21-10 and Table 21-11 summarizes the OPEX estimation.

Table 21-10: Mining Fleet Equipment Operating Cost

Equipment OPEX without Labour	Cost ($/h)
Shovels	1,092.2
Drills	410.7
Trucks	562.1
Ancillary (Including Pickup trucks)	27.4

Table 21-11: Mine Operating Cost Estimate for Each Operational Phase

Mine OPEX Estimate	Phase 1	Phase 2	Phase 3	Phase 4
Shovel Fleet	3	3	3	3
Drill Fleet	7	7	7	7
Truck Fleet	26	30	40	41
Ancillary Fleet	69	73	81	78
Fleet OPEX/Year ($M/a)	135.9	150.1	185.1	188.0
Mining Labour Cost ($/t Moved)	0.31	0.31	0.31	0.31
Fleet Cost ($/t Moved)	0.89	1.09	1.24	1.52
Blasting ($/t Moved)[1]	0.52	0.52	0.52	0.52
Indirect %10 ($/t Moved)	0.17	0.19	0.21	0.23
Total Cost ($/t Moved)	**1.89**	**2.11**	**2.28**	**2.58**

Notes: [1] Conservative estimate based on similar Canadian operations.
The phased increase in mining fleet requirements reflects progressive pit deepening and the corresponding increase in haulage distances as mining advances.

Mine Operations Labour

Table 21-12 presents the proposed preliminary mining labour requirements to operate the mine and the number of personnel required to maintain the mobile equipment, together with the estimated annual salary for the various positions. The labour requirements are based on a fly-in/fly-out 2-week operating cycle with two 12-hour shifts per day. Autonomous mining fleet should be considered in the next phases of the study.

Table 21-12: Mine Operations Labour Requirement and Operating Cost ($)

Position	Staff	Annual Salary	Total Annual Salary + 40% Overhead	Cost per Tonne Moved
Operations Management:				
Mine Manager	1	270,000	378,000	-
Chief Mine Engineer	1	270,000	378,000	-
Mine Operations Superintendent	2	185,000	518,000	-
Mobile Maintenance Manager	1	150,000	210,000	-
Total	**5**		**1,484,000**	**0.014**
Mine Operations				
Pit Foreman	4	130,000	728,000	-
Dispatch Supervisor	1	90,000	126,000	-
Dispatchers	4	83,000	464,800	-
Heavy Equipment Operators	132	96,000	17,740,800	-
Pit Utility Operators	4	80,000	448,000	-
Total	**145**		**19,507,600**	**0.187**
Mobile Maintenance				
Mobile Maintenance Technicians	32	100,000	4,480,000	-
Maintenance Foreman	2	150,000	420,000	-
Total	**34**		**4,900,000**	**0.047**
Geotechnical Team				
Geotechnical Superintendent	1	200,000	280,000	-
Senior Geotechnical Engineers	2	150,000	420,000	-
Geotechnical Engineers	4	130,000	728,000	-
Geotechnical Technologist	3	80,000	336,000	-
Slope Monitoring Technicians	2	70,000	196,000	-
Total	**12**		**1,960,000**	**0.019**
Mine Engineering Team				
Senior Mine Planner	4	150,000	840,000	-
Mine Planner	6	130,000	1,092,000	-
Mine Surveyors	4	80,000	448,000	-
Total	**14**		**2,380,000**	**0.023**

Position	Staff	Annual Salary	Total Annual Salary + 40% Overhead	Cost per Tonne Moved
Drilling and Blasting Team				
Senior Drill and Blast Engineer	1	150,000	210,000	-
Drill and Blast Engineer	2	130,000	364,000	-
Drill and Blast Foreman	2	130,000	364,000	-
Blasters	8	80,000	896,000	-
Blaster Assistant	4	70,000	392,000	-
Total	**17**		**2,226,000**	**0.021**
Mining Labour Cost per Tonnage Moved				**0.31**

21.2.2 Processing Operating Cost Estimate

The processing OPEX estimate was prepared for the 120,000 t/d process facility considered in the PEA, and reflects the processing scenarios evaluated. The estimate includes costs associated with crushing, grinding, flotation recovery to produce a copper concentrate and a cobalt-pyrite concentrate; downstream hydrometallurgical treatment of the cobalt-bearing concentrate; and, for the scandium-recovery case, additional treatment of a portion of the flotation tailings. The remaining flotation tailings are assumed to be deposited in the tailings storage facility.

The estimate is based on a nominal throughput of 120,000 t/d, corresponding to 42.0 Mt/a, with assumed crusher availability of 70%, mill availability of 92%, and 350 operating days per year.

Summary of Processing Operating Cost

Table 21-13 presents the estimated annual processing OPEX for the processing scenarios considered in the PEA. The Scenario A estimate includes the OPEX for the 120,000 t/d concentrator, comprising power; process and maintenance labour; reagents; process plant supplies; and maintenance supplies. This scenario reflects the base flotation processing route.

For Scenario B, the processing OPEX estimate includes the full Scenario A concentrator cost, together with the additional OPEX for the downstream hydrometallurgical plant and the scandium oxide processing circuit. Scenario B therefore represents the expanded flowsheet case, incorporating both the cobalt-related hydrometallurgical treatment circuit and the additional scandium-recovery circuit.

The total annual OPEX for the concentrator under Scenario A is estimated at $333.2 million or $7.93/t processed. All estimates are based on the use of grid power; costs would be significantly higher if the operation were powered by diesel generation.

Table 21-13: Summary of the Estimated Annual Processing Operating Cost ($)

Cost Item	Staff	Total Annual Cost	Cost per Tonne Milled	Comment
Power Cost Using Grid Power	-	76,289,169	1.816	See Table 21-15
Process Plant Labour	91	12,122,240	0.289	See Table 21-14
Process Plant Supplies	-	145,320,000	3.460	See Table 21-17
Process Plant Reagents	-	56,425,950	1.343	See Table 21-18
Plant Maintenance Labour	55	7,768,160	0.185	See Table 21-14
Plant Maintenance Supplies	-	35,280,000	0.840	See Table 21-16
Total Process Plant (Scenario A)	**146**	**333,205,519**	**7.933**	
Hydrometallurgical Plant (Scenario B)	-[1]	129,486,000	3.083	See Table 21-19
Scandium Oxide Processing (Scenario B)	-[1]	146,569,800[2]	3.490	See Table 21-19

Notes: [1] The labour component for Scenario B is assumed to be the same as for Scenario A. As such, the costs shown for hydrometallurgical and scandium oxide processing represent the incremental OPEX associated with the additional Scenario B circuits above the base Scenario A concentrator cost.
[2] Scandium Oxide Processing total annual cost based on 156 ton/year production and $939.55/kg produced Sc_2O_3.

Processing Plant Labour

Table 21-14 gives the estimated proposed processing facility labour requirements to operate the plant and to maintain the equipment, together with the estimated annual salary for the various positions. The labour requirements are based on a fly-in/fly-out 2-week operating cycle with two 12-hour shifts per day. This equates to 2,190 hours per year worked by the shift operators, and 2,080 hours per year worked by the salaried staff. Certain specialized-unit process-plant maintenance will be conducted during the annual maintenance shutdown period, and specialized mill-relining contractors will conduct SAG and ball mill relining as required.

Table 21-14: Process Plant Labour Requirements and Annual Operating Cost Estimate ($)

Position	Staff	Annual Salary	Annual Total	Cost per Tonne Milled
Plant Operation Management				
Plant Manager	1	320,800	320,800	-
Chief Metallurgist	1	292,000	292,000	-
Plant Foreman	2	194,400	388,800	-
Mill Clerk	4	110,880	443,520	-
Total	**8**	**-**	**1,445,120**	**0.034**
Plant Operation Team				
Shift Supervisors	6	174,880	1,049,280	-
Crusher and Stockpile	9	110,880	997,920	-
Control Room	4	120,640	482,560	-
Grinding Section	9	110,880	997,920	-
Flotation Section	9	120,640	1,085,760	-

Position	Staff	Annual Salary	Annual Total	Cost per Tonne Milled
Dewatering Section	4	110,880	443,520	-
Concentrate Loadout	4	110,880	443,520	-
Tailings Operations	6	110,880	665,280	-
Trainee Operators	4	87,040	348,160	-
Total	**55**	**-**	**6,513,920**	**0.155**
Assay and Metallurgical Team				
Senior Metallurgist	6	194,400	1,166,400	-
Junior Metallurgist	4	174,880	699,520	-
Senior Chemist	4	194,400	777,600	-
Assayer	8	110,880	887,040	-
Laboratory Technicians	6	105,440	632,640	-
Total	**28**	-	**4,163,200**	**0.099**
Total Operations	**91**	-	**12,122,240**	**0.289**
Maintenance and Engineering Management				
Maintenance Manager	2	292,000	584,000	-
Total	**2**	**-**	**584,000**	**0.014**
Maintenance Team				
Shift Supervisors	6	174,880	1,049,280	-
Planners	3	135,840	407,520	-
Maintenance Clerks	3	110,880	332,640	-
Millwrights	10	135,840	1,358,400	-
Welders	6	135,840	815,040	-
Electricians	8	135,840	1,086,720	-
Instrument Technician	10	135,840	1,358,400	-
Apprentices	7	110,880	776,160	-
Total	**53**	-	**7,184,160**	**0.171**
Total Maintenance and Engineering	**55**	-	**7,768,160**	**0.185**
Total Plant Labour Cost	**146**	-	**19,890,400**	**0.474**

Process Plant Power

Table 21-15 gives the estimated power required for two processing lines incorporating crushing, grinding, and flotation sections, assuming a loading factor and utilization of 75%. Note that these numbers are only an approximation, since detailed electrical loading and efficiency factors have not been included in this estimate.

Table 21-15: Power Operating Cost Estimate ($)

	kWh per Year	Per kWh	Per Year	Per Tonne Milled
Plant Power	1,089,845,266	0.07	76,289,169	1.816

Maintenance Supplies

The maintenance department annual supplies requirement to keep the plant operational is estimated to be $35.3 million per year, or $0.84/t milled (Table 21-16). The estimated cost of supplies for the process plant is $145.3 million per year, or $3.46/t milled (Table 21-17). The cost estimates for supplies are based on the processing rate of 42 Mt/a.

Table 21-16: Maintenance Supplies—Operating Cost Estimate ($)

Item	Cost per Year	Cost per Tonne Milled	Comments
Crusher and Conveyor Area	4,200,000	0.10	
Stockpile Area	1,260,000	0.03	
Grinding Area	10,500,000	0.25	
Flotation Area	10,500,000	0.25	Includes the regrind mills
Concentrate Dewatering	2,100,000	0.05	
Concentrate Storage	840,000	0.02	Includes the loadout area
Reagent Preparation Area	1,260,000	0.03	
Assay Laboratory	840,000	0.02	
Tailings Area	1,680,000	0.04	
Miscellaneous	2,100,000	0.05	
Total	**35,280,000**	**0.84**	

Table 21-17: Process Plant Operating Supplies—Estimate ($)

Item	Cost per Year	Cost per Tonne Milled	Comments
Crusher Liners	2,520,000	0.06	
Conveyor Supplies	1,260,000	0.03	
SAG Mill Liners	15,120,000	0.36	
Ball Mill Liners	16,800,000	0.40	
Regrind Mill Discs & Liners	12,600,000	0.30	
SAG Mill Media	17,640,000	0.42	
Ball Mill Media	46,200,000	1.10	
Regrind Mill Media	8,400,000	0.20	Includes ceramic media
Concentrate Dewatering	420,000	0.01	Includes the loadout Area
Filter Cloths	8,400,000	0.20	
Reagent Area Supplies	840,000	0.02	
Assay Laboratory Supplies	6,720,000	0.16	
Miscellaneous	8,400,000	0.20	Includes forklift, light motor vehicles, etc.
Total	**145,320,000**	**3.46**	

Process Plant Consumables

Table 21-18 gives the process plant reagent consumables, which included the estimated daily reagent usage, approximate unit costs, and annual reagent cost. The reagent consumption assumptions are based on limited bench-scale testing and have not been optimized for the flotation process. Accordingly, an overall consumption factor of 50% has been applied to all reagents to derive the consumables cost estimate used in this study.

The reagent costs are based on indicative pricing rather than budget quotations, and these estimates are expected to be refined as the Project advances and enters into commercial discussions and contractual arrangements with reagent suppliers. In addition, reagent purity has not been accounted for at this stage, and freight and handling costs have not yet been specifically determined. The reagents listed in Table 21-18 are those used in the flotation and dewatering circuits of the process plant and do not include glycol, diesel, anti-scalant, water treatment chemicals, or oils and lubricants used elsewhere in the plant.

For the purpose of this PEA, the same reagent cost per processed tonne has been retained for the concentrator in both Scenario A and B. While Scenario B includes an additional cobalt–pyrite flotation stream, requiring reagents such as sulphuric acid, copper sulphate activator, and PAX collector, the available metallurgical testwork is not yet considered sufficiently advanced to support a reliable scenario-specific apportionment of reagent consumption between the two flowsheet configurations. Accordingly, the reagent schedule presented in Table 21-18 has been treated as a preliminary composite estimate for the concentrator flowsheet at the current level of study.

Retaining the same unit consumables cost basis for both scenarios is considered appropriate and conservative at this stage, particularly given that reagent consumption has not been optimized, and the estimate already incorporates the 50% consumption factor to account for uncertainty in reagent demand. Further refinement of reagent consumption, by scenario, should be undertaken in subsequent study phases as additional metallurgical testing and process design information become available.

Table 21-18: Reagent Consumption and Estimated Operating Cost ($)

Reagent	Daily Consumption (kg)	Annual Consumption (kg)	Cost per kg	Annual Cost	Cost per Year ($/t milled)
Lime	60,000	21,000,000	0.40	8,400,000	0.200
Sulphuric Acid	132,000	46,200,000	0.30	13,860,000	0.330
Sodium Cyanide	1,200	420,000	4.49	1,885,800	0.045
Copper Sulphate	48,000	16,800,000	3.57	59,976,000	1.428
SIPX	6,000	2,100,000	2.29	4,809,000	0.115
PAX	4,800	1,680,000	3.57	5,997,600	0.143
A3477	3,600	1,260,000	3.10	3,906,000	0.093
MIBC	7,200	2,520,000	5.14	12,952,800	0.308
Flocculant	600	210,000	5.07	1,064,700	0.025
Total		**112,851,900**		**2.687**	
Optimization Factor—50%		**56,425,950**		**1.343**	

21.2.3 Hydrometallurgical Processing Plant

Table 21-19 summarizes the OPEX estimate for the hydrometallurgical processing plant included in Scenario B. The estimate covers the downstream treatment of the cobalt–pyrite concentrate, including concentrate drying; roasting; waste-heat recovery and acid generation; cobalt extraction and recovery; and gold extraction and recovery; as well as the associated scandium-recovery cost where applicable. The roasting and acid plant are expected to generate significant recoverable waste heat, which can be used for steam and electrical power generation. This is consistent with established industry practice for roasting and sulphuric acid plants and is expected to substantially reduce the net power demand of the hydrometallurgical circuit, subject to confirmation by detailed energy balance analysis in future study phases.

Consistent with the staged development strategy for Scenario B, construction and commissioning of the hydrometallurgical plant are assumed to be undertaken in Year 3–4 of concentrator plant production.

Table 21-19: Hydrometallurgical Plant Operating Cost Summary ($)

Cost Item	Cost per Unit	Unit	Cost per Tonne Milled
Concentrate Drying	11.07	t pyrite	0.14
Roasting, Waste Heat Recovery and Acid Plant	137.00	t acid	2.18
Cobalt Extraction and Recovery	9,996.79	t cathode	0.59
Gold Extraction and Recovery	20.55	t calcine	0.18
Hydrometallurgical Plant OPEX	**-**	**-**	**3.08**
Scandium Extraction and Recovery	939.55	kg Sc_2O_3	3.49

The hydrometallurgical plant operating cost estimate was increased by 13.7%, from a base value of $3.08/t milled to $3.50/t milled, to account for the stockpiling of cobalt-pyrite concentrate produced prior to commissioning of the hydrometallurgical plant and its subsequent processing on a pro-rated basis over the remaining operating life. The financial model was developed to assess alternative implementation strategies, including variations in the timing of hydrometallurgical plant construction and commissioning, and the treatment of cobalt-pyrite with or without interim stockpiling prior to start-up of the downstream circuit.

21.2.4 General and Administrative Cost Estimates

An estimate of the General and Administrative labour requirement and supply costs has been compiled in Table 21-20 and Table 21-21, respectively. These have been compiled with the help of a similar-sized mine in Canada. At this stage it is not known whether the Hat Project will follow this type of outline of administrative department labour and expenses, or, because it is in a remote location, certain job descriptions may be allowed to conduct their tasks remotely, or from the corporate city-based offices, or whether some jobs may be aggregated into more general job descriptions, or contracted out to independent service companies. The information presented below can be used as a guide. Note that the list is not inclusive, and certain items are site-specific, such as the funding and possible staffing required for road maintenance.

Table 21-20: General and Administrative Labour Requirements—Estimated Operating Cost ($)

Position	Staff	Annual Salary	Annual Total	Cost per Tonne Milled
General & Administrative				
General Manager	1	398,520	398,520	
Secretary	1	135,840	135,840	
Administrative Manager	1	174,880	174,880	
HSEC Manager	1	194,400	194,400	
IT Services Manager	1	135,840	135,840	
Total	**5**		**1,039,480**	**0.025**
Accounting				
Chief Accountant	1	174,880	174,880	
Payroll Clerk	1	110,880	110,880	
Accounting Clerk	1	110,880	110,880	
Total	**3**		**396,640**	**0.009**
Purchasing and Warehouse				
Chief Purchaser	1	174,880	174,880	
Assistant Purchaser	1	135,840	135,840	
Warehouse Shipper / Receiver	4	110,880	443,520	
Janitorial Staff	4	110,880	443,520	
Gate Security	6	110,880	665,280	
Total	**16**		**2,656,320**	**0.063**
HR and Training				
HR Manager	1	174,880	174,880	
HR Clerk	1	110,880	110,880	
Total	**2**		**285,760**	**0.007**
Health and Safety				
Health and Safety Coordinator	1	174,880	174,880	
Safety and Training Officer	1	135,840	135,840	
First Aid Staff	4	110,880	443,520	
Total	**6**		**754,240**	**0.018**
Environmental				
Environmental & Community Liaison Manager	1	174,880	174,880	
Environmental Technician	2	120,640	241,280	
Total	**3**		**416,160**	**0.010**
Site Services				
Site Services General Supervisor	1	194,400	194,400	
Site Services Supervisor	1	174,880	174,880	
Site Yard General Labourers	4	87,040	348,160	
Site Electrician	2	135,840	271,680	
Site Mechanic	2	135,840	271,680	
Site Carpenter	1	135,840	135,840	
Total	**11**		**1,396,640**	**0.033**
Total G&A Labour	**46**		**6,945,240**	**0.165**

Table 21-21: General and Administrative Supplies—Estimated Operating Cost ($)

Item	Annual Cost	Cost per Tonne Milled	Comments
Camp	14,665,000	0.35	$100/d per staff
Crew Transportation	12,600,000	0.30	-
Freight	12,600,000	0.30	-
Administration and HR	4,200,000	0.10	Includes communication services and recruitment
Insurance and Legal	4,200,000	0.10	-
Environmental	840,000	0.02	-
Safety and First Aid	840,000	0.02	Includes medical services
Training	1,764,000	0.04	-
Security	2,100,000	0.05	Includes patrol vehicles
Site Services	1,260,000	0.03	-
Purchasing	1,260,000	0.03	-
Community Public Relations	1,260,000	0.03	-
Miscellaneous	2,100,000	0.05	Includes road maintenance
Total—G&A Supplies	**59,689,000**	**1.42**	**-**

21.2.5 Tailings Storage Facility Operating Cost

Table 21-22 presents the preliminary OPEX estimate for operation and management of the TSF and associated water treatment requirements. Based on a total LOM mill feed of approximately 1,025.7 Mt over 25 years of operation, the estimated average TSF OPEX is $0.97/t milled.

The estimate comprises annual costs for TSF management, monitoring, engineering, and maintenance, operation of the tailings and reclaim water pipeline systems, and water treatment.

Table 21-22: Preliminary TSF Operating Cost Estimate ($)

Operation	Annual Cost	LOM Cost	Cost per Tonne Milled
Management, Monitoring, Engineering, Maintenance	9,000,000	225,000,000	0.22
Operating Tailings and Reclaim Pipelines	29,360,000	734,000,000	0.72
Water Treatment	1,500,000	37,500,000	0.04
Total			**0.97**

21.2.6 Operating Cost Estimate Summary

Table 21-23 summarizes the average OPEX based on the production schedule and different processing scenarios. Figure 21-2 shows the annual OPEX by processing scenario over the LOM for the presented schedule and production (Section 16.6).

Table 21-23: Average Operating Cost over LOM ($)

Operating Cost over LOM	Unit	Value
Average Mining Cost[1]	t moved	2.32
Processing Cost—Up to Concentrate production (Scenario A)	t milled	7.93
Hydrometallurgical + Acid Generation (Scenario B)	t milled	3.08
Scandium Oxide Processing (Scenario B)	kg Sc_2O_3	939.55
G&A	t milled	1.59
TSF Associated Operating Cost	t milled	0.97
Average Total Operating Cost (Scenario A/B)	t milled	16.22/21.92

Note: [1] Mining cost range $1.89–2.58/t mined and rehandle cost of $0.81/t moved.

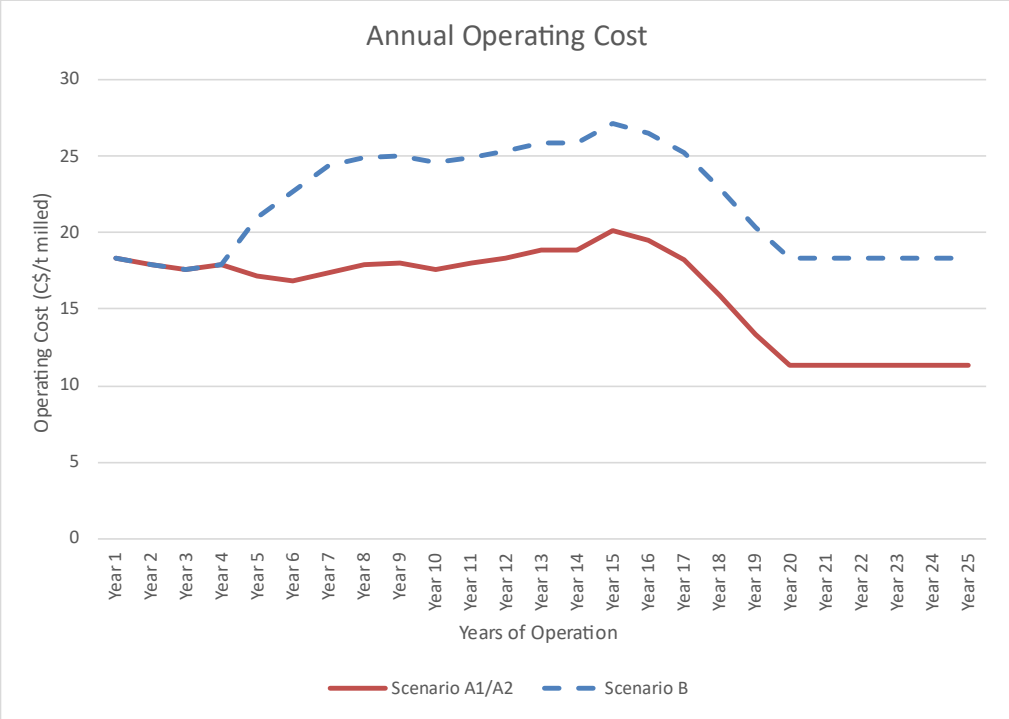


Figure 21-2: Annual Operating Cost by Scenario over the Life-of-Mine

22 ECONOMIC ANALYSIS

An economic model was developed for the Hat Project to estimate annual cash flows and assess Project economics and sensitivity to key input parameters. The model is based on a two-year construction period followed by a 25-year operating life, with low-grade stockpiles processed during the period from Year 19 to Year 25. The PEA contemplates a conventional open pit mine and processing operation at a nominal throughput of 120,000 t/d (42 Mt/a).

All values presented in this section are reported in Canadian dollars, unless otherwise stated. Totals may not sum due to rounding.

22.1 Cautionary Statement

This economic analysis presents the results of a PEA completed to evaluate the potential economic viability of the Hat Project. The analysis is preliminary in nature and includes Inferred Mineral Resources, which 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 results presented herein will be realized.

The economic outcomes presented in this section are intended to support early-stage Project evaluation and decision-making. They should not be construed as a demonstration of economic viability, nor as a substitute for a pre-feasibility study or feasibility study as defined under NI 43-101.

22.2 Method

The financial analysis for the Hat Project was completed using a discounted cash flow model based on the annual life-of-mine production schedule derived from the mine plan and the processing assumptions for the three scenarios evaluated: Scenario A1, a Cu-Au-Ag-Co flotation base case using current testwork recoveries; Scenario A2, the same base case using expected recoveries; and Scenario B, a Cu-Au-Ag-Co flowsheet with an added hydrometallurgical circuit and scandium recovery circuit. (Refer to Section 17).

The financial model estimates annual revenues, operating costs, capital expenditures, royalties, taxes, and resulting cash flows over the projected mine life. All costs and financial results are presented in constant Canadian dollars ($). Metal prices are based on US dollar denominated assumptions and were converted using a fixed exchange rate of 1.37 CAD:USD.

The NPV was calculated using a 5% discount rate, and the internal rate of return (IRR) is defined as the discount rate at which the present value of the projected annual cash flows, including the initial capital outlay, is equal to zero.

22.3 Financial Model Parameters

22.3.1 Commodity Prices and Exchange Rates

The economic analysis was prepared using analyst consensus forecast metal prices considered reasonable and appropriate for a PEA-level study. The selected price assumptions reflect real prices long-term consensus outlook (Table 22-1).

Where applicable, by-product metal credits have been incorporated into the revenue model based on assumed payable terms and recoveries. All commodity prices are expressed in US$.

Table 22-1: Metal Prices Long-Term Forecast

Metal	Unit	Analyst Consensus Prices[1]	Spot Prices[2]
Copper	US$/lb	4.88	6.00
Gold	US$/oz	3,272	5,200
Silver	US$/oz	50.22	90.0
Cobalt	US$/lb	19.57	25.50
Scandium oxide (Sc_2O_3)	US$/kg	1,500	1,500

Notes: [1] Analyst consensus forecast prices as of February 20, 2026 (EMCF).
[2] Spot prices as February 25, 2026.

22.3.2 Commercial Terms

As described in Section 17, under Scenarios A1 and A2 the Hat Project is assumed to produce a marketable copper concentrate containing payable copper, gold, silver, and cobalt. Under Scenario B, the Project is assumed to produce on-site saleable products comprising of gold-silver doré, cobalt cathode, and scandium oxide. Table 22-2 summarizes the smelter terms, refining charges, transport and selling costs, and payable assumptions applied in the economic analysis for these products.

Table 22-2: Summary of Commercial Terms Applied to Cu–Au Concentrate and On-Site Products

Cu–Au Concentrate Assumptions	Unit	
Transport and Selling	$/dmt	95.90
TC Cu–Au Concentrate	$/dmt	77.00
Refining Cost—Cu	$/lb	0.11
Refining Cost—Au	$/oz	6.85
Refining Cost—Ag	$/oz	0.48
Refining Cost—Co	$/lb	0.16
Payable—Cu	%	97
Payable—Au	%	95
Payable—Ag	%	90
Payable—Co	%	80
On-Site Production (Scenario B)		

Cu–Au Concentrate Assumptions	Unit	
Payable—Au	%	97
Payable—Ag	%	97
Payable—Co	%	97
Payable—Sc_2O_3	%	95

Commercial terms were applied through the NSR formulation as deductions from gross payable metal value; they are therefore excluded from operating cost estimates.

22.3.3 Capital Cost Assumptions

The capital cost estimate includes initial capital, sustaining capital, and closure-related capital, as described in Section 21. Initial capital encompasses mine development, processing facilities, infrastructure, indirect costs, and an appropriate contingency consistent with PEA-level studies.

Sustaining capital includes expenditures required to maintain operations over the life of the Project, including mobile equipment replacements and Life Cycle Cost, tailings expansions, and sustaining infrastructure. Closure costs are included as an end-of-life cash outflow.

The capital cost estimate has been prepared at an AACE Class 5 level and reflects a PEA-level accuracy range and is subject to significant uncertainty. Table 22-3 summarizes the capital cost with the breakdown of the Initial and Sustaining CAPEX on Figure 22-1.

Table 22-3: Capital Cost Estimate Summary

Capital Cost Summary	Scenario A1 ($)	Scenario A2 ($)	Scenario B ($)
Total Initial CAPEX	3,552	3,601	3,828
Total Sustaining CAPEX	2,755	2,755	4,006
Closure and Reclamation	503	503	503

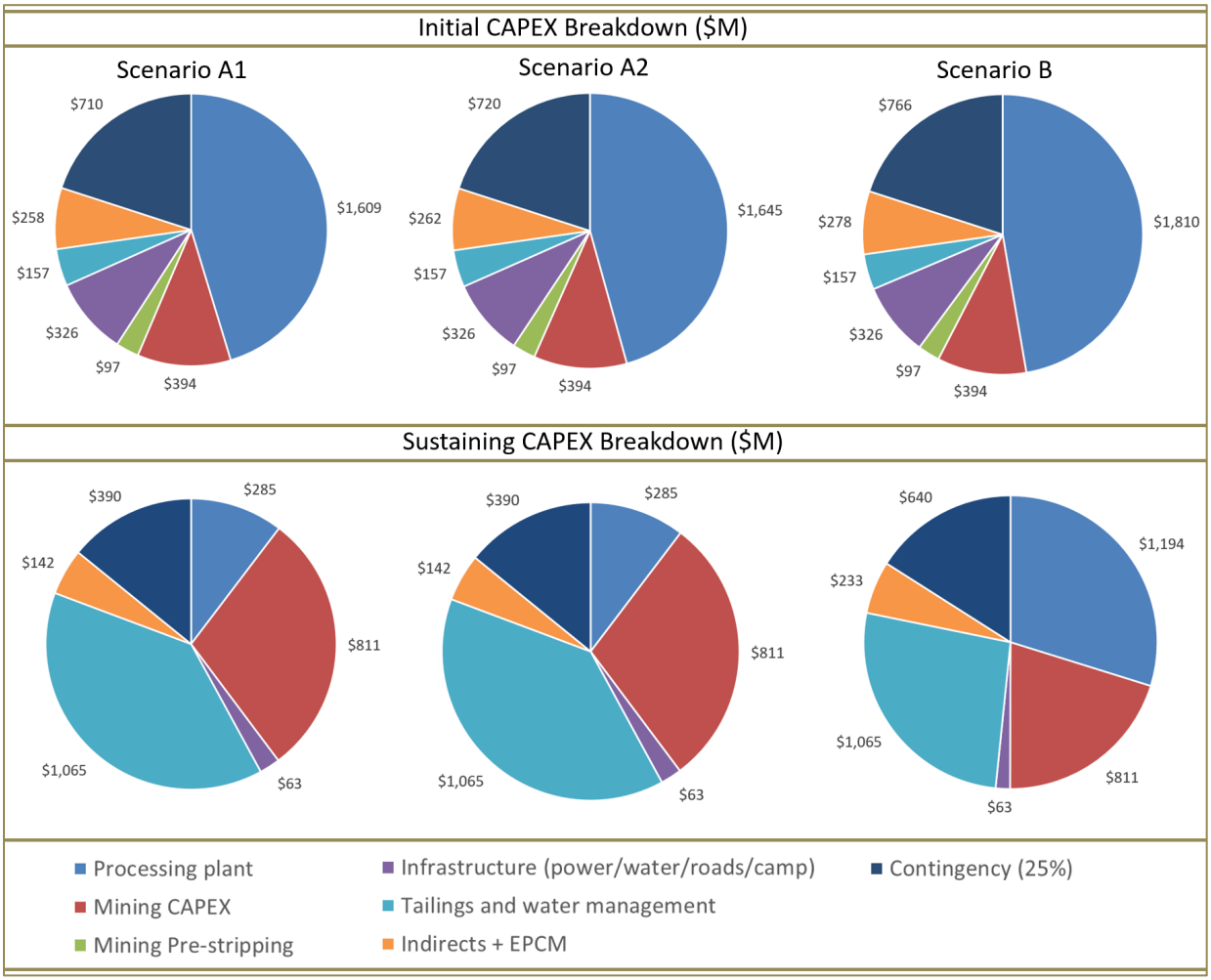


Figure 22-1: Initial and Sustaining CAPEX Breakdown

22.3.4 Operating Cost Assumptions

Operating costs include mining, processing, general and administrative (G&A), and other site operating costs (Table 22-4). Costs are expressed on a unit basis (e.g., $/t mined, $/t milled, or $/lb payable metal) and are applied annually based on the mine and process schedules (Figure 22-2).

Operating costs are assumed constant in real terms for the duration of the mine life.

Table 22-4: Operating Cost Breakdown

Operating Cost Over LOM	Unit	Value
Mining Cost[1]	$/t moved	1.89 to 2.58
Processing Cost—Up to Concentrate production (Scenario A)	$/t milled	7.93
Hydrometallurgical + Acid Generation (Scenario B)	$/t milled	3.08
Scandium Oxide Processing (Scenario B)	$/kg Sc_2O_3	939.55
G&A	$/t milled	1.59
TSF Associated Operating Cost	$/t milled	0.97
Average Total Operating Cost (Scenario A/B)	**$/t milled**	**16.22/21.92**

Notes: [1]The phased increase in mining fleet requirements reflects progressive pit deepening and the corresponding increase in haulage distances as mining advances.

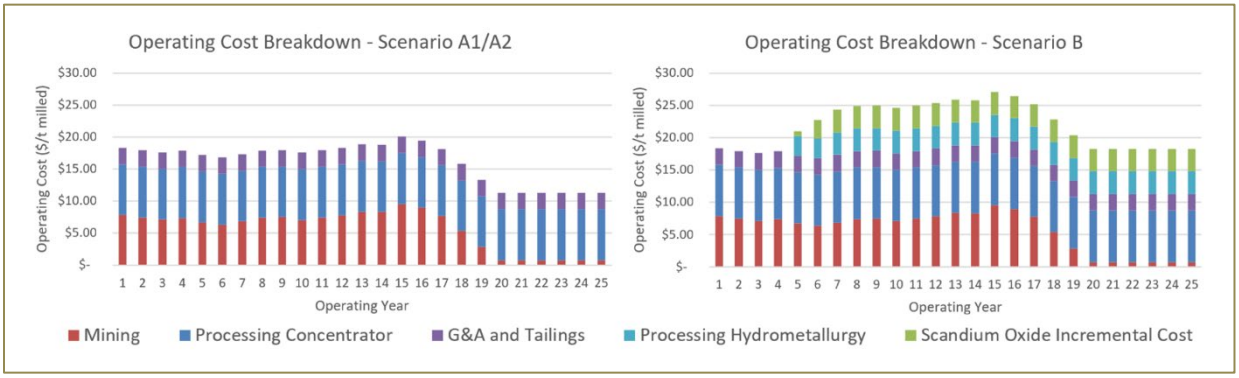


Figure 22-2: Annual Operating Cost Breakdown, Scenario A and B

22.3.5 Production Summary

Table 22-5 summarizes the production and payable production over the LOM for each scenario based on the production schedule presented on Section 16.6.

Table 22-5: Production Summary

Metric	Unit	Scenario A1	Scenario A2	Scenario B
Mining Summary				
Average Strip Ratio	t:t	1.6		
Production Summary LOM				
Average Annual Throughput	Mt	42		
CuEq Head Grade[1]	%	0.42		
Cu Head Grade	%	0.19		
Au Head Grade	g/t	0.19		
Ag Head Grade	g/t	0.51		
Co Head Grade	g/t	77.73		

Metric	Unit	Scenario A1	Scenario A2	Scenario B
Sc Head Grade	g/t	28.35		
Cu Recovery	%	80	89	85
Au Recovery	%	66	75	89
Ag Recovery	%	53	53	68
Co Recovery	%	30	30	78
Sc Recovery	%	N/A		72
Overall Mass of Tailings to Process	%	N/A		12.5
Year of Production Start of Sc_2O_3	year	N/A		4
Average Annual Cu Production	kt	63.6	70.8	67.6
Total Cu Production	**kt**	**1,590.5**	**1,769.4**	**1,689.9**
Average Annual Payable Cu	kt	61.7	68.7	65.7
Total Payable Cu	**kt**	**1,542.8**	**1,716.3**	**1,642.2**
Average Annual Au Production	koz	161.1	183.1	217.3
Total Au Production	**koz**	**4,028.2**	**4,577.5**	**5,432.0**
Average Annual Payable Au	koz	153.1	173.9	207.5
Total Payable Au	**koz**	**3,826.8**	**4,348.7**	**5,188.6**
Average Annual Ag Production	koz	271.3	271.3	348
Total Ag Production	**koz**	**6781.6**	**6781.6**	**8700.9**
Average Annual Payable Ag	koz	244.1	244.1	318.6
Total Payable Ag	**koz**	**6,103.4**	**6,103.4**	**7,965.3**
Average Annual Co Production	kt	1.0	1.0	2.5
Total Co Production	**kt**	**23.9**	**23.9**	**62.2**
Average Annual Payable Co	kt	0.8	0.8	2.3
Total Payable Co	**kt**	**19.1**	**19.1**	**56.3**
Average Annual Sc_2O_3 Production	t	N/A		128.4[2]
Total Sc_2O_3 Production	**t**	**N/A**		**3,209.5**
Total Sc_2O_3 Payable	**t**	**N/A**		**3,049.0**

Notes: [1]Scandium not included for copper-equivalent calculation.
[2] Average yearly Sc_2O_3 Production of 165 t/a after Scandium recovery circuit ramp-up.

22.3.6 Taxes and Royalties

Taxes and royalties for the Project have been estimated using the applicable British Columbia and Canadian fiscal framework. The economic model includes B.C. Mineral Tax, provincial income tax, federal income tax, and applicable royalty payments. The tax calculation considers available deductions and tax pools, including eligible capital expenditures and development and exploration expenses, where applicable. Mineral tax has been modelled in accordance with the relevant B.C. provisions and corresponding deductions have been reflected in the corporate income tax calculations as appropriate. The Hat property is subject to a 2% NSR royalty. For the purposes of the preliminary economic analysis,

capital cost allowance has been applied using an assumed rate of 25%; however, the actual treatment of depreciation and tax deductions will depend on the final asset classification, applicable tax legislation at the time of development, and the tax strategy ultimately adopted by the Project. Provincial GST and PST are not included in the capital costs and operating costs.

Accordingly, a more detailed tax review and optimization strategy should be completed in subsequent study phases. Table 22-6 summarises the principal tax and royalty assumptions adopted and the resulting estimated undiscounted life-of-mine amounts.

Table 22-6: Total Taxation and Royalty estimate

Item	Unit	Scenario A1	Scenario A2	Scenario B
Net Present Value Discount Rate	%	5	5	5
BC Mineral Tax Rates	%	13	13	13
BC Provincial Income Tax Rates	%	12	12	12
Federal Income Tax Rates	%	15	15	15
BC Mineral Tax	$M	2,388	2,971	3,501
BC Income Tax	$M	1,641	2,118	2,517
Federal Income Tax	$M	1,274	1,859	2,308
Total Taxes Estimate	$M	5,303	6,949	8,327
Royalties	$M	794	888	1,116

22.4 Economic Results

Table 22-7 summarizes the resulting financial metrics for Scenarios A1, A2, B. The Project generates average annual EBITDA of $886 million (A1), $1,071 million (A2), and $1,284 million (B). On a post-tax basis, NPV (5%) is estimated at $4,963 million (A1), $6,727 million (A2), and $7,274 million (B) with corresponding post-tax IRRs of 19%, 23%, and 19%, and post-tax payback in Year 6 (A1), Year 5 (A2), and Year 7 (B). Total post-tax free cash flow is estimated at $10,050 million (A1), $12,961 million (A2), and $15,437 million (B), reflecting the higher cash generation under the improved recovery case (A2) and the additional revenue streams in Scenario B, partially offset by the added capital and operating requirements of the hydrometallurgical and scandium circuits.

Figure 22-3 presents the annual and cumulative cash flow profile reflects an initial construction period followed by a ramp-up phase and steady-state operations. Peak negative cash flow occurs during the construction phase, after which cumulative cash flow becomes positive during operations.

Table 22-7: Summary of Economic Results—Consensus Metal Prices

Metric	Unit	Scenario A1	Scenario A2	Scenario B
Financial Metrics				
Exchange Rate	CAD:USD	1.37		
Long Term Copper Price	US$/lb	4.88		
Long Term Gold Price	US$/oz	3,272.60		
Long Term Silver Price	US$/oz	50.22		
Long Term Cobalt Price	US$/lb	19.57		
Long Term Scandium Oxide Price	US$/kg	N/A		1,500
Average Annual EBITDA	$M	886	1,071	1,284
Total EBITDA	**$M**	**22,162**	**26,770**	**32,101**
Average Annual Free Cash Flow (Pre-tax)	$M	756	940	1,104
Free Cash Flow (Pre-tax)[1]	$M	18,904	23,511	27,592
Total Provincial Tax (Including BC Mineral Tax)	**$M**	**(4,029)**	**(5,090)**	**(6,019)**
Total Federal Tax	**$M**	**(1,274)**	**(1,859)**	**(2,308)**
Total Taxes	**$M**	**(5,303)**	**(6,949)**	**(8,327)**
Average Annual Free Cash Flow (Post-tax)	$M	544	662	771
Free Cash Flow (Post-tax)[1]	$M	13,601	16,562	19,265
Total Free Cash Flow (Pre-tax)[2]	**$M**	**15,352**	**19,910**	**23,764**
Total Free Cash Flow (Post-tax)[2]	**$M**	**10,050**	**12,961**	**15,437**
NPV 5% (Pre-Tax)	$M	7,883	10,576	11,567
NPV 5% (Pre-Tax)	US$M	5,754	7,720	8,443
IRR (Pre-Tax)	%	24	29	23
Payback (Pre-Tax)	years	Year 5	Year 4	Year 6
NPV 5% (Post-Tax)	$M	4,963	6,727	7,274
NPV 5% (Post-Tax)	US$M	3,623	4,911	5,309
IRR (Post-Tax)	%	19	23	19
Payback (Post-Tax)	years	Year 6	Year 5	Year 7

Notes: [1] Free Cash Flow during production periods only.
[2] Total life of mine Free Cash Flow, including initial capital costs and closure.

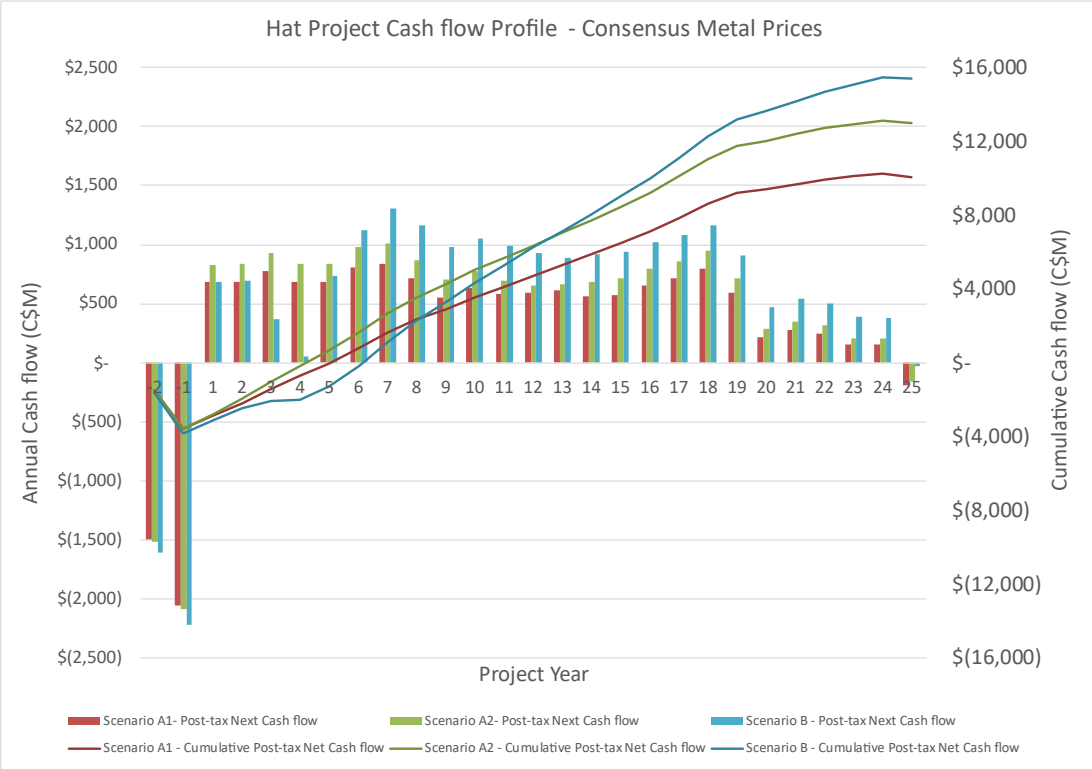


Figure 22-3: Annual and Cumulative Post-Tax Cashflow—Consensus Metal Prices

Table 22-8 summarizes the average net revenue contribution by payable metal for each scenario under the consensus metal price assumptions, illustrating the relative importance of the Project's principal revenue drivers. The results indicate that the Project is primarily a copper–gold operation in all scenarios, with copper and gold together accounting for the majority of net revenue. In Scenarios A1 and A2, copper remains the largest single contributor, while in Scenario B the revenue mix becomes more diversified due to the improved gold recovery and additional contribution from scandium and the higher contribution from cobalt, with gold slightly exceeding copper as the largest individual contributor. Overall, Scenario B is presented as an upside case rather than the base development case, reflecting the additional processing complexity and reliance on secondary revenue streams, while the Project's underlying economics remain principally supported by copper and gold.

Table 22-8: Average Net Revenue by Metal—Consensus Metal Prices

Net Revenue by Metal	Scenario A1 (%)	Scenario A2 (%)	Scenario B (%)
Copper	52.3	52.0	39.7
Gold	43.4	44.2	41.6
Silver	1.1	1.0	1.0
Cobalt	3.1	2.8	6.1
Scandium	0.0	0.0	11.5
Total	**100.0**	**100.0**	**100.0**

Table 22-9 and Figure 22-4 presents the financial results based on spot metal price assumptions, which are intended to illustrate the potential up-side of the Project under a stronger commodity price environment. Using these assumptions, the Project is estimated to generate average annual EBITDA of $1,514 million (A1), $1,775 million (A2), and $2,096 million (B). The corresponding post-tax NPV (5%) is estimated at $11,047 million (A1), $13,526 million (A2), and $14,848 million (B), with post-tax IRRs of 34%, 39%, and 32% respectively. Post-tax payback is achieved in Year 3 (A1), Year 3 (A2), and Year 4 (B). As these results are based on spot pricing, they are presented for illustrative purposes only and should not be interpreted as the base case for the Project.

Table 22-9: Summary of Economic Results—Spot Metal Prices

Metric	Unit	Scenario A1	Scenario A2	Scenario B
Financial Metrics				
Exchange Rate	CAD:USD	1.37		
Spot Copper Price	US$/lb	6.00		
Spot Gold Price	US$/oz	5,200.00		
Spot Silver Price	US$/oz	90.00		
Spot Cobalt Price	US$/lb	25.54		
Long Term Scandium Oxide Price[3]	US$/kg	N/A		1,500
Average Annual EBITDA	$M	1,514	1,775	2,096
Total EBITDA	**$M**	**37,843**	**44,376**	**52,391**
Average Annual Free Cash Flow (Pre-tax)	$M	1,383	1,645	1,915
Free Cash Flow (Pre-tax)[1]	$M	34,585	41,118	47,882
Total Provincial Tax (Including BC Mineral Tax)	**$M**	**(7,657)**	**(9,163)**	**(10,732)**
Total Federal Tax	**$M**	**(3,328)**	**(4,166)**	**(4,963)**
Total Taxes	**$M**	**(10,985)**	**(13,329)**	**(15,696)**
Average Annual Free Cash Flow (Post-tax)	$M	944	1,112	1,287
Free Cash Flow (Post-tax)[1]	$M	23,600	27,789	32,187
Total Free Cash Flow (Pre-tax)[2]	**$M**	**31,033**	**37,517**	**44,054**
Total Free Cash Flow (Post-tax)[2]	**$M**	**20,048**	**24,188**	**28,358**
NPV 5% (Pre-Tax)	$M	17,230	21,073	23,258
NPV 5% (Pre-Tax)	US$M	12,577	15,382	16,977
IRR (Pre-Tax)	%	43	50	40
Payback (Pre-Tax)	years	Year 3	Year 3	Year 3
NPV 5% (Post-Tax)	$M	11,047	13,526	14,848
NPV 5% (Post-Tax)	US$M	8,064	9,873	10,838
IRR (Post-Tax)	%	34	39	32
Payback (Post-Tax)	years	Year 3	Year 3	Year 4

Notes: [1] Free Cash Flow during production periods only.
[2] Total life of mine Free Cash Flow, including initial capital costs and closure
[3] Long Term Scandium Oxide price used for spot metal prices scenario

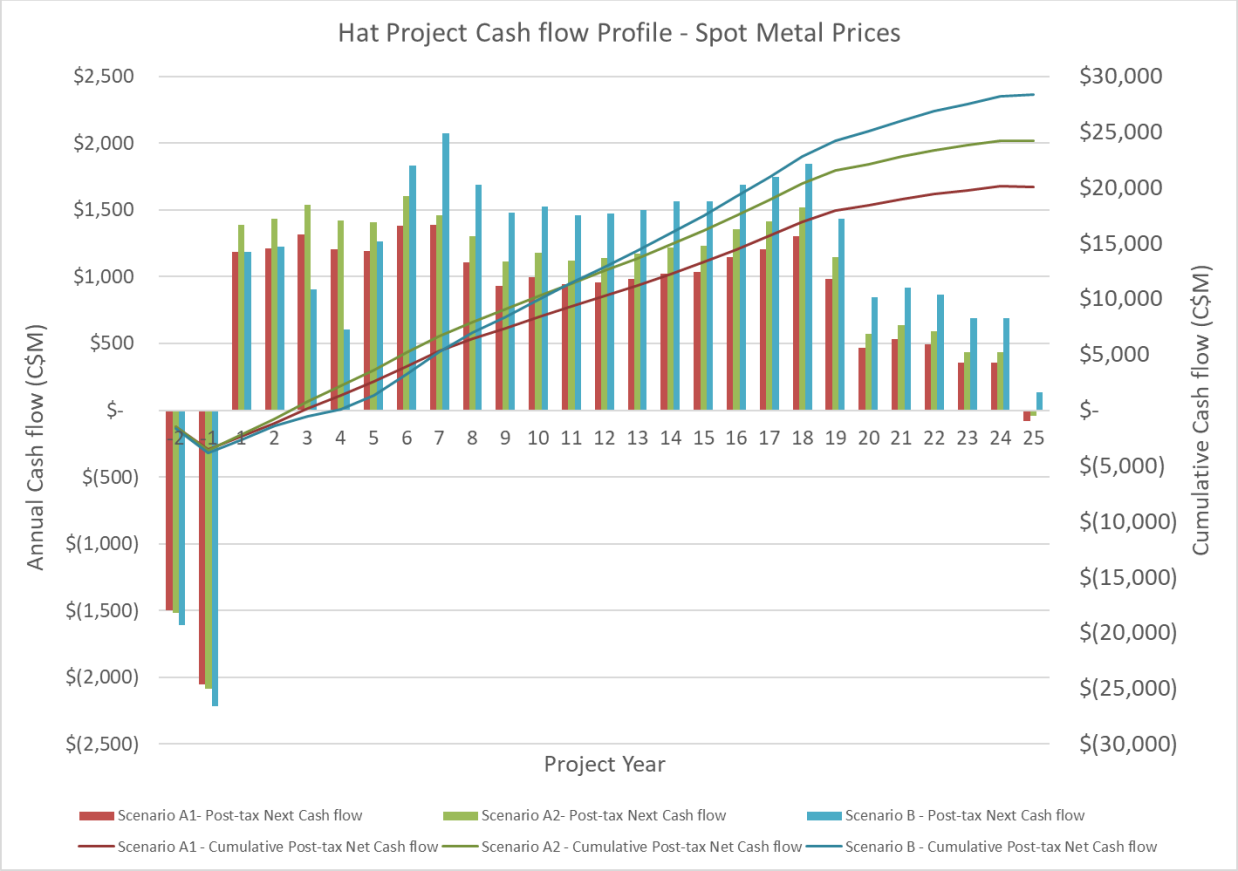


Figure 22-4: Annual and Cumulative Post-Tax Cashflow—Spot Metal prices

The annual breakdown of production, processing and economics is provided for Scenario A2 on Table 22-10 to Table 22-12, and for Scenario B on Table 22-13 to Table 22-15.

Table 22-10: Annual Production and Projected Cash Flow, Scenario A2—Part 1

	Inputs	Unit	Total/Avg.	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25
MINING																															
Open Pit																															
LOM	-	-	25	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Operating Days	350	d	8,750	-	-	-	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350
Tonnes Milled Per Day	120,000	t/d	-	-	-	-	106,281	120,637	119,890	120,560	119,944	121,169	120,309	119,736	120,718	121,678	121,098	120,440	120,288	120,238	120,842	120,386	119,851	120,387	85,297	-	-	-	-	-	-
Tonnes Moved Per Day		t/d	-	-	-	-	441,073	466,035	432,687	443,215	393,362	369,344	387,790	411,455	410,574	386,040	397,192	407,966	428,411	418,850	477,032	437,701	369,610	254,123	122,826	120,000	120,000	120,000	120,000	120,000	56,219
Ramping Up/Down Factor		%	-	-	-	-	89	101	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	71	100	100	100	100	100	100
Annual Production Ex-pit to Mill	42,000	'000 t	783,912	-	-		37,198	42,223	41,962	42,196	41,981	42,409	42,108	41,908	42,251	42,587	42,384	42,154	42,101	42,083	42,295	42,135	41,948	42,136	29,854	-	-	-	-	-	-
Waste		'000 t	1,643,615	-	-	60,000	89,400	101,447	92,322	96,019	83,290	77,277	83,602	85,930	84,743	80,347	84,026	87,201	94,068	94,319	113,885	101,499	81,473	41,887	10,882	-	-	-	-	-	-
Ore to Stockpile		'000 t	-	-	-	-	27,778	19,442	17,157	16,910	12,406	9,585	10,016	16,171	16,706	12,180	12,607	13,433	13,776	10,196	10,782	9,562	5,943	4,921	2,254	-	-	-	-	-	-
From Stockpile to Mill		'000 t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	12,146	42,000	42,000	42,000	42,000	42,000	19,677
Total Moved		**'000 t**	**2,839,026**	-	-	**60,000**	**154,375**	**163,112**	**151,440**	**155,125**	**137,677**	**129,270**	**135,727**	**144,009**	**143,701**	**135,114**	**139,017**	**142,788**	**149,944**	**146,597**	**166,961**	**153,195**	**129,364**	**88,943**	**42,989**	**42,000**	**42,000**	**42,000**	**42,000**	**42,000**	**19,677**
Stripping Ratio			1.60	-	-	-	1.38	1.65	1.56	1.62	1.53	1.49	1.60	1.48	1.44	1.47	1.53	1.57	1.68	1.80	2.15	1.96	1.70	0.89	0.34	-	-	-	-	-	-
PROCESSING																															
Mill Feed		'000 t	1,025,735	-	-	-	37,198	42,223	41,962	42,196	41,981	42,409	42,108	41,908	42,251	42,587	42,384	42,154	42,101	42,083	42,295	42,135	41,948	42,136	42,000	42,000	42,000	42,000	42,000	42,000	19,677
Au Grade		g/t	0.19	-	-	-	0.21	0.20	0.22	0.22	0.20	0.22	0.23	0.22	0.21	0.19	0.19	0.19	0.20	0.22	0.20	0.22	0.21	0.22	0.16	0.12	0.12	0.12	0.09	0.09	0.09
Ag Grade		g/t	0.51	-	-	-	0.47	0.52	0.49	0.40	0.41	0.41	0.37	0.33	0.33	0.41	0.41	0.42	0.43	0.41	0.36	0.35	0.40	0.34	0.37	0.35	0.35	0.35	0.33	0.33	0.33
Cu Grade		%	0.19%	-	-	-	0.19%	0.21%	0.21%	0.19%	0.21%	0.24%	0.23%	0.21%	0.20%	0.21%	0.21%	0.23%	0.23%	0.23%	0.23%	0.25%	0.26%	0.26%	0.21%	0.11%	0.11%	0.10%	0.09%	0.09%	0.09%
Co Grade		%	0.00777%	-	-	-	0.01%	0.01%	0.01%	0.01%	0.01%	0.01%	0.01%	0.01%	0.01%	0.01%	0.01%	0.01%	0.01%	0.01%	0.01%	0.01%	0.01%	0.01%	0.01%	0.01%	0.01%	0.01%	0.01%	0.01%	0.01%
Sc Grade		g/t	28.35	-	-	-	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35
Contained Metals																															
Contained Au		oz	6,103,387	-	-	-	251,189	277,737	297,045	293,357	276,365	305,669	310,273	302,186	282,770	263,643	255,251	255,489	264,449	292,258	277,848	296,718	284,360	303,330	217,697	164,441	164,441	156,059	125,911	125,911	58,988
Contained Ag		oz	12,795,387	-	-	-	565,872	709,156	655,695	537,600	548,195	560,494	506,816	442,528	449,700	561,379	562,931	572,364	576,802	560,329	486,835	471,788	544,615	466,473	502,713	478,315	478,315	470,008	440,133	440,133	206,198
Contained Cu		t	1,988,096	-	-	-	68,925	87,153	87,468	80,241	86,172	101,548	97,786	87,301	84,304	91,121	89,969	95,348	96,219	95,963	98,184	105,443	110,339	110,275	89,797	45,154	45,154	43,341	36,822	36,822	17,251
Contained Co		t	79,734	-	-	-	2,691	3,226	3,371	3,399	3,505	3,730	3,626	3,520	3,546	3,655	3,589	3,469	3,355	3,258	3,063	3,088	3,358	2,723	2,877	3,059	3,059	3,056	3,043	3,043	1,426
Contained Sc (Metal)		kg	29,079,585	-	-	-	1,054,568	1,197,017	1,189,609	1,196,253	1,190,149	1,202,296	1,193,770	1,188,084	1,197,825	1,207,352	1,201,599	1,195,066	1,193,555	1,193,058	1,199,055	1,194,529	1,189,225	1,194,543	1,190,700	1,190,700	1,190,700	1,190,700	1,190,700	1,190,700	1,190,700	557,831
Copper Concentrate																															
Au Recovery		%	75	-	-	-	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75
Ag Recovery		%	53	-	-	-	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53
Cu Recovery		%	89	-	-	-	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89
Co Recovery		%	30	-	-	-	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30
Sc Recovery		%	0	-	-	-	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Secondary Product Stream				-	-	-																									
Au Recovery		%	0	-	-	-	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Ag Recovery		%	0	-	-	-	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Cu Recovery		%	0	-	-	-	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Co Recovery		%	0	-	-	-	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Sc Recovery	72	%	72	-	-	-	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72
Net Recovery				-	-	-																									
Au Recovery		%	75	-	-	-	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75
Ag Recovery		%	53	-	-	-	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53
Cu Recovery		%	89	-	-	-	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89
Co Recovery		%	30	-	-	-	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30
Sc Recovery		%	72	-	-	-	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72
Copper Concentrate				-	-	-																									
Au		oz	4,577,540	-	-	-	188,392	208,303	222,784	220,018	207,274	229,252	232,705	226,640	212,078	197,732	191,438	191,617	198,337	219,193	208,386	222,539	213,270	227,498	163,273	123,331	123,331	117,044	94,434	94,434	44,241
Ag		oz	6,781,555	-	-	-	299,912	375,853	347,519	284,928	290,543	297,062	268,613	234,540	238,341	297,531	298,353	303,353	305,705	296,975	258,023	250,048	288,646	247,231	266,438	253,507	253,507	249,104	233,270	233,270	109,285
Cu		t	1,769,406	-	-	-	61,343	77,566	77,846	71,414	76,693	90,377	87,029	77,697	75,031	81,097	80,073	84,860	85,634	85,407	87,383	93,844	98,202	98,144	79,919	40,187	40,187	38,574	32,771	32,771	15,353
Co		t	23,920	-	-	-	807	968	1,011	1,020	1,051	1,119	1,088	1,056	1,064	1,096	1,077	1,041	1,006	977	919	927	1,007	817	863	918	918	917	913	913	428
Sc		kg	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Secondary Product Stream ("Pyrite Concentrate")																															
Au		oz	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ag		oz	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Cu		t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Co		t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sc to Sc_2O3 Conversion Factor	1.534		-	-	-	-	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534
Overall Mass % of Scandium Processing	12.5	%	-	-	-	-	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5
Sc_2O_3 Available for Recovery		kg	4,014,299	-	-	-	145,578	165,243	164,220	165,137	164,294	165,971	164,794	164,009	165,354	166,669	165,875	164,973	164,765	164,696	165,524	164,899	164,167	164,901	164,371	164,371	164,371	164,371	164,371	164,371	77,006
Copper Concentrate		t	-	-	-	-	245,372	310,266	311,386	285,657	306,771	361,509	348,117	310,790	300,122	324,389	320,291	339,440	342,538	341,630	349,533	375,377	392,806	392,577	319,677	160,749	160,749	154,295	131,085	131,085	61,412
Au Grade in Concentrate		g/t	-	-	-	-	23.9	20.9	22.3	24.0	21.0	19.7	20.8	22.7	22.0	19.0	18.6	17.6	18.0	20.0	18.5	18.4	16.9	18.0	15.9	23.9	23.9	23.6	22.4	22.4	22.4
Ag Grade in Concentrate		g/t	-	-	-	-	38.0	37.7	34.7	31.0	29.5	25.6	24.0	23.5	24.7	28.5	29.0	27.8	27.8	27.0	23.0	20.7	22.9	19.6	25.9	49.1	49.1	50.2	55.3	55.3	55.3
Cu Grade in Concentrate	25	%	-	-	-	-	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25
Co Grade in Concentrate		%	-	-	-	-	0.33	0.31	0.32	0.36	0.34	0.31	0.31	0.34	0.35	0.34	0.34	0.31	0.29	0.29	0.26	0.25	0.26	0.21	0.27	0.57	0.57	0.59	0.70	0.70	0.70
Sc Grade in Concentrate			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Secondary Product Stream ("Co-Pyrite Concentrate") Produced		t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Au Grade in Concentrate		g/t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ag Grade in Concentrate		g/t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Cu Grade in Concentrate		%	-	-	-	-	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Co Grade in Concentrate	0.25	%	-	-	-	-	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			-	-	-	-																									
Total Recovered			-	-	-	-																									
Au		**oz**	**4,577,540**	-	-	-	**188,392**	**208,303**	**222,784**	**220,018**	**207,274**	**229,252**	**232,705**	**226,640**	**212,078**	**197,732**	**191,438**	**191,617**	**198,337**	**219,193**	**208,386**	**222,539**	**213,270**	**227,498**	**163,273**	**123,331**	**123,331**	**117,044**	**94,434**	**94,434**	**44,241**
Ag		**oz**	**6,781,555**	-	-	-	**299,912**	**375,853**	**347,519**	**284,928**	**290,543**	**297,062**	**268,613**	**234,540**	**238,341**	**297,531**	**298,353**	**303,353**	**305,705**	**296,975**	**258,023**	**250,048**	**288,646**	**247,231**	**266,438**	**253,507**	**253,507**	**249,104**	**233,270**	**233,270**	**109,285**
Cu		**t**	**1,769,406**	-	-	-	**61,343**	**77,566**	**77,846**	**71,414**	**76,693**	**90,377**	**87,029**	**77,697**	**75,031**	**81,097**	**80,073**	**84,860**	**85,634**	**85,407**	**87,383**	**93,844**	**98,202**	**98,144**	**79,919**	**40,187**	**40,187**	**38,574**	**32,771**	**32,771**	**15,353**
Co		**t**	**23,920**	-	-	-	**807**	**968**	**1,011**	**1,020**	**1,051**	**1,119**	**1,088**	**1,056**	**1,064**	**1,096**	**1,077**	**1,041**	**1,006**	**977**	**919**	**927**	**1,007**	**817**	**863**	**918**	**918**	**917**	**913**	**913**	**428**
Sc_2O_3		**kg**	**-**	-	-	-	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**

Table 22-11: Annual Production and Projected Cash Flow, Scenario A2—Part 2

	Inputs	Unit	Total/Avg.	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25
REVENUE																															
Metal Prices		C$					C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$
Au	3,272.60	US$/oz Au	-	-	-	-	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483
Ag	50.22	US/oz Ag	-	-	-	-	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8
Cu	4.88	US/lb Cu	-	-	-	-	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7
Co	19.57	US/lb Co	-	-	-	-	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8
Sc	1,500.00	US/kg Sc	-	-	-	-	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055
Exchange Rate	1.37	CAD:USD	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37
Copper Concentrate Payable %																															
Au		%	-	-	-	-	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0
Ag		%	-	-	-	-	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0
Cu		%	-	-	-	-	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0
Co		%	-	-	-	-	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0
Sc		%	-	-	-	-	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Secondary Product Stream Payable %																															
Au		%	Gold/Silver Dore	-			97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0
Ag		%	Gold/Silver Dore	-			97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0
Cu		%	Reported Back to Cu Concentrate	-			0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Co		%	-	-			97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0
Scandium Processing Payable %																															
Scandium Ramp Up		%	-	-	-	-	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Sc	95	%	-	-	-	-	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0
Copper Concentrate Payable																															
Au		oz	4,348,663	-	-	-	178,972	197,888	211,645	209,017	196,910	217,789	221,069	215,308	201,474	187,846	181,866	182,036	188,420	208,234	197,967	211,412	202,606	216,123	155,109	117,164	117,164	111,192	89,712	89,712	42,029
Ag		oz	6,103,400	-	-	-	269,921	338,267	312,767	256,435	261,489	267,356	241,751	211,086	214,507	267,778	268,518	273,018	275,134	267,277	232,220	225,043	259,782	222,508	239,794	228,156	228,156	224,194	209,943	209,943	98,356
Cu		t	1,716,323	-	-	-	59,503	75,239	75,511	69,272	74,392	87,666	84,418	75,367	72,780	78,664	77,671	82,314	83,065	82,845	84,762	91,029	95,256	95,200	77,522	38,982	38,982	37,417	31,788	31,788	14,892
Co		t	19,136	-	-	-	646	774	809	816	841	895	870	845	851	877	861	832	805	782	735	741	806	653	690	734	734	733	730	730	342
Sc		kg	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Secondary Product Stream Payable																															
Au		oz	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ag		oz	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Cu		t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Co		t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Payable Scandium																															
Sc_2O_3		kg	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Gross Revenue																															
Au Gross Revenue		C$ '000	19,497,067	-	-	-	802,415	887,221	948,900	937,121	882,840	976,450	991,156	965,324	903,299	842,200	815,391	816,150	844,773	933,608	887,576	947,857	908,378	968,979	695,427	525,302	525,302	498,524	402,220	402,220	188,436
Ag Gross Revenue		C$ '000	419,922	-	-	-	18,571	23,273	21,519	17,643	17,991	18,394	16,633	14,523	14,758	18,423	18,474	18,784	18,930	18,389	15,977	15,483	17,873	15,309	16,498	15,697	15,697	15,425	14,444	14,444	6,767
Cu Gross Revenue		C$ '000	25,297,018	-	-	-	877,015	1,108,960	1,112,962	1,021,002	1,096,470	1,292,115	1,244,248	1,110,833	1,072,704	1,159,440	1,144,793	1,213,234	1,224,308	1,221,061	1,249,311	1,341,682	1,403,979	1,403,160	1,142,596	574,553	574,553	551,485	468,526	468,526	219,500
Co Gross Revenue		C$ '000	1,131,089	-	-	-	38,174	45,765	47,827	48,213	49,716	52,910	51,437	49,933	50,309	51,849	50,919	49,205	47,589	46,216	43,445	43,812	47,632	38,622	40,814	43,394	43,394	43,346	43,171	43,171	20,225
Sc Gross Revenue		C$ '000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Gross Revenue		**C$ '000**	**46,345,096**	-	-	-	**1,736,175**	**2,065,219**	**2,131,208**	**2,023,979**	**2,047,016**	**2,339,869**	**2,303,475**	**2,140,613**	**2,041,071**	**2,071,912**	**2,029,578**	**2,097,374**	**2,135,599**	**2,219,274**	**2,196,309**	**2,348,834**	**2,377,862**	**2,426,070**	**1,895,334**	**1,158,947**	**1,158,947**	**1,108,780**	**928,361**	**928,361**	**434,928**
Transport																															
Moisture Content	10	%		-	-	-	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10
Cu Concentrate	95.90 C$/dmt	C$ '000	746,618	-	-	-	25,884	32,730	32,848	30,134	32,361	38,136	36,723	32,785	31,660	34,220	33,788	35,808	36,134	36,039	36,872	39,599	41,437	41,413	33,723	16,957	16,957	16,277	13,828	13,828	6,478
Treatment																															
Cu Concentrate	77.00 US$/dmt	C$ '000	746,618	-	-	-	25,884	32,730	32,848	30,134	32,361	38,136	36,723	32,785	31,660	34,220	33,788	35,808	36,134	36,039	36,872	39,599	41,437	41,413	33,723	16,957	16,957	16,277	13,828	13,828	6,478
Refining cost																															
Au	5.00 US$/oz Au	C$ '000	29,788	-	-	-	1,226	1,356	1,450	1,432	1,349	1,492	1,514	1,475	1,380	1,287	1,246	1,247	1,291	1,426	1,356	1,448	1,388	1,480	1,062	803	803	762	615	615	288
Ag	0.35 US$/oz Ag	C$ '000	2,927	-	-	-	129	162	150	123	125	128	116	101	103	128	129	131	132	128	111	108	125	107	115	109	109	108	101	101	47
Cu	0.08 US/lb Cu	C$ '000	414,705	-	-	-	14,377	18,180	18,245	16,738	17,975	21,182	20,398	18,210	17,585	19,007	18,767	19,889	20,071	20,017	20,481	21,995	23,016	23,003	18,731	9,419	9,419	9,041	7,681	7,681	3,598
Co	0.12 US$/lb Co	C$ '000	6,936	-	-	-	234	281	293	296	305	324	315	306	308	318	312	302	292	283	266	269	292	237	250	266	266	266	265	265	124
Total Charges		**C$ '000**	**1,947,593**	-	-	-	**67,735**	**85,438**	**85,834**	**78,856**	**84,477**	**99,398**	**95,789**	**85,663**	**82,696**	**89,180**	**88,029**	**93,184**	**94,054**	**93,932**	**95,959**	**103,017**	**107,695**	**107,653**	**87,604**	**44,512**	**44,512**	**42,729**	**36,317**	**36,317**	**17,014**
Gross Less Charge		C$ '000	44,397,504	-	-	-	1,668,439	1,979,781	2,045,374	1,945,123	1,962,540	2,240,471	2,207,686	2,054,950	1,958,374	1,982,732	1,941,549	2,004,190	2,041,545	2,125,342	2,100,350	2,245,818	2,270,167	2,318,417	1,807,730	1,114,436	1,114,436	1,066,051	892,045	892,045	417,914
Royalty 2% NSR	2%	C$ '000	887,950	-	-	-	33,369	39,596	40,907	38,902	39,251	44,809	44,154	41,099	39,167	39,655	38,831	40,084	40,831	42,507	42,007	44,916	45,403	46,368	36,155	22,289	22,289	21,321	17,841	17,841	8,358
Net Smelter Return		**C$ '000**	**43,509,553**	-	-	-	**1,635,071**	**1,940,186**	**2,004,466**	**1,906,221**	**1,923,289**	**2,195,662**	**2,163,532**	**2,013,851**	**1,919,207**	**1,943,077**	**1,902,718**	**1,964,106**	**2,000,714**	**2,082,835**	**2,058,343**	**2,200,901**	**2,224,764**	**2,272,049**	**1,771,576**	**1,092,147**	**1,092,147**	**1,044,730**	**874,204**	**874,204**	**409,556**
Unit NSR		**C$/t milled**	**42.42**	-	-	-	**43.96**	**45.95**	**47.77**	**45.18**	**45.81**	**51.77**	**51.38**	**48.05**	**45.42**	**45.63**	**44.89**	**46.59**	**47.52**	**49.49**	**48.67**	**52.23**	**53.04**	**53.92**	**42.18**	**26.00**	**26.00**	**24.87**	**20.81**	**20.81**	**20.81**
Revenue by Metal																															
Net Revenue by Metal																															
Au		%	44	-	-	-	48	45	46	48	45	44	45	47	46	42	42	41	41	44	42	42	40	42	38	47	47	47	45	45	45
Ag		%	1	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	2	2	2
Cu		%	52	-	-	-	49	52	50	49	52	53	52	50	51	54	55	56	55	53	55	55	57	56	58	48	48	48	49	49	49
Co		%	3	-	-	-	2	2	2	2	3	2	2	2	3	3	3	2	2	2	2	2	2	2	2	4	4	4	5	5	5
Sc		%	0	-	-	-	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Revenue per Metal Unit (NSR Factor)																															
Au		per g Au	-	-	-	-	134.00	134.00	134.00	134.00	134.00	134.00	134.00	134.00	134.00	134.00	134.00	134.00	134.00	134.00	134.00	134.00	134.00	134.00	134.00	134.00	134.00	134.00	134.00	134.00	134.00
Ag		per g Ag	-	-	-	-	1.94	1.94	1.94	1.94	1.94	1.94	1.94	1.94	1.94	1.94	1.94	1.94	1.94	1.94	1.94	1.94	1.94	1.94	1.94	1.94	1.94	1.94	1.94	1.94	1.94
Cu		per lb Cu	-	-	-	-	5.88	5.88	5.88	5.88	5.88	5.88	5.88	5.88	5.88	5.88	5.88	5.88	5.88	5.88	5.88	5.88	5.88	5.88	5.88	5.88	5.88	5.88	5.88	5.88	5.88
Co		per lb Co	-	-	-	-	20.89	20.89	20.89	20.89	20.89	20.89	20.89	20.89	20.89	20.89	20.89	20.89	20.89	20.89	20.89	20.89	20.89	20.89	20.89	20.89	20.89	20.89	20.89	20.89	20.89
Sc		per kg Sc	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

Table 22-12: Annual Production and Projected Cash Flow, Scenario A2—Part 3

	Inputs	Unit	Total/Avg.	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25
OPERATING COST																															
Mining (Open Pit)		C/t moved	2.32	-	-	-	1.89	1.93	1.97	2.01	2.05	2.09	2.12	2.16	2.20	2.24	2.28	2.32	2.35	2.39	2.43	2.47	2.51	2.54	2.58	2.58	2.58	2.58	2.58	2.58	2.58
Mining (Rehandle)	.81/t moved	C/t moved		-	-	-	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81
Processing	7.93/t milled	C/t milled	7.93	-	-	-	7.93	7.93	7.93	7.93	7.93	7.93	7.93	7.93	7.93	7.93	7.93	7.93	7.93	7.93	7.93	7.93	7.93	7.93	7.93	7.93	7.93	7.93	7.93	7.93	7.93
Processing Co-Au-Acid Plant	3.083/t milled	C/t milled		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Scandium Processing	685.80 US$/kg Sc	US$/kg Sc_2O_3/ C$/kg Sc_2O_3	939.55	-	-	-	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55
G&A and Tailings	2.56/t milled	C/t milled	2.56	-	-	-	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56
TOTAL OPERATING COST																															
Mining (Open Pit)		C$ '000	5,980,015	-	-	-	292,476	315,268	298,501	311,698	281,905	269,636	288,295	311,397	316,227	302,499	316,556	330,604	352,908	350,639	405,733	378,141	324,264	226,348	120,884	34,020	34,020	34,020	34,020	34,020	15,938
Processing		C$ '000	8,138,181	-	-	-	295,130	334,996	332,923	334,782	333,074	336,473	334,087	332,496	335,222	337,888	336,278	334,450	334,027	333,888	335,566	334,300	332,815	334,303	333,228	333,228	333,228	333,228	333,228	333,228	156,114
Processing (Scandium)		C$ '000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
G&A and Tailings		C$ '000	2,621,778	-	-	-	95,079	107,922	107,254	107,853	107,302	108,397	107,629	107,116	107,994	108,853	108,335	107,746	107,609	107,565	108,105	107,697	107,219	107,698	107,352	107,352	107,352	107,352	107,352	107,352	50,293
Total Operating Cost		**C$ '000**	**16,739,974**	-	-	-	**682,684**	**758,185**	**738,677**	**754,332**	**722,281**	**714,507**	**730,011**	**751,009**	**759,443**	**749,241**	**761,169**	**772,799**	**794,544**	**792,092**	**849,405**	**820,138**	**764,298**	**668,350**	**561,464**	**474,600**	**474,600**	**474,600**	**474,600**	**474,600**	**222,345**
Unit Operating Cost		**C/t Milled**	**16.22**	-	-	-	**18.4**	**18.0**	**17.6**	**17.9**	**17.2**	**16.8**	**17.3**	**17.9**	**18.0**	**17.6**	**18.0**	**18.3**	**18.9**	**18.8**	**20.1**	**19.5**	**18.2**	**15.9**	**13.4**	**11.3**	**11.3**	**11.3**	**11.3**	**11.3**	**11.3**
C1 Cash Cost		C/lb-CuEq Produced	2.41	-	-	-	2.51	2.35	2.21	2.38	2.25	1.95	2.03	2.24	2.38	2.31	2.40	2.36	2.38	2.28	2.48	2.24	2.06	1.76	1.90	2.61	2.61	2.72	3.25	3.25	3.25
Operating Cashflow EBITDA		**C$ '000**	**26,769,579**				**952,386**	**1,182,001**	**1,265,789**	**1,151,888**	**1,201,008**	**1,481,155**	**1,433,521**	**1,262,842**	**1,159,764**	**1,193,837**	**1,141,549**	**1,191,307**	**1,206,170**	**1,290,743**	**1,208,938**	**1,380,764**	**1,460,466**	**1,603,699**	**1,210,112**	**617,547**	**617,547**	**570,130**	**399,604**	**399,604**	**187,210**
CAPITAL COST																															
Direct Cost																															
Mining		C$ '000	683,132	-	232,481	161,224	-	39,159	19,580	9,886	20,088	29,369	121	3,616	2,752	24,699	42,368	43,369	29,369	10,798	968	13,286	-	-	-	-	-	-	-	-	-
Mining Pre-stripping		C$ '000	97,200	-		97,200	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Processing and Processing Infrastructure		C$ '000	1,645,238	-	658,095	987,143	-	-	-	-	-	-	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
General Infrastructure		C$ '000	183,619	-	73,448	110,171	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Tailings and Site Water Management		C$ '000	1,221,858	-		156,721	36,403	89,501	41,340	41,485	41,352	41,616	41,431	41,307	41,519	41,726	41,601	41,459	41,426	41,415	41,546	41,447	41,332	41,448	41,364	41,364	41,364	41,364	41,364	41,364	27,598
Power Line		C$ '000	205,520	-	140,000	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520
Total Direct Cost		**C$ '000**	**4,036,567**	-	**1,104,023**	**1,514,980**	**38,923**	**131,181**	**63,440**	**53,891**	**63,960**	**73,506**	**44,072**	**47,443**	**46,791**	**68,945**	**86,489**	**87,348**	**73,316**	**54,733**	**45,034**	**57,253**	**43,852**	**43,968**	**43,884**	**43,884**	**43,884**	**43,884**	**43,884**	**43,884**	**30,118**
Other Costs																															
EPCM / Owners / Indirect Cost	10	C$ '000	403,657	-	110,402	151,498	3,892	13,118	6,344	5,389	6,396	7,351	4,407	4,744	4,679	6,894	8,649	8,735	7,332	5,473	4,503	5,725	4,385	4,397	4,388	4,388	4,388	4,388	4,388	4,388	3,012
Subtotal Costs		**C$ '000**	**4,440,223**	-	**1,214,426**	**1,666,478**	**42,815**	**144,299**	**69,784**	**59,280**	**70,356**	**80,856**	**48,479**	**52,188**	**51,470**	**75,839**	**95,138**	**96,083**	**80,647**	**60,206**	**49,537**	**62,978**	**48,237**	**48,364**	**48,272**	**48,272**	**48,272**	**48,272**	**48,272**	**48,272**	**33,130**
Contingency	25	C$ '000	1,110,056	-	303,606	416,619	10,704	36,075	17,446	14,820	17,589	20,214	12,120	13,047	12,867	18,960	23,784	24,021	20,162	15,052	12,384	15,745	12,059	12,091	12,068	12,068	12,068	12,068	12,068	12,068	8,282
Subtotal Cost+ Contingency		C$ '000	**5,550,279**	-	**1,518,032**	**2,083,097**	**53,519**	**180,373**	**87,230**	**74,100**	**87,945**	**101,070**	**60,599**	**65,235**	**64,337**	**94,799**	**118,922**	**120,104**	**100,809**	**75,258**	**61,921**	**78,723**	**60,296**	**60,455**	**60,341**	**60,341**	**60,341**	**60,341**	**60,341**	**60,341**	**41,412**
Sustaining Cost																															
Sustaining Mining (LCC)		C$ '000	521,152	-	-	-	5,809	10,008	19,475	19,733	32,066	60,033	19,132	19,885	40,690	26,429	9,973	33,820	42,697	26,943	23,234	30,489	34,530	37,961	18,974	1,854	1,854	1,854	1,854	1,854	
Sustaining Processing and Infrastructure		C$ '000	285,000	-	-	-	-	-	-	-	-	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	
Closure Cost																															
Reclamation and closure		C$ '000	503,300	-	-	-	-	-	-	-	-	-	-	-	75,950	-	-	-	-	75,950	-	-	-	-	-	75,950	-	-	-	-	275,450
Total Capital Cost		**C$ '000**	**6,859,731**	-	**1,518,032**	**2,083,097**	**59,328**	**190,382**	**106,705**	**93,833**	**120,011**	**176,103**	**94,731**	**100,119**	**195,977**	**136,229**	**143,894**	**168,924**	**158,506**	**193,151**	**100,155**	**124,211**	**109,827**	**113,417**	**94,315**	**153,145**	**77,195**	**77,195**	**77,195**	**77,195**	**316,862**
Initial Capital Cost		**C$ '000**	**3,601,129**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital Cost and Closure		**C$ '000**	**3,258,602**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
AISC Cash Cost		**C/lb-CuEq Produced**	**2.80**	-	-	-	**2.73**	**2.94**	**2.53**	**2.68**	**2.63**	**2.43**	**2.29**	**2.54**	**2.75**	**2.73**	**2.85**	**2.87**	**2.85**	**2.62**	**2.77**	**2.57**	**2.35**	**2.06**	**2.22**	**3.03**	**3.03**	**3.17**	**3.78**	**3.78**	**3.85**
PRE-TAX CASH FLOW																															
Net Pre-Tax Cashflow		C$ '000	19,909,848	-	(1,518,032)	(2,083,097)	893,058	991,619	1,159,085	1,058,055	1,080,997	1,305,052	1,338,790	1,162,723	963,787	1,057,608	997,655	1,022,384	1,047,664	1,097,592	1,108,784	1,256,553	1,350,639	1,490,282	1,115,797	464,402	540,352	492,935	322,409	322,409	(129,652)
Cumulative Pre-Tax Cashflow		C$ '000		-	(1,518,032)	(3,601,129)	(2,708,072)	(1,716,452)	(557,368)	500,687	1,581,685	2,886,737	4,225,527	5,388,250	6,352,037	7,409,645	8,407,299	9,429,683	10,477,347	11,574,939	12,683,722	13,940,275	15,290,914	16,781,196	17,896,993	18,361,395	18,901,747	19,394,682	19,717,091	20,039,500	19,909,848
Discounted Cashflow NPV	5	C$ '000	**10,576,145**	-			850,531	899,428	1,001,261	870,465	846,990	973,850	951,453	786,977	621,266	649,280	583,308	569,301	555,598	554,359	533,344	575,641	589,279	619,243	441,559	175,028	193,955	168,510	104,967	99,969	(38,287)
Net Pre-Tax Cashflow		US$ '000	14,532,736	-	(1,108,053)	(1,520,509)	651,867	723,810	846,047	772,303	789,049	952,593	977,219	848,703	703,494	771,977	728,215	746,265	764,718	801,162	809,331	917,192	985,868	1,087,797	814,450	338,980	394,418	359,807	235,335	235,335	(94,636)
Cumulative Pre-Tax Cashflow		US$ '000		-	(1,108,053)	(2,628,562)	(1,976,695)	(1,252,885)	(406,838)	365,465	1,154,514	2,107,107	3,084,326	3,933,029	4,636,523	5,408,500	6,136,715	6,882,980	7,647,698	8,448,860	9,258,192	10,175,383	11,161,251	12,249,048	13,063,499	13,402,478	13,796,896	14,156,702	14,392,037	14,627,372	14,532,736
Discounted Cashflow NPV	5	US$ '000	**7,719,814**	-	-	-	620,826	656,517	730,847	635,376	618,241	710,839	694,491	574,436	453,479	473,927	425,772	415,548	405,546	404,641	389,302	420,176	430,131	452,002	322,306	127,758	141,573	123,000	76,618	72,970	(27,946)
Payback Year Pre-Tax		**Year 4**																													
TAXES (Calculation done in "Income Tax" and "Mineral Tax Sheet")																															
Federal Tax After CTM Credit		C$ '000	(1,859,027)	-	-	-	(12,253)	(10,039)	(18,742)	(20,365)	(26,467)	(39,407)	(39,299)	(35,318)	(33,585)	(35,321)	(69,545)	(133,905)	(135,838)	(148,421)	(137,802)	(162,141)	(172,983)	(192,571)	(141,814)	(67,147)	(67,005)	(61,840)	(40,407)	(40,972)	(15,837)
Provincial Tax		C$ '000	(2,118,139)	-	-	-	(36,306)	(29,745)	(55,533)	(60,342)	(78,421)	(116,761)	(116,442)	(104,646)	(99,511)	(104,654)	(100,829)	(107,124)	(108,671)	(118,737)	(110,241)	(129,713)	(138,387)	(154,057)	(113,451)	(53,718)	(53,604)	(49,472)	(32,326)	(32,777)	(12,670)
BC$ Mineral Taxes		C$ '000	(2,971,362)	-	-	-	(19,715)	(109,462)	(151,920)	(139,090)	(140,969)	(168,358)	(176,229)	(152,707)	(125,720)	(137,270)	(129,032)	(131,312)	(135,152)	(143,673)	(145,811)	(164,344)	(177,269)	(195,391)	(146,218)	(60,485)	(70,359)	(64,069)	(41,448)	(41,448)	(3,911)
Total Taxes		**C$ '000**	**(6,948,528)**	-	-	-	**(68,274)**	**(149,246)**	**(226,195)**	**(219,797)**	**(245,858)**	**(324,526)**	**(331,970)**	**(292,671)**	**(258,817)**	**(277,244)**	**(299,406)**	**(372,341)**	**(379,661)**	**(410,831)**	**(393,854)**	**(456,197)**	**(488,639)**	**(542,018)**	**(401,484)**	**(181,351)**	**(190,968)**	**(175,381)**	**(114,181)**	**(115,197)**	**(32,419)**
AFTER-TAX CASH FLOW																															
After-Tax Cashflow		C$ '000	12,961,321	-	(1,518,032)	(2,083,097)	824,784	842,373	932,889	838,258	835,140	980,525	1,006,820	870,052	704,970	780,363	698,249	650,042	668,002	686,761	714,930	800,355	862,000	948,264	714,313	283,052	349,384	317,554	208,228	207,212	(162,070)
Cumulative After-Tax Cashflow		C$ '000		-	(1,518,032)	(3,601,129)	(2,776,345)	(1,933,972)	(1,001,083)	(162,825)	672,315	1,652,840	2,659,661	3,529,712	4,234,683	5,015,046	5,713,295	6,363,337	7,031,340	7,718,101	8,433,030	9,233,386	10,095,385	11,043,649	11,757,962	12,041,014	12,390,398	12,707,951	12,916,179	13,123,391	12,961,321
Discounted Cashflow NPV	5%	C$ '000	**6,727,392**	-	-	-	785,509	764,057	805,865	689,637	654,354	731,683	715,528	588,885	454,430	479,075	408,252	361,968	354,256	346,861	343,893	366,652	376,088	394,023	282,678	106,679	125,409	108,556	67,793	64,250	(47,860)
After-Tax Cashflow		US$ '000	9,460,818	-	(1,108,053)	(1,520,509)	602,032	614,871	680,941	611,867	609,591	715,712	734,905	635,074	514,577	569,608	509,671	474,483	487,593	501,285	521,847	584,201	629,197	692,163	521,396	206,607	255,025	231,791	151,991	151,249	(118,300)
Cumulative After-Tax Cashflow		US$ '000		-	(1,108,053)	(2,628,562)	(2,026,529)	(1,411,659)	(730,718)	(118,850)	490,741	1,206,453	1,941,358	2,576,432	3,091,009	3,660,618	4,170,288	4,644,772	5,132,365	5,633,650	6,155,497	6,739,698	7,368,894	8,061,058	8,582,454	8,789,061	9,044,086	9,275,877	9,427,868	9,579,118	9,460,818
Discounted Cashflow NPV	5%	US$ '000	**4,910,505**	-	-	-	573,364	557,706	588,223	503,385	477,631	534,075	522,284	429,843	331,701	349,690	297,994	264,210	258,581	253,183	251,017	267,629	274,516	287,608	206,334	77,868	91,539	79,238	49,484	46,898	(34,934)
Payback Year		**Year 5**																													
PROJECT ECONOMICS																															
Pre-Tax IRR		%	29			%	29																								
Pre-tax NPV at 5% Discounting	5%	C$ '000	10,576,145			US$ '000	**7,719,814**																								
Pre-tax NPV at 8% Discounting	7.5%	C$ '000	7,851,142			US$ '000	5,730,761																								
Pre-tax NPV at 10% Discounting	10%	C$ '000	5,861,477			US$ '000	4,278,451																								
After-Tax IRR		%	23			%	23																								
After-Tax NPV at 5% Discounting	5%	C$ '000	6,727,392			US$ '000	**4,910,505**																								
After-Tax NPV at 8% Discounting	8%	C$ '000	4,871,578			US$ '000	3,555,896																								
After-tax NPV at 10% Discounting	10%	C$ '000	3,499,754			US$ '000	2,554,565																								

Table 22-13: Annual Production and Projected Cash Flow, Scenario B—Part 1

	Inputs	Unit	Total/Avg	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25
MINING																															
Open Pit																															
LOM			25	-	-	-																									
Operating Days	350	d	8,750	-	-	-	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350
Tonnes milled per day	120,000	t/d		-	-	-	106,281	120,637	119,890	120,560	119,944	121,169	120,309	119,736	120,718	121,678	121,098	120,440	120,288	120,238	120,842	120,386	119,851	120,387	85,297	-	-	-	-	-	-	-
Tonnes moved per day		/d		-	-	-	441,073	466,035	432,687	443,215	393,362	369,344	387,790	411,455	410,574	386,040	397,192	407,966	428,411	418,850	477,032	437,701	369,610	254,123	122,826	120,000	120,000	120,000	120,000	120,000	56,219
Ramping Up/Down Factor				-	-	-	89	101	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	71	100	100	100	100	100	100
Annual Production Ex-pit to Mill	42,000	'000 t	783,912	-	-	-	37,198	42,223	41,962	42,196	41,981	42,409	42,108	41,908	42,251	42,587	42,384	42,154	42,101	42,083	42,295	42,135	41,948	42,136	29,854	-	-	-	-	-	-
Waste		'000 t	1,643,615	-	-	60,000	89,400	101,447	92,322	96,019	83,290	77,277	83,602	85,930	84,743	80,347	84,026	87,201	94,068	94,319	113,885	101,499	81,473	41,887	10,882	-	-	-	-	-	-
Ore to Stockpile		'000 t		-	-	-	27,778	19,442	17,157	16,910	12,406	9,585	10,016	16,171	16,706	12,180	12,607	13,433	13,776	10,196	10,782	9,562	5,943	4,921	2,254	-	-	-	-	-	-
From Stockpile to Mill		'000 t		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	12,146	42,000	42,000	42,000	42,000	42,000	19,677
Total Moved		'000 t	2,839,026	-	-	60,000	154,375	163,112	151,440	155,125	137,677	129,270	135,727	144,009	143,701	135,114	139,017	142,788	149,944	146,597	166,961	153,195	129,364	88,943	42,989	42,000	42,000	42,000	42,000	42,000	19,677
Stripping Ratio			1.60	-	-	-	1.38	1.65	1.56	1.62	1.53	1.49	1.60	1.48	1.44	1.47	1.53	1.57	1.68	1.80	2.15	1.96	1.70	0.89	0.34	-	-	-	-	-	-
PROCESSING				-	-	-																									
Mill Feed		'000 t	1,025,735	-	-	-	37,198	42,223	41,962	42,196	41,981	42,409	42,108	41,908	42,251	42,587	42,384	42,154	42,101	42,083	42,295	42,135	41,948	42,136	42,000	42,000	42,000	42,000	42,000	42,000	19,677
Au Grade		g/t	0.19	-	-	-	0.21	0.20	0.22	0.22	0.20	0.22	0.23	0.22	0.21	0.19	0.19	0.19	0.20	0.22	0.20	0.22	0.21	0.22	0.16	0.12	0.12	0.12	0.09	0.09	0.09
Ag Grade		g/t	0.51	-	-	-	0.47	0.52	0.49	0.40	0.41	0.41	0.37	0.33	0.33	0.41	0.41	0.42	0.43	0.41	0.36	0.35	0.40	0.34	0.37	0.35	0.35	0.35	0.33	0.33	0.33
Cu Grade		%	0.19	-	-	-	0.19	0.21	0.21	0.19	0.21	0.24	0.23	0.21	0.20	0.21	0.21	0.23	0.23	0.23	0.23	0.25	0.26	0.26	0.21	0.11	0.11	0.10	0.09	0.09	0.09
Co Grade		%	0.00777	-	-	-	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01
Sc Grade		g/t	28.35	-	-	-	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35	28.35
Contained Metals				-	-	-																									
Contained Au		oz	6,103,387	-	-	-	251,189	277,737	297,045	293,357	276,365	305,669	310,273	302,186	282,770	263,643	255,251	255,489	264,449	292,258	277,848	296,718	284,360	303,330	217,697	164,441	164,441	156,059	125,911	125,911	58,988
Contained Ag		oz	12,795,387	-	-	-	565,872	709,156	655,695	537,600	548,195	560,494	506,816	442,528	449,700	561,379	562,931	572,364	576,802	560,329	486,835	471,788	544,615	466,473	502,713	478,315	478,315	470,008	440,133	440,133	206,198
Contained Cu		t	1,988,096	-	-	-	68,925	87,153	87,468	80,241	86,172	101,548	97,786	87,301	84,304	91,121	89,969	95,348	96,219	95,963	98,184	105,443	110,339	110,275	89,797	45,154	45,154	43,341	36,822	36,822	17,251
Contained Co		t	79,734	-	-	-	2,691	3,226	3,371	3,399	3,505	3,730	3,626	3,520	3,546	3,655	3,589	3,469	3,355	3,258	3,063	3,088	3,358	2,723	2,877	3,059	3,059	3,056	3,043	3,043	1,426
Contained Sc (Metal)		kg	29,079,585	-	-	-	1,054,568	1,197,017	1,189,609	1,196,253	1,190,149	1,202,296	1,193,770	1,188,084	1,197,825	1,207,352	1,201,599	1,195,066	1,193,555	1,193,058	1,199,055	1,194,529	1,189,225	1,194,543	1,190,700	1,190,700	1,190,700	1,190,700	1,190,700	1,190,700	1,190,700	557,831
Copper Concentrate		%		-	-	-																									
Au Recovery		%	66	-	-	-	66	66	66	66	66	66	66	66	66	66	66	66	66	66	66	66	66	66	66	66	66	66	66	66	66
Ag Recovery		%	53	-	-	-	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53	53
Cu Recovery		%	80	-	-	-	80	80	80	80	80	80	80	80	80	80	80	80	80	80	80	80	80	80	80	80	80	80	80	80	80
Co Recovery		%	30	-	-	-	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30
Sc Recovery		%	0	-	-	-	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Secondary Product Stream		%		-	-	-																									
Au Recovery		%	23	-	-	-	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23
Ag Recovery		%	15	-	-	-	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15
Cu Recovery		%	5	-	-	-	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Co Recovery		%	48	-	-	-	48	48	48	48	48	48	48	48	48	48	48	48	48	48	48	48	48	48	48	48	48	48	48	48	48
Sc Recovery	72%	%	72	-	-	-	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72
Net Recovery		%		-	-	-																									
Au Recovery		%	89	-	-	-	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89	89
Ag Recovery		%	68	-	-	-	68	68	68	68	68	68	68	68	68	68	68	68	68	68	68	68	68	68	68	68	68	68	68	68	68
Cu Recovery		%	85	-	-	-	85	85	85	85	85	85	85	85	85	85	85	85	85	85	85	85	85	85	85	85	85	85	85	85	85
Co Recovery		%	78	-	-	-	78	78	78	78	78	78	78	78	78	78	78	78	78	78	78	78	78	78	78	78	78	78	78	78	78
Sc Recovery		%	72	-	-	-	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72	72
Copper Concentrate				-	-	-																									
Au		oz	4,028,236	-	-	-	165,785	183,306	196,050	193,616	182,401	201,742	204,780	199,443	186,628	174,005	168,466	168,623	174,536	192,890	183,380	195,834	187,677	200,198	143,680	108,531	108,531	102,999	83,102	83,102	38,932
Ag		oz	6,781,555	-	-	-	299,912	375,853	347,519	284,928	290,543	297,062	268,613	234,540	238,341	297,531	298,353	303,353	305,705	296,975	258,023	250,048	288,646	247,231	266,438	253,507	253,507	249,104	233,270	233,270	109,285
Cu		t	1,590,477	-	-	-	55,140	69,723	69,974	64,193	68,937	81,238	78,229	69,840	67,443	72,896	71,976	76,279	76,975	76,771	78,547	84,354	88,271	88,220	71,837	36,123	36,123	34,673	29,457	29,457	13,800
Co		t	23,920	-	-	-	807	968	1,011	1,020	1,051	1,119	1,088	1,056	1,064	1,096	1,077	1,041	1,006	977	919	927	1,007	817	863	918	918	917	913	913	428
Sc		kg	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Secondary Product Stream ("Pyrite Concentrate")																															
Au		oz	1,403,779	-	-	-	57,773	63,879	68,320	67,472	63,564	70,304	71,363	69,503	65,037	60,638	58,708	58,762	60,823	67,219	63,905	68,245	65,403	69,766	50,070	37,822	37,822	35,893	28,960	28,960	13,567
Ag		oz	1,919,308	-	-	-	84,881	106,373	98,354	80,640	82,229	84,074	76,022	66,379	67,455	84,207	84,440	85,855	86,520	84,049	73,025	70,768	81,692	69,971	75,407	71,747	71,747	70,501	66,020	66,020	30,930
Cu		T	99,405	-	-	-	3,446	4,358	4,373	4,012	4,309	5,077	4,889	4,365	4,215	4,556	4,498	4,767	4,811	4,798	4,909	5,272	5,517	5,514	4,490	2,258	2,258	2,167	1,841	1,841	863
Co		t	38,272	-	-	-	1,292	1,549	1,618	1,631	1,682	1,790	1,740	1,690	1,702	1,754	1,723	1,665	1,610	1,564	1,470	1,482	1,612	1,307	1,381	1,468	1,468	1,467	1,461	1,461	684
Sc to Sc2O3 Conversion Factor	1.534		-	-	-	-	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534	1.534
Overall mass % of Sc Processing	12.5%	%	-	-	-	-	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5
Sc_2O_3 Available for Recovery		kg	4,014,299	-	-	-	145,578	165,243	164,220	165,137	164,294	165,971	164,794	164,009	165,354	166,669	165,875	164,973	164,765	164,696	165,524	164,899	164,167	164,901	164,371	164,371	164,371	164,371	164,371	164,371	77,006
Copper Concentrate		t	-	-	-	-	220,559	278,891	279,897	256,770	275,750	324,952	312,914	279,362	269,773	291,586	287,902	305,114	307,899	307,083	314,187	337,418	353,084	352,879	287,350	144,494	144,494	138,692	117,829	117,829	55,202
Au grade in concentrate		g/t	-	-	-	-	23.4	20.4	21.8	23.5	20.6	19.3	20.4	22.2	21.5	18.6	18.2	17.2	17.6	19.5	18.2	18.1	16.5	17.6	15.6	23.4	23.4	23.1	21.9	21.9	21.9
Ag grade in concentrate		g/t	-	-	-	-	42.3	41.9	38.6	34.5	32.8	28.4	26.7	26.1	27.5	31.7	32.2	30.9	30.9	30.1	25.5	23.0	25.4	21.8	28.8	54.6	54.6	55.9	61.6	61.6	61.6
Cu grade in concentrate	25%	%	-	-	-	-	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25
Co grade in concentrate		%	-	-	-	-	0.37	0.35	0.36	0.40	0.38	0.34	0.35	0.38	0.39	0.38	0.37	0.34	0.33	0.32	0.29	0.27	0.29	0.23	0.30	0.64	0.64	0.66	0.77	0.77	0.77
Sc grade in concentrate			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Secondary Product Stream ("Co-Pyrite Concentrate") Produced		t	-	-	-	-	516,671	619,417	647,323	652,553	672,890	716,117	696,188	675,826	680,921	701,758	689,174	665,981	644,098	625,526	588,014	592,979	644,691	522,741	552,399	587,328	587,328	586,671	584,309	584,309	273,743
Au grade in concentrate		g/t	-	-	-	-	3.5	3.2	3.3	3.2	2.9	3.1	3.2	3.2	3.0	2.7	2.6	2.7	2.9	3.3	3.4	3.6	3.2	4.2	2.8	2.0	2.0	1.9	1.5	1.5	1.5
Ag grade in concentrate		g/t	-	-	-	-	5.1	5.3	4.7	3.8	3.8	3.7	3.4	3.1	3.1	3.7	3.8	4.0	4.2	4.2	3.9	3.7	3.9	4.2	4.2	3.8	3.8	3.7	3.5	3.5	3.5
Cu grade in concentrate		%	-	-	-	-	0.67	0.70	0.68	0.61	0.64	0.71	0.70	0.65	0.62	0.65	0.65	0.72	0.75	0.77	0.83	0.89	0.86	1.05	0.81	0.38	0.38	0.37	0.32	0.32	0.32
Co grade in concentrate	0.25%	%	-	-	-	-	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25
Total Recovered			-	-	-	-																									
Au		**oz**	**5,432,015**	-	-	-	**223,558**	**247,186**	**264,370**	**261,088**	**245,965**	**272,045**	**276,143**	**268,946**	**251,665**	**234,642**	**227,173**	**227,385**	**235,359**	**260,109**	**247,285**	**264,079**	**253,080**	**269,964**	**193,751**	**146,353**	**146,353**	**138,892**	**112,061**	**112,061**	**52,500**
Ag		**oz**	**8,700,863**	-	-	-	**384,793**	**482,226**	**445,873**	**365,568**	**372,772**	**381,136**	**344,635**	**300,919**	**305,796**	**381,737**	**382,793**	**389,207**	**392,225**	**381,024**	**331,048**	**320,816**	**370,339**	**317,202**	**341,845**	**325,254**	**325,254**	**319,605**	**299,290**	**299,290**	**140,215**
Cu		**t**	**1,689,882**	-	-	-	**58,586**	**74,080**	**74,348**	**68,205**	**73,246**	**86,315**	**83,118**	**74,205**	**71,658**	**77,452**	**76,474**	**81,046**	**81,786**	**81,569**	**83,456**	**89,627**	**93,788**	**93,733**	**76,327**	**38,381**	**38,381**	**36,840**	**31,298**	**31,298**	**14,663**
Co		**t**	**62,193**	-	-	-	**2,099**	**2,516**	**2,630**	**2,651**	**2,734**	**2,909**	**2,828**	**2,746**	**2,766**	**2,851**	**2,800**	**2,706**	**2,617**	**2,541**	**2,389**	**2,409**	**2,619**	**2,124**	**2,244**	**2,386**	**2,386**	**2,383**	**2,374**	**2,374**	**1,112**
Sc2O3		**kg**	**3,209,492**	-	-	-	**-**	**-**	**-**	**-**	**32,859**	**132,777**	**164,794**	**164,009**	**165,354**	**166,669**	**165,875**	**164,973**	**164,765**	**164,696**	**165,524**	**164,899**	**164,167**	**164,901**	**164,371**	**164,371**	**164,371**	**164,371**	**164,371**	**164,371**	**77,006**

Table 22-14: Annual Production and Projected Cash Flow, Scenario B—Part 2

	Inputs	Unit	Total/Avg	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25
REVENUE																															
Metal Prices							C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$	C$
Au	3,272.60	US$/oz Au					4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483	4,483
Ag	50.22	US$/oz Ag					68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8	68.8
Cu	4.88	US$/lb Cu					6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7
Co	19.57	US$/lb Co					26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8	26.8
Sc	1,500.00	US$/kg Sc					2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055	2,055
Exchange Rate	1.37	C$/US$	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37	1.37
Copper Concentrate Payable %																															
Au		%	-	-	-	-	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0
Ag		%	-	-	-	-	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0
Cu		%	-	-	-	-	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0
Co		%	-	-	-	-	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0
Sc		%	-	-	-	-	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Secondary Product Stream Payable %			-	-	-	-																									
Au		%	Gold/Silver Dore	-	-	-	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0
Ag		%	Gold/Silver Dore	-	-	-	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0
Cu		%	-	-	-	-	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Co		%	-	-	-	-	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0
Scandium Processing Payable %																															
Sc Ramp Up		%	-	-	-	-	0.0	0.0	0.0	0.0	20.0	80.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Sc	95%	%	-	-	-	-	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0
Copper Concentrate Payable			-	-	-	-																									
Au		oz	3,826,824	-	-	-	157,496	174,141	186,247	183,935	173,281	191,655	194,541	189,471	177,297	165,304	160,042	160,191	165,809	183,246	174,211	186,042	178,294	190,188	136,496	103,105	103,105	97,849	78,946	78,946	36,986
Ag		oz	6,103,400	-	-	-	269,921	338,267	312,767	256,435	261,489	267,356	241,751	211,086	214,507	267,778	268,518	273,018	275,134	267,277	232,220	225,043	259,782	222,508	239,794	228,156	228,156	224,194	209,943	209,943	98,356
Cu		t	1,642,167	-	-	-	53,486	67,631	67,875	62,267	67,731	82,863	81,858	73,197	70,722	76,353	75,402	79,845	80,563	80,353	82,187	88,183	92,227	92,174	75,259	38,384	38,384	36,887	31,502	31,502	15,336
Co		t	19,136	-	-	-	646	774	809	816	841	895	870	845	851	877	861	832	805	782	735	741	806	653	690	734	734	733	730	730	342
Sc		kg	-										-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Secondary Product Stream Payable																															
Au		oz	1,361,819	-	-	-	-	-	-	-	12,333	54,562	85,689	83,884	79,552	75,284	73,412	73,465	75,464	81,669	78,454	82,664	79,907	84,140	65,033	53,150	53,150	51,280	44,553	44,553	29,621
Ag		oz	1,861,938	-	-	-	-	-	-	-	15,954	65,249	96,872	87,517	88,560	104,811	105,037	106,410	107,056	104,659	93,964	91,775	102,372	91,001	96,275	92,724	92,724	91,515	87,168	87,168	53,127
Cu		t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Co		t	37,124	-	-	-	-	-	-	-	326	1,389	2,086	2,037	2,049	2,100	2,069	2,013	1,960	1,915	1,824	1,836	1,961	1,666	1,737	1,822	1,822	1,821	1,815	1,815	1,062
Payable Scandium																															
Sc2O3		kg	3,049,017	-	-	-	-	-	-	-	31,216	126,138	156,555	155,809	157,086	158,336	157,581	156,725	156,526	156,461	157,248	156,654	155,959	156,656	156,152	156,152	156,152	156,152	156,152	156,152	73,156
Gross Revenue																															
Au Gross Revenue		C$ '000	23,263,082	-	-	-	706,125	780,755	835,032	824,666	832,193	1,103,902	1,256,399	1,225,578	1,151,573	1,078,671	1,046,684	1,047,590	1,081,741	1,187,735	1,132,812	1,204,737	1,157,632	1,229,938	903,547	700,562	700,562	668,611	553,705	553,705	298,627
Ag Gross Revenue		C$ '000	548,026	-	-	-	18,571	23,273	21,519	17,643	19,088	22,884	23,298	20,544	20,851	25,635	25,701	26,105	26,295	25,590	22,442	21,797	24,917	21,570	23,122	22,077	22,077	21,721	20,442	20,442	10,422
Cu Gross Revenue		C$ '000	24,204,027	-	-	-	788,328	996,818	1,000,415	917,754	998,292	1,221,321	1,206,509	1,078,858	1,042,377	1,125,365	1,111,351	1,176,835	1,187,430	1,184,324	1,211,353	1,299,734	1,359,338	1,358,555	1,109,249	565,749	565,749	543,678	464,303	464,303	226,036
Co Gross Revenue		C$ '000	3,325,402	-	-	-	38,174	45,765	47,827	48,213	69,006	135,026	174,744	170,321	171,428	175,954	173,220	168,182	163,429	159,395	151,246	152,325	163,558	137,068	143,510	151,097	151,097	150,955	150,442	150,442	82,981
Sc Gross Revenue		C$ '000	6,265,731	-	-	-	-	-	-	-	64,149	259,214	321,720	320,187	322,813	325,380	323,830	322,069	321,662	321,528	323,144	321,924	320,495	321,928	320,892	320,892	320,892	320,892	320,892	320,892	150,335
Total Gross Revenue		**C$ '000**	**57,606,268**	-	-	-	**1,551,198**	**1,846,611**	**1,904,793**	**1,808,277**	**1,982,727**	**2,742,346**	**2,982,670**	**2,815,488**	**2,709,041**	**2,731,004**	**2,680,786**	**2,740,781**	**2,780,557**	**2,878,571**	**2,840,998**	**3,000,517**	**3,025,939**	**3,069,059**	**2,500,321**	**1,760,378**	**1,760,378**	**1,705,857**	**1,509,784**	**1,509,784**	**768,402**
Transport				-	-	-																									
Moisture Content	10%	%		-	-	-	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10
Cu Concentrate	$95.90 C$/dmt	C$ '000	671,118	-	-	-	23,267	29,420	29,526	27,087	29,089	34,279	33,009	29,470	28,458	30,759	30,371	32,187	32,480	32,394	33,144	35,594	37,247	37,225	30,313	15,243	15,243	14,631	12,430	12,430	5,823
Treatment				-	-	-																									
Cu Concentrate	$77.00 US/dmt	C$ '000	671,118	-	-	-	23,267	29,420	29,526	27,087	29,089	34,279	33,009	29,470	28,458	30,759	30,371	32,187	32,480	32,394	33,144	35,594	37,247	37,225	30,313	15,243	15,243	14,631	12,430	12,430	5,823
Refining cost				-	-	-																									
Au	$5.00US/oz Au	C$ '000	35,542	-	-	-	1,079	1,193	1,276	1,260	1,271	1,687	1,920	1,872	1,759	1,648	1,599	1,601	1,653	1,815	1,731	1,841	1,769	1,879	1,380	1,070	1,070	1,022	846	846	456
Ag	$0.35 US/oz Ag	C$ '000	3,819	-	-	-	129	162	150	123	133	159	162	143	145	179	179	182	183	178	156	152	174	150	161	154	154	151	142	142	73
Cu	$0.08 US/lb Cu	C$ '000	396,787	-	-	-	12,923	16,341	16,400	15,045	16,365	20,022	19,779	17,686	17,088	18,449	18,219	19,292	19,466	19,415	19,858	21,307	22,284	22,271	18,184	9,275	9,275	8,913	7,612	7,612	3,706
Co	$0.12 US/lb Co	C$ '000	20,391	-	-	-	234	281	293	296	423	828	1,072	1,044	1,051	1,079	1,062	1,031	1,002	977	927	934	1,003	840	880	927	927	926	922	922	509
Total Charges		**C$ '000**	**1,798,775**	-	-	-	**60,899**	**76,817**	**77,172**	**70,897**	**76,371**	**91,254**	**88,951**	**79,686**	**76,961**	**82,873**	**81,801**	**86,479**	**87,265**	**87,174**	**88,960**	**95,422**	**99,723**	**99,592**	**81,231**	**41,911**	**41,911**	**40,273**	**34,382**	**34,382**	**16,390**
Gross Less Charge		C$ '000	55,807,493	-	-	-	1,490,299	1,769,793	1,827,622	1,737,380	1,906,357	2,651,092	2,893,719	2,735,802	2,632,080	2,648,131	2,598,985	2,654,302	2,693,292	2,791,397	2,752,037	2,905,095	2,926,216	2,969,467	2,419,090	1,718,468	1,718,468	1,665,585	1,475,402	1,475,402	752,012
Royalty 2% NSR	2%	C$ '000	1,116,150	-	-	-	29,806	35,396	36,552	34,748	38,127	53,022	57,874	54,716	52,642	52,963	51,980	53,086	53,866	55,828	55,041	58,102	58,524	59,389	48,382	34,369	34,369	33,312	29,508	29,508	15,040
Net Smelter Return		**C$ '000**	**54,691,343**	-	-	-	**1,460,493**	**1,734,398**	**1,791,069**	**1,702,632**	**1,868,229**	**2,598,070**	**2,835,845**	**2,681,086**	**2,579,438**	**2,595,169**	**2,547,006**	**2,601,216**	**2,639,426**	**2,735,569**	**2,696,997**	**2,846,993**	**2,867,692**	**2,910,078**	**2,370,708**	**1,684,098**	**1,684,098**	**1,632,273**	**1,445,894**	**1,445,894**	**736,972**
Unit NSR		**C$/t milled**	**53.32**	-	-	-	**39.26**	**41.08**	**42.68**	**40.35**	**44.50**	**61.26**	**67.35**	**63.98**	**61.05**	**60.94**	**60.09**	**61.71**	**62.69**	**65.00**	**63.77**	**67.57**	**68.36**	**69.06**	**56.45**	**40.10**	**40.10**	**38.86**	**34.43**	**34.43**	**37.45**
Revenue by Metal																															
Net Revenue by Metal																															
Au		%	42	-	-	-	47	44	46	47	44	42	43	45	44	41	40	39	40	42	41	41	40	41	37	41	41	40	37	37	40
Ag		%	1	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Cu		%	40	-	-	-	49	52	51	49	48	43	39	37	37	39	40	41	41	39	41	42	43	42	43	31	31	30	29	29	28
Co		%	6	-	-	-	3	3	3	3	4	5	6	6	6	7	7	6	6	6	5	5	6	5	6	9	9	9	10	10	11
Sc		%	11	-	-	-	0	0	0	0	3	10	11	12	12	12	12	12	12	12	12	11	11	11	13	19	19	19	22	22	20
Revenue per Metal Unit (NSR Factor)																															
Au		$ per g Au	-	-	-	-	99.37	99.37	99.37	99.37	106.44	127.66	143.14	143.36	143.95	144.62	144.95	144.94	144.59	143.65	144.12	143.52	143.90	143.33	146.71	150.59	150.59	151.44	155.44	155.44	178.95
Ag		$ per g Ag	-	-	-	-	1.51	1.51	1.51	1.51	1.60	1.88	2.12	2.14	2.13	2.10	2.10	2.10	2.10	2.10	2.12	2.13	2.11	2.13	2.12	2.12	2.12	2.13	2.14	2.14	2.33
Cu		$ per lb Cu	-	-	-	-	5.53	5.53	5.53	5.53	5.61	5.83	5.99	6.00	6.01	6.00	6.00	6.00	6.00	6.00	5.99	5.99	5.98	5.98	6.00	6.09	6.09	6.10	6.13	6.13	6.39
Co		$ per lb Co	-	-	-	-	8.03	8.03	8.03	8.03	11.15	20.51	27.30	27.41	27.38	27.27	27.33	27.46	27.59	27.71	27.97	27.94	27.59	28.52	28.25	27.98	27.98	27.98	28.00	28.00	32.97
Sc		$ per kg Sc	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

Table 22-15: Annual Production and Projected Cash Flow, Scenario B—Part 3

	Inputs	Unit	Total/Avg	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25
OPERATING COST																															
OPEX sensitivity Factor (Applied to all costs)																															
Mining (Open Pit)		C$/t moved	2.32	-	-	-	1.89	1.93	1.97	2.01	2.05	2.09	2.12	2.16	2.20	2.24	2.28	2.32	2.35	2.39	2.43	2.47	2.51	2.54	2.58	2.58	2.58	2.58	2.58	2.58	2.58
Mining (Rehandle)	$.81/t moved	C$/t moved		-	-	-	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81
Processing	$7.93/t milled	C$/t milled	10.84	-	-	-	7.93	7.93	7.93	7.93	11.02	11.02	11.44	11.44	11.44	11.44	11.44	11.44	11.44	11.44	11.44	11.44	11.44	11.44	11.44	11.44	11.44	11.44	11.44	11.44	11.44
Processing Co-Au-Acid Plant	$3.083/t milled	C$/t milled		-	-	-																									
Scandium Processing	$685.80 US/kg Sc	US$/kg Sc2O3 C$/kg Sc2O3	939.55	-	-	-	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55	939.55
G&A and tailings	$2.56/t milled	C$/t milled	2.56	-	-	-	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56	2.56
TOTAL OPERATING COST				-	-	-																									
Mining (Open Pit)		C$ '000	5,980,015	-	-	-	292,476	315,268	298,501	311,698	281,905	269,636	288,295	311,397	316,227	302,499	316,556	330,604	352,908	350,639	405,733	378,141	324,264	226,348	120,884	34,020	34,020	34,020	34,020	34,020	15,938
Processing		C$ '000	11,123,590	-	-	-	295,130	334,996	332,923	334,782	462,479	467,199	481,634	479,340	483,270	487,114	484,792	482,156	481,547	481,346	483,766	481,940	479,800	481,946	480,395	480,395	480,395	480,395	480,395	480,395	225,060
Processing (Scandium)		C$ '000	2,864,692	-	-	-	-	-	-	-	29,329	118,513	147,090	146,390	147,590	148,764	148,055	147,250	147,064	147,003	147,741	147,184	146,530	147,185	146,712	146,712	146,712	146,712	146,712	146,712	68,733
G&A and tailings		C$ '000	2,621,778	-	-	-	95,079	107,922	107,254	107,853	107,302	108,397	107,629	107,116	107,994	108,853	108,335	107,746	107,609	107,565	108,105	107,697	107,219	107,698	107,352	107,352	107,352	107,352	107,352	107,352	50,293
Total Operating Cost		**C$ '000**	**22,590,075**	-	-	-	**682,684**	**758,185**	**738,677**	**754,332**	**881,015**	**963,745**	**1,024,648**	**1,044,242**	**1,055,081**	**1,047,230**	**1,057,738**	**1,067,756**	**1,089,128**	**1,086,553**	**1,145,346**	**1,114,962**	**1,057,813**	**963,177**	**855,343**	**768,479**	**768,479**	**768,479**	**768,479**	**768,479**	**360,025**
Unit Operating Cost		**C$/t milled**	**21.92**	-	-	-	**18.4**	**18.0**	**17.6**	**17.9**	**21.0**	**22.7**	**24.3**	**24.9**	**25.0**	**24.6**	**25.0**	**25.3**	**25.9**	**25.8**	**27.1**	**26.5**	**25.2**	**22.9**	**20.4**	**18.3**	**18.3**	**18.3**	**18.3**	**18.3**	**18.3**
C1 Cash Cost		C$/lb CuEq produced	2.59	-	-	-	2.26	2.13	2.00	2.14	2.40	2.10	2.21	2.42	2.56	2.52	2.60	2.56	2.58	2.47	2.66	2.42	2.26	1.99	2.20	3.00	3.00	3.13	3.70	3.70	3.70
Operating Cashflow EBITDA		**C$ '000**	**32,101,268**	-	-	-	**777,808**	**976,213**	**1,052,392**	**948,300**	**987,215**	**1,634,325**	**1,811,197**	**1,636,844**	**1,524,358**	**1,547,939**	**1,489,268**	**1,533,460**	**1,550,298**	**1,649,016**	**1,551,651**	**1,732,031**	**1,809,878**	**1,946,900**	**1,515,365**	**915,619**	**915,619**	**863,794**	**677,415**	**677,415**	**376,947**
CAPITAL COST																															
Direct Cost																															
Mining		C$ '000	683,132	-	232,481	161,224	-	39,159	19,580	9,886	20,088	29,369	121	3,616	2,752	24,699	42,368	43,369	29,369	10,798	968	13,286	-	-	-	-	-	-	-	-	-
Mining Pre-stripping		C$ '000	97,200	-		97,200		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Processing and Processing Infrastructure		C$ '000	2,719,762	-	724,143	1,086,214	-	-	363,762	545,644	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
General Infrastructure		C$ '000	183,619	-	73,448	110,171	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Tailings and Site Water Management		C$ '000	1,221,858	-		156,721	36,403	89,501	41,340	41,485	41,352	41,616	41,431	41,307	41,519	41,726	41,601	41,459	41,426	41,415	41,546	41,447	41,332	41,448	41,364	41,364	41,364	41,364	41,364	41,364	27,598
Power Line		C$ '000	205,520	-	140,000	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520
Total Direct Cost		**C$ '000**	**5,111,091**	-	**1,170,071**	**1,614,051**	**38,923**	**131,181**	**427,202**	**599,535**	**63,960**	**73,506**	**44,072**	**47,443**	**46,791**	**68,945**	**86,489**	**87,348**	**73,316**	**54,733**	**45,034**	**57,253**	**43,852**	**43,968**	**43,884**	**43,884**	**43,884**	**43,884**	**43,884**	**43,884**	**30,118**
Other Costs				-																											
EPCM / Owners / Indirect Cost	10%	C$ '000	511,109	-	117,007	161,405	3,892	13,118	42,720	59,953	6,396	7,351	4,407	4,744	4,679	6,894	8,649	8,735	7,332	5,473	4,503	5,725	4,385	4,397	4,388	4,388	4,388	4,388	4,388	4,388	3,012
Subtotal Costs		**C$ '000**	**5,622,201**	-	**1,287,078**	**1,775,456**	**42,815**	**144,299**	**469,923**	**659,488**	**70,356**	**80,856**	**48,479**	**52,188**	**51,470**	**75,839**	**95,138**	**96,083**	**80,647**	**60,206**	**49,537**	**62,978**	**48,237**	**48,364**	**48,272**	**48,272**	**48,272**	**48,272**	**48,272**	**48,272**	**33,130**
Contingency	25%	C$ '000	1,405,550	-	321,769	443,864	10,704	36,075	117,481	164,872	17,589	20,214	12,120	13,047	12,867	18,960	23,784	24,021	20,162	15,052	12,384	15,745	12,059	12,091	12,068	12,068	12,068	12,068	12,068	12,068	8,282
Subtotal Cost+ Contingency		**C$ '000**	**7,027,751**	-	**1,608,847**	**2,219,320**	**53,519**	**180,373**	**587,403**	**824,360**	**87,945**	**101,070**	**60,599**	**65,235**	**64,337**	**94,799**	**118,922**	**120,104**	**100,809**	**75,258**	**61,921**	**78,723**	**60,296**	**60,455**	**60,341**	**60,341**	**60,341**	**60,341**	**60,341**	**60,341**	**41,412**
Sustaining Cost				-																											
Sustaining Mining (LCC)		C$ '000	521,152	-	-	-	5,809	10,008	19,475	19,733	32,066	60,033	19,132	19,885	40,690	26,429	9,973	33,820	42,697	26,943	23,234	30,489	34,530	37,961	18,974	1,854	1,854	1,854	1,854	1,854	
Sustaining Processing and Infrastructure		C$ '000	285,000	-	-	-	-	-	-	-	-	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	
Closure Cost																															
Reclamation and closure		C$ '000	503,300	-	-	-	-	-	-	-	-	-	-	-	75,950	-	-	-	-	75,950	-	-	-	-	-	75,950	-	-	-	-	275,450
Total Capital Cost		**C$ '000**	**8,337,203**	-	**1,608,847**	**2,219,320**	**59,328**	**190,382**	**606,878**	**844,093**	**120,011**	**176,103**	**94,731**	**100,119**	**195,977**	**136,229**	**143,894**	**168,924**	**158,506**	**193,151**	**100,155**	**124,211**	**109,827**	**113,417**	**94,315**	**153,145**	**77,195**	**77,195**	**77,195**	**77,195**	**316,862**
Initial Capital Cost		**C$ '000**	**3,828,168**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital Cost and Closure		**C$ '000**	**4,509,035**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
AISC Cash Cost		**C$/lb CuEq produced**	**3.08**	-	-	-	**2.46**	**2.66**	**3.65**	**4.53**	**2.74**	**2.54**	**2.45**	**2.69**	**2.90**	**2.90**	**3.01**	**3.03**	**3.01**	**2.78**	**2.93**	**2.73**	**2.53**	**2.27**	**2.50**	**3.37**	**3.37**	**3.52**	**4.16**	**4.16**	**4.22**
PRE-TAX CASH FLOW																															
Net Pre-Tax Cashflow		C$ '000	23,764,065	-	(1,608,847)	(2,219,320)	718,480	785,831	445,514	104,207	867,204	1,458,222	1,716,466	1,536,725	1,328,381	1,411,710	1,345,373	1,364,537	1,391,792	1,455,865	1,451,496	1,607,820	1,700,051	1,833,484	1,421,050	762,475	838,425	786,599	600,220	600,220	60,085
Cumulative Pre-Tax Cashflow		C$ '000		-	(1,608,847)	(3,828,168)	(3,109,688)	(2,323,857)	(1,878,343)	(1,774,136)	(906,932)	551,290	2,267,756	3,804,481	5,132,862	6,544,573	7,889,946	9,254,483	10,646,275	12,102,140	13,553,636	15,161,456	16,861,508	18,694,991	20,116,042	20,878,516	21,716,941	22,503,540	23,103,760	23,703,981	23,764,065
Discounted Cashflow NPV	5%	C$ '000	**11,567,025**	-			684,266	712,772	384,852	85,732	679,477	1,088,147	1,219,860	1,040,116	856,286	866,668	786,612	759,825	738,097	735,311	698,194	736,561	741,727	761,850	562,358	287,369	300,946	268,899	195,414	186,109	17,743
Net Pre-Tax Cashflow		US$ '000	17,346,033	-	(1,174,341)	(1,619,942)	524,438	573,599	325,193	76,064	632,996	1,064,395	1,252,895	1,121,697	969,621	1,030,446	982,024	996,012	1,015,907	1,062,675	1,059,486	1,173,591	1,240,913	1,338,309	1,037,263	556,551	611,989	574,160	438,117	438,117	43,857
Cumulative Pre-Tax Cashflow		US$ '000		-	(1,174,341)	(2,794,283)	(2,269,845)	(1,696,246)	(1,371,053)	(1,294,990)	(661,994)	402,402	1,655,296	2,776,994	3,746,615	4,777,060	5,759,085	6,755,097	7,771,004	8,833,679	9,893,165	11,066,756	12,307,670	13,645,979	14,683,242	15,239,793	15,851,782	16,425,942	16,864,059	17,302,176	17,346,033
Discounted Cashflow NPV	5%	US$ '000	**8,443,084**	-			499,465	520,271	280,914	62,578	495,969	794,268	890,409	759,209	625,026	632,604	574,169	554,617	538,757	536,723	509,631	537,636	541,406	556,095	410,480	209,758	219,669	196,277	142,638	135,846	12,951
Payback year Pre-tax		**Year 6**																													
TAXES																															
Federal Tax After CTM Credit		C$ '000	(2,308,351)	-	-	-	(4,178)	(681)	(10,728)	(7,596)	(8,737)	(36,647)	(46,449)	(43,860)	(42,938)	(45,160)	(44,287)	(122,067)	(176,481)	(191,907)	(180,015)	(206,032)	(217,057)	(236,153)	(180,697)	(105,275)	(105,267)	(99,628)	(76,208)	(76,829)	(43,475)
Provincial Tax		C$ '000	(2,517,376)	-	-	-	(12,378)	(2,018)	(31,786)	(22,507)	(25,889)	(108,582)	(137,626)	(129,956)	(127,222)	(133,808)	(131,221)	(138,363)	(141,185)	(153,526)	(144,012)	(164,825)	(173,645)	(188,922)	(144,558)	(84,220)	(84,213)	(79,703)	(60,967)	(61,463)	(34,780)
BC Mineral Taxes		C$ '000	(3,501,149)	-	-	-	(16,152)	(85,069)	(36,644)	(19,661)	(97,815)	(189,338)	(227,110)	(203,097)	(174,869)	(185,033)	(175,945)	(177,482)	(181,584)	(191,980)	(192,058)	(211,723)	(224,399)	(241,700)	(187,491)	(100,805)	(110,679)	(103,804)	(79,080)	(79,080)	(8,552)
Total Taxes		**C$ '000**	**(8,326,876)**	-	-	-	**(32,708)**	**(87,768)**	**(79,158)**	**(49,764)**	**(132,441)**	**(334,567)**	**(411,185)**	**(376,914)**	**(345,029)**	**(364,001)**	**(351,453)**	**(437,912)**	**(499,250)**	**(537,414)**	**(516,085)**	**(582,580)**	**(615,100)**	**(666,775)**	**(512,746)**	**(290,300)**	**(300,159)**	**(283,135)**	**(216,255)**	**(217,373)**	**(86,806)**
AFTER-TAX CASH FLOW																															
After-Tax Cashflow		C$ '000	15,437,190	-	(1,608,847)	(2,219,320)	685,772	698,063	366,356	54,443	734,763	1,123,655	1,305,281	1,159,811	983,352	1,047,709	993,921	926,625	892,542	918,452	935,411	1,025,240	1,084,951	1,166,709	908,304	472,175	538,266	503,464	383,965	382,848	(26,722)
Cumulative After-Tax Cashflow		C$ '000		-	(1,608,847)	(3,828,168)	(3,142,396)	(2,444,333)	(2,077,977)	(2,023,534)	(1,288,771)	(165,116)	1,140,165	2,299,977	3,283,329	4,331,038	5,324,959	6,251,584	7,144,126	8,062,578	8,997,989	10,023,229	11,108,180	12,274,889	13,183,194	13,655,368	14,193,634	14,697,099	15,081,064	15,463,911	15,437,190
Discounted Cashflow NPV	5%	C$ '000	**7,273,935**	-	-	-	653,116	633,164	316,472	44,791	575,706	838,489	927,639	785,006	633,877	643,203	581,125	515,979	473,334	463,880	449,949	469,674	473,361	484,792	359,447	177,958	193,206	172,109	125,008	118,709	(7,891)
After-Tax Cashflow		US$ '000	11,268,022	-	(1,174,341)	(1,619,942)	500,563	509,535	267,413	39,740	536,323	820,186	952,760	846,578	717,775	764,751	725,490	676,369	651,491	670,403	682,782	748,351	791,935	851,612	662,996	344,653	392,895	367,492	280,267	279,451	(19,505)
Cumulative After-Tax Cashflow		US$ '000		-	(1,174,341)	(2,794,283)	(2,293,720)	(1,784,185)	(1,516,772)	(1,477,032)	(940,709)	(120,523)	832,238	1,678,815	2,396,590	3,161,342	3,886,832	4,563,200	5,214,691	5,885,094	6,567,875	7,316,226	8,108,161	8,959,773	9,622,769	9,967,422	10,360,317	10,727,809	11,008,076	11,287,527	11,268,022
Discounted Cashflow NPV	5%	US$ '000	**5,309,442**	-			476,727	462,163	231,001	32,694	420,223	612,036	677,109	572,997	462,684	469,491	424,179	376,627	345,499	338,599	328,430	342,828	345,519	353,863	262,370	129,896	141,027	125,627	91,247	86,649	(5,760)
Payback year		**Year 7**		-																											
PROJECT ECONOMICS																															
Pre-Tax IRR		%	23			%	23																								
Pre-tax NPV at 5% discounting	5%	C$ '000	11,567,025			US$ '000	**8,443,084**																								
Pre-tax NPV at 8% discounting	7.5%	C$ '000	8,136,870			US$ '000	5,939,321																								
Pre-tax NPV at 10% discounting	10%	C$ '000	5,692,771			US$ '000	4,155,307																								
After-Tax IRR		%	19			%	19																								
After-Tax NPV at 5% discounting	5%	C$ '000	7,273,935			US$ '000	**5,309,442**																								
After-Tax NPV at 8% discounting	8%	C$ '000	4,938,098			US$ '000	3,604,451																								
After-tax NPV at 10% discounting	10%	C$ '000	3,255,086			US$ '000	2,375,975																								

22.5 Sensitivity Analysis

Sensitivity cases were evaluated for the key value drivers using after-tax NPV (5%) and after-tax IRR, including ±20% copper and gold prices, +20% initial capital, ±20% exchange rate, ±20% head grade, +20% operating costs and, for B, a ±40% scandium price sensitivity (Table 22-16).

Overall, the sensitivity analysis demonstrates that the Project's after-tax economics remain positive across the tested ranges (Figure 22-5 to Figure 22-7), with the greatest variability in after-tax NPV(5%) and IRR driven by simultaneous changes in the overall metal price deck. Changes to copper and gold prices individually have a meaningful but smaller effect, while +20% initial CAPEX and +20% OPEX reduce value but do not eliminate Project attractiveness in any of the evaluated scenarios. Scenario B shows additional exposure to scandium oxide price, with after-tax NPV(5%) varying within a narrower range relative to the broader multi-metal price cases, indicating that scandium provides incremental upside while the base-case copper–gold Project remains financially robust on its own.

Table 22-16: Sensitivity Analysis Results

Variable	Case	Metal Price / Comment	Scenario A1		Scenario A2		Scenario B	
			NPV (5%) $M	IRR (%)	NPV (5%) $M	IRR (%)	NPV (5%) $M	IRR (%)
Base Case		Consensus forecast	4,963	19	6,727	23	7,274	19
Copper price	-20%	US$3.90/lb Cu	3,218	15	4,807	19	5,433	16
Copper price	20%	US$5.86/lb Cu	6,688	23	8,632	28	9,099	22
Gold price	-20%	US$2,618.08/oz Au	3,625	16	5,223	19	5,539	16
Gold price	20%	US$3,927.12/oz Au	6,289	22	8,222	27	8,996	22
Metal price	-20%	All metal prices	1,708	10	3,165	14	2,993	11
Metal price	20%	All metal prices	8,118	27	10,233	32	11,444	26
Initial CAPEX	20%	Variable per Scenario	4,448	16	6,222	19	6,732	16
OPEX	20%	Variable per Scenario	3,660	16	5,438	20	5,591	16
Scandium Oxide price	-40%	US$900/kg Sc_2O_3					6,496	18
Scandium Oxide price	40%	US$2,100/kg Sc_2O_3					8,050	20
Exchange Rate	-20%	CAD:USD 1.10	2,568	14	4,000	18	4,418	15
Exchange Rate	20%	CAD:USD 1.67	7,562	23	9,699	27	10,380	22
Copper Recovery	Base +5%	85%/94%/90%	5,463	20	7,224	24	7,775	20
Copper Recovery	Base -5%	75%/84%/80%	4,461	18	6,231	22	6,772	18
Gold Recovery	Base +5%	71%/80%/94%	5,465	20	7,226	24	7,779	20
Gold Recovery	Base -5%	61%/70%/84%	4,459	18	6,228	22	6,769	18
Discount Rate	8%	NPV 8%	3,439	19	4,872	23	4,938	19
Discount Rate	10%	NPV 10%	2,310	19	3,500	23	3,255	19
Head Grade	-20%	All Head Grades -20%	1,858	11	3,316	15	3,342	12
Head Grade	20%	All Head Grades +20%	7,986	26	10,087	31	11,109	26

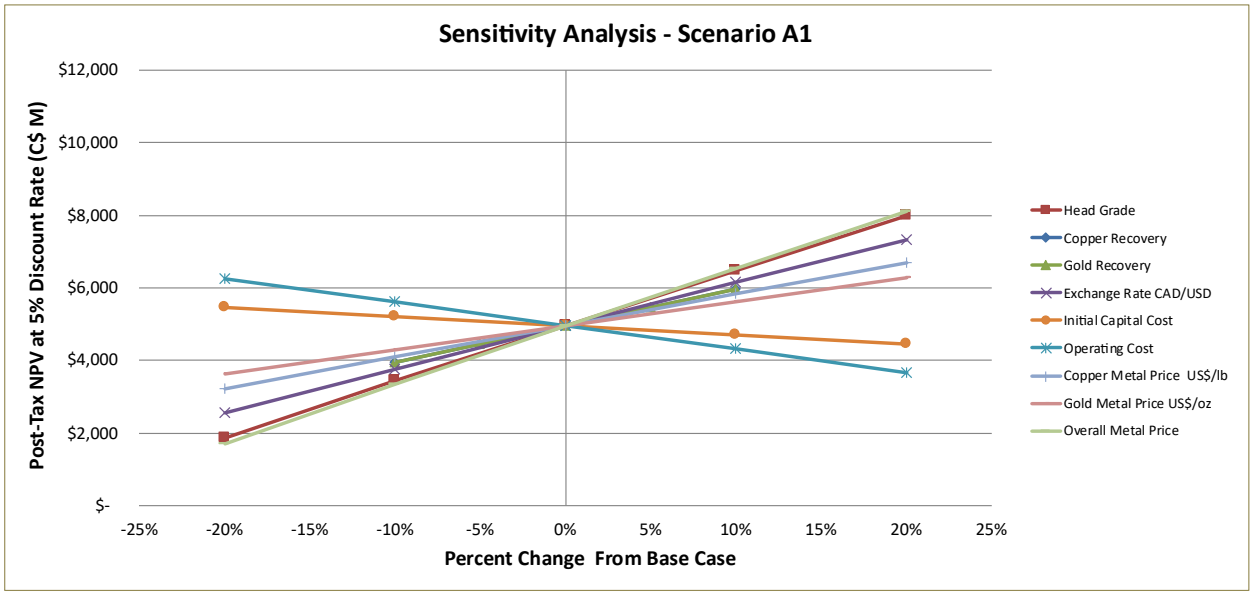


Figure 22-5: Sensitivity Analysis—Scenario A1

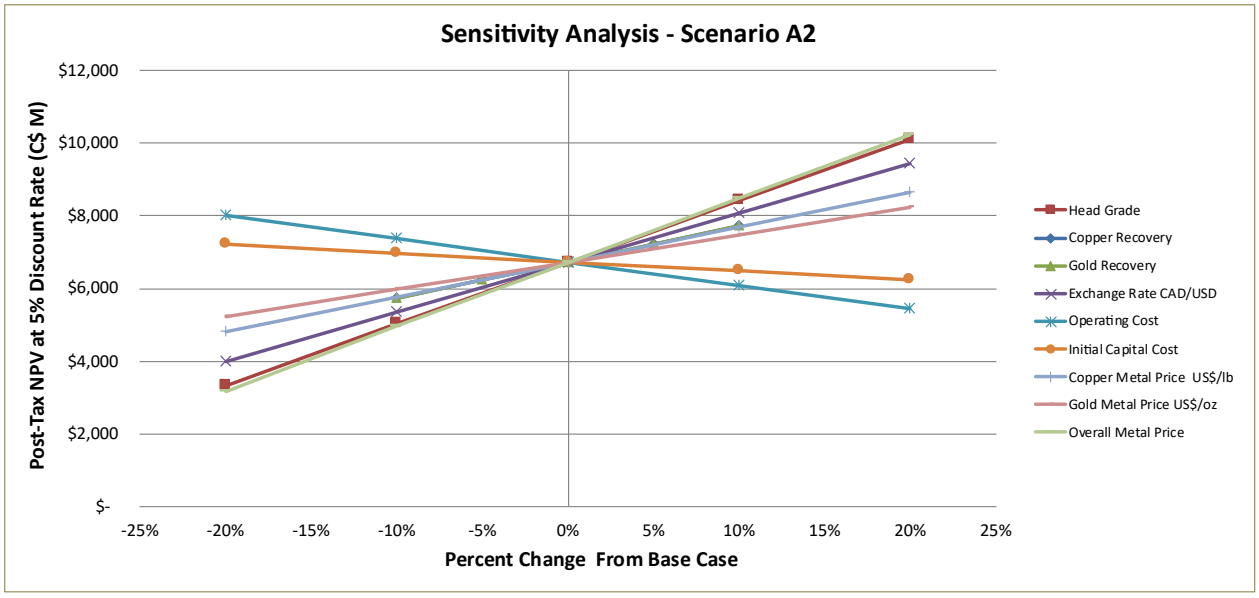


Figure 22-6: Sensitivity Analysis—Scenario A2

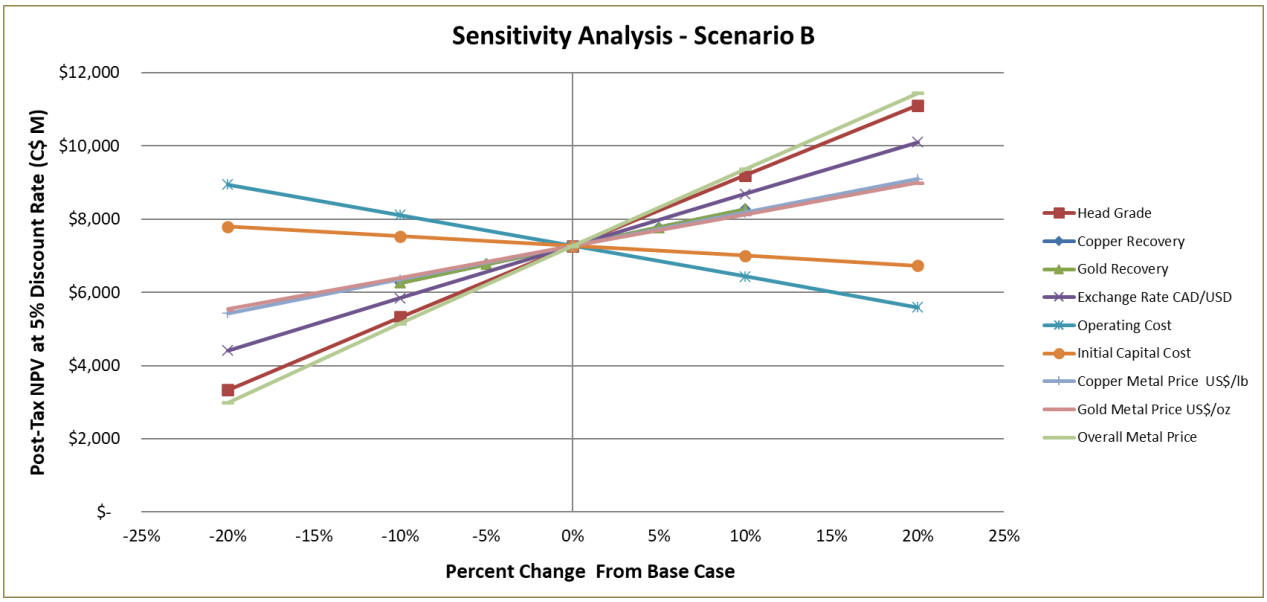


Figure 22-7: Sensitivity Analysis—Scenario B

22.6 Economic Analysis Conclusion

The economic analysis demonstrates that, under the assumptions applied, the Project shows potential for positive economic performance at the PEA level. The results are subject to the inherent uncertainties associated with early-stage studies and the inclusion of Inferred Mineral Resources.

Further work, including additional technical studies, cost refinement, and risk reduction, is required to advance the Project to higher levels of confidence and support future economic evaluations.

The PEA demonstrates an operation with an after-tax NPV(5%) of $4.96 billion (A1), $6.73 billion (A2), or $7.27 billion (B), and an IRR of 19% (A1), 23% (A2), or 19% (B) at analyst consensus metal prices. Using a spot-price scenario, the Project delivers a compelling after-tax NPV(5%) of $11.05 billion (A1), 13.53 billion (A2), or $14.85 billion (B) and an IRR of 34% (A1), 39% (A2), or 32% (B) (Table 22-17)

Project economics show the greatest leverage to overall metal prices, with NPV (5%) ranging from $3.2 billion to $10.2 billion (IRR: 14%–32%) at ±20% on all metals; even under additional +20% CAPEX and +20% OPEX sensitivities, applied on top of a 25% contingency already embedded in the base case, all scenarios deliver IRRs of 16% or better, and Scenario B provides additional scandium oxide upside with NPV(5%) of $6.5 billion–$8.1 billion (IRR: 18%–20%) at ±40% metal price.

Table 22-17: Summary Financial Results (Pre-Tax and Post-Tax)

	Scenario A1		Scenario A2		Scenario B	
	NPV (5%) ($M)	IRR (%)	NPV (5%) ($M)	IRR (%)	NPV (5%) ($M)	IRR (%)
Pre-Tax						
Consensus Metal Prices	7,883	24	10,576	29	11,567	23
Spot Metal Prices	17,230	43	21,073	50	23,258	40
Post-Tax						
Consensus Metal Prices	4,963	19	6,727	23	7,274	19
Spot Metal Prices	11,047	34	13,326	39	14,848	32

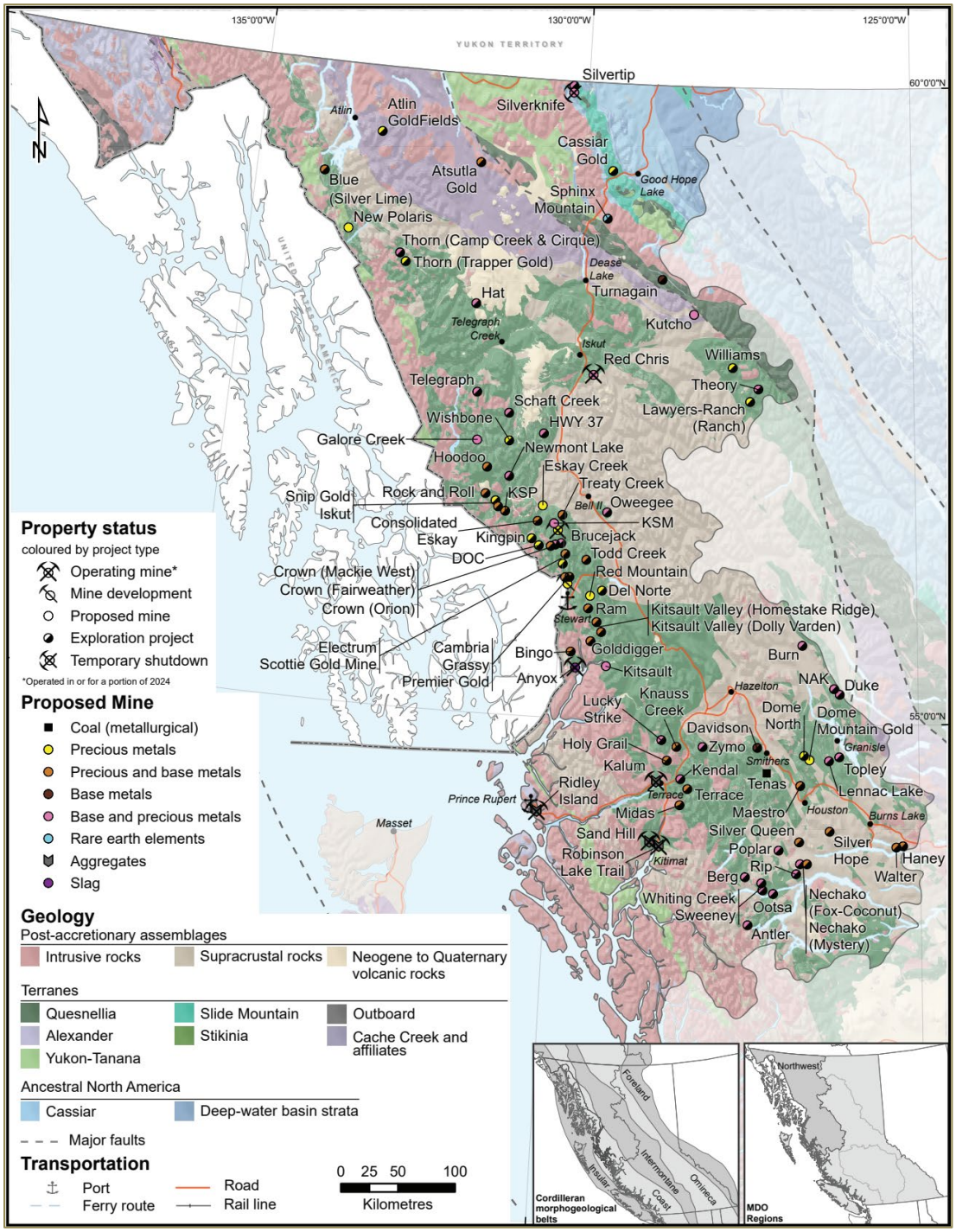
23 ADJACENT PROPERTIES

Several properties that lie within, adjacent to, or in reasonable proximity to the Hat Project are described in this section. The Qualified Person has not independently verified the technical information disclosed for these adjacent properties, and such information is not necessarily indicative of the mineralization, exploration potential, or economic potential of the Hat Project.

23.1 Golden Triangle, British Columbia

The Hat Project is located within the Golden Triangle of northwestern British Columbia, a well-established metallogenic district that hosts numerous porphyry and epithermal deposits, including operating mines and several advanced-stage copper-gold-silver projects. The district provides important regional geological and exploration context for the Hat Project. Figure 23-1 shows mines, proposed mines, and selected exploration projects located in the Northwest Region of British Columbia.

The Star Project, held by Star Copper Corp., is an early-stage copper-gold porphyry exploration project in northwestern British Columbia that shares a property boundary with the Hat Project. Based on the public disclosures reviewed, no current NI 43-101 mineral resource estimate has been identified for the Star Project. Star Copper describes the project as early stage and reports approximately 19,000 m of historical drilling.



Source: After Corcoran (2025).

Figure 23-1: Mines, Proposed Mines, and Selected Exploration Projects in the Northwest Region of British Columbia
Star Copper—Star Project

23.2 Newmont—Red Chris Mine

The Red Chris Mine is an operating copper-gold mine in northwestern British Columbia, owned by Newmont Corporation (70%) and Imperial Metals Corporation (30%). Reported open pit mineral reserves are 43.1 Mt grading 0.43% Cu and 0.37 g/t Au, while reported underground mineral reserves are 245.3 Mt grading 0.52% Cu and 0.64 g/t Au. Reported Indicated mineral resources include 478.1 Mt grading 0.34% Cu and 0.34 g/t Au and Inferred mineral resources of 88.7 Mt grading 0.36% Cu and 0.35 g/t Au (Newmont Corporation, 2026).

23.3 Teck—Schaft Creek

The Schaft Creek Project is an advanced-stage copper-gold-molybdenum-silver porphyry project in northwestern British Columbia. The project is operated by Teck, which holds a 75% interest, with Copper Fox Metals holding the remaining 25% interest. The updated January 15, 2021, mineral resource estimate reported Measured and Indicated Mineral Resources of 1,346 Mt grading 0.26% Cu, 0.16 g/t Au, 0.017% Mo, and 1.25 g/t Ag, with an additional 344 Mt of Inferred Mineral Resources grading 0.17% Cu, 0.11 g/t Au, 0.013% Mo, and 0.84 g/t Ag (Tetra Tech Canada Inc, 2021).

23.4 Teck and Newmont—Galore Creek

The Galore Creek Project is a large undeveloped copper-gold-silver project in northwestern British Columbia held through Galore Creek Mining Corporation, a 50:50 partnership between Teck and Newmont. The 2023 mineral resource statement reported Measured Mineral Resources of 425.7 Mt grading 0.44% Cu, 0.29 g/t Au, and 4.1 g/t Ag, Indicated Mineral Resources of 771.2 Mt grading 0.47% Cu, 0.22 g/t Au, and 4.8 g/t Ag, for a combined Measured and Indicated Mineral Resource of 1,196.8 Mt grading 0.46% Cu, 0.25 g/t Au, and 4.5 g/t Ag. Inferred Mineral Resources were reported as 237.8 Mt grading 0.26% Cu, 0.19 g/t Au, and 2.6 g/t Ag (Galore Creek Mining Corporation, 2023).

23.5 Brixton Metals—Thorn Project

The Thorn Project is a district-scale exploration property owned by Brixton Metals. The June 23, 2021, NI 43-101 technical report reported an Inferred Mineral Resource of 8.078 Mt grading 94.01 g/t AgEq, containing approximately 24.415 Moz AgEq. The resource includes the Oban, Glenfiddich, and Talisker zones (Brixton Metals Corporation, 2021).

24 OTHER RELEVANT DATA AND INFORMATION

No other relevant data or information to be reported.

25 INTERPRETATION AND CONCLUSIONS

25.1 Geology, Mineralization, and Exploration

The geology, mineralization, and exploration history of the Hat Project are described in detail in Sections 7, 8, and 9 of this Technical Report. The interpretations in those sections are supported by diamond drilling, subsurface sampling, geophysical surveys, geochemical data, and geological mapping.

Based on the available information, the QP does not identify any geological or exploration-related risks that would materially impact continued exploration or development of the Hat Project.

25.2 Mineral Resource

The Hat Project exhibits strong geological continuity and consistent distribution of copper, gold, silver, cobalt, and scandium grades. Integration of geochemistry, geophysics, lithology, and grade-distribution patterns supports a coherent genetic model in which the spatial distribution of commodities is predictable within the currently drilled areas.

The 2024–2025 drilling program increased total drilled metres by approximately 38%, strengthening the robustness of the Mineral Resource estimate and materially extending the mineralized envelope beyond the previous resource frame. Mineralization intersected outside the 2024 resource shell highlights additional upside potential.

The updated Mineral Resource model achieved higher levels of classification. The Project now contains a substantial Measured and Indicated inventory, meeting the 20% error at 80% confidence criterion at a production rate of 60 kt/d. This provides a solid foundation for advancing to pre-feasibility studies, where the target confidence level is 10%–15% error at 90% confidence.

Scandium represents an additional asset to the Project. Although present at low grades, it's persistent in situ distribution and demonstrated metallurgical recoverability indicate potential for supplementary value, with scalability to industrial extraction.

Risks identified for the Mineral Resource are:

- Mineral Resources are estimates and may be revised as new drilling data refine geological and grade-distribution interpretations.
- As a low-grade deposit, the Hat Project is sensitive to key modelling inputs (i.e., cut-off grade, interpolation strategy), and uncertainties remain inherent at this stage of development.
- Future drilling may identify zones of lower continuity or higher-grade variability not captured in the current model.
- Overreliance on early-stage assumptions without ongoing validation could lead to misclassification or suboptimal mining planning.

- Delays in integrating new drilling data may limit the ability to adjust exploration priorities and development strategies effectively.

Opportunities identified for the Mineral Resource are:

- Newly discovered mineralization outside the 2024 pit shell provides clear potential to expand the Mineral Resource base.
- The increase in Measured and Indicated classifications supports the initiation of pre-feasibility-level studies.
- Current and planned drilling programs offer a cost-effective means to grow the Mineral Resource and upgrade classification.
- Additional geotechnical, hydrogeological, and metallurgical data collection will further support future technical evaluations and derisk the Project.

25.3 Mining Methods

The mining methods developed for the Hat Project are considered appropriate for a PEA-level evaluation and support a conventional large-scale open-pit truck-and-shovel operation at a nominal processing rate of 120,000 t/d over a 25-year mine life. Pit optimization, preliminary pit design, stockpiling strategy, production scheduling, and conceptual fleet selection indicate that the Project has the potential to support a technically viable mine plan based on the current Mineral Resource and economic assumptions.

However, the mine plan remains conceptual in nature and is subject to important uncertainties typical of this stage of study. In particular, pit slope design is based on limited geotechnical information, the production schedule is based primarily on optimized pit shells and a final pit design rather than a fully engineered sequence of operational phases, and equipment sizing has not yet been validated through detailed haulage or fleet software simulation. Additional geotechnical, hydrogeological, operational mine planning, and engineering work will therefore be required to refine the mine design, support the production schedule, and improve confidence in the capital and operating cost estimates in future study stages.

25.4 Mineral Processing and Metallurgical Testing

The metallurgical test work completed to date on material from the Hat Deposit is limited in scope and supports only a conceptual process design appropriate for a Preliminary Economic Assessment. The scoping-level results indicate that copper-bearing material from the deposit is expected to respond to conventional recovery methods in a manner generally consistent with other copper-gold porphyry deposits. On this basis, a conceptual flowsheet incorporating conventional unit operations and preliminary equipment sizing has been developed.

The available metallurgical testwork provides the basis for the evaluation of all three processing scenarios considered in the PEA. Scenario A1 reflects the recoveries achieved in the current test program and represents a direct testwork-based case. Scenario A2, which has been selected as the base-case flotation

scenario, reflects expected flotation performance following further optimization and development work. Although the current program did not include a flotation optimization campaign specifically designed to demonstrate the recoveries assumed in Scenario A2, the improvements relative to Scenario A1 are considered modest and reasonable, and the assumed recoveries remain conservative relative to those commonly achieved in comparable copper-gold porphyry flotation operations. Scenario B is also supported at a conceptual level by the available testwork and incorporates additional downstream hydrometallurgical processing for cobalt and scandium recovery; however, this scenario is inherently more complex and carries a higher degree of technical uncertainty than the flotation-only cases.

However, the current metallurgical database is not sufficient to support the detailed design of process facilities. The available test work does not adequately define the variability of deposit material, the mineralogical and liberation characteristics of the economic minerals, the comminution properties required for circuit design, or the flotation response needed to establish optimized and reproducible plant operating parameters.

Testing completed to date also did not permit meaningful optimization of reagent selection, reagent dosage, or mineral-specific flotation strategies. As a result, the provisional design criteria currently reflect relatively high reagent consumptions, which may not represent optimized process performance. Additional testing is expected to improve understanding of the process response and may reduce reagent consumption and refine recovery assumptions.

At the PEA stage, tailings management has been considered on the basis of conventional slurry deposition in a tailings storage facility, with reclaim water returned to the process plant for recycling. This approach is considered appropriate for this level of study; however, alternative tailings management approaches, including filtered dry-stack tailings, have not yet been evaluated in sufficient detail.

Overall, the metallurgical and process design work completed to date is adequate to support a PEA-level assessment of the three processing scenarios; however, additional metallurgical testing will be required to confirm the optimized flotation performance assumed in the base-case Scenario A2 and to further de-risk the downstream hydrometallurgical recovery stages included in Scenario B.

25.5 Infrastructure

The infrastructure concept for the Hat Project is considered suitable for a large-scale open-pit copper-gold operation in northwestern British Columbia. Existing access via Telegraph Creek Road and Golden Bear Road provides regional connectivity, while a new approximately 54 km all-season access road is proposed to connect the site to that network.

Grid power from BC Hydro is the preferred long-term supply option. Preliminary review indicates that a new transmission connection must be included in project development, with further work required to confirm interconnection, routing, and capacity.

The site layout reflects a conventional mine arrangement. The primary crusher, process plant, stockpiles, waste storage facility, tailings storage facility, and support infrastructure have been positioned to reduce

material handling distances and maintain appropriate separation between operating and accommodation areas. The positioning of the facilities is preliminary and further studies will be need to optimize the location and arrangements. A pit-proximal primary crusher with conveying to the plant is considered appropriate for the planned scale of operation.

Logistics are considered manageable based on the existing regional road network and the proposed site access road. Truck haulage of concentrate to an established port facility in northwestern British Columbia is considered a practical basis for the PEA.

Further work is required to refine infrastructure design, confirm the power supply strategy, and complete the geotechnical, geochemical, and environmental investigations required for the next stage of study. Overall, the infrastructure described herein is considered technically feasible at the PEA level.

25.6 Tailings Storage Facility

The tailings storage facility concept designed for the Hat Project is considered appropriate for a PEA-level assessment. An alternatives assessment evaluating several locations and deposition options led to a cycloned sand tailings facility as the preferred option. This configuration features a starter dam constructed from NAG waste rock or borrow material, followed by centerline raises using compacted cycloned sand. The conceptual design integrates reclaim water return, seepage collection, diversion of non-contact runoff, alongside design criteria intended to support geotechnical stability and water management.

The TSF design is conceptual, as it was developed without site-specific geotechnical or hydrogeological field investigations, tailings property characterization, or geochemical studies to delineate PAG versus NAG materials. Opportunities exist to optimize all design aspects as updated site characterization and assessments become available. Specifically, additional work is required to confirm foundation conditions, dam construction materials, tailings geotechnical properties, ARD potential, hydrology and hydrogeology, seismic design criteria, water management requirements, and closure performance. As the facility falls within a preliminary ‘High’ to ‘Extreme’ consequence classification, subsequent study stages are needed to refine the design and support compliance with applicable regulatory requirements and industry standards.

25.7 Environmental Studies, Permitting, and Social or Community Impact

The Hat Project is expected to require both federal and provincial environmental assessment and permitting, together with a broad range of additional authorizations for mine construction and operation. While the current work identifies a viable regulatory pathway, substantial additional baseline studies and technical investigations will be required to support future design, effects assessment, consultation, and permitting. In particular, further work is needed in hydrology, hydrogeology, water quality, aquatic studies, wildlife, vegetation, air quality, meteorology, geochemistry, socioeconomic conditions, and closure planning. Overall, the environmental and permitting framework is considered manageable at the

PEA level, but it remains subject to significant uncertainty typical of this stage of study and will require considerable advancement in subsequent phases.

The environmental, permitting, and social review completed for the Hat Project is considered adequate to support a PEA-level assessment, but remains at an early stage and is based primarily on desktop review, preliminary regulatory analysis, and limited field-based work. Existing information has identified key environmental and permitting considerations relevant to Project development, as described in the following subsections.

25.7.1 Environmental Conclusions

Physical Environment

The Hat Project is located in a relatively pristine environment in northwest British Columbia's Interior Plateau, with elevations ranging from 675 to 1,500 masl. The area is entirely forested with several small streams draining to Hatchau Lake and Little Tahltan River. Two drainage systems are present: the Hackett River system draining to the Sheslay River, and the Little Tahltan River flowing to the Tahltan River, both ultimately draining to the Pacific Ocean through the Alaskan Panhandle.

Baseline environmental studies completed to date include a desktop review of existing environmental information (Wildwood Engineering & Environmental Ltd., 2025). However, significant additional baseline work is required before environmental assessment submission, including 12-24 months of field data collection for meteorology, air quality, surface water, groundwater, and biological resources.

Biological Environment

The Project area contains several environmental components that will require management and mitigation of potential impacts associated with further development:

Species at Risk

The Project overlaps Ungulate Winter Range for Stone's sheep (*Ovis dalli stonei*) and the range for the Level-Kawdy herd of northern caribou (*Rangifer tarandus*). Fish species of concern identified through desktop information include cutthroat trout (*Oncorhynchus clarkii*, Blue-listed), eulachon (*Thaleichthys pacificus*, Endangered/Threatened), green sturgeon (*Acipenser medirostris*, Special Concern), longfin smelt (*Spirinchus thaleichthys*, Blue-listed), and giant threespine stickleback (*Gasterosteus aculeatus*, Special Concern). Confirmation of suitable habitat or presence of these species, or others, would be part of the required field data collection.

Endangered Ecosystems

Two classified polygons for secured endangered species/ecosystems fall within the claim boundary. No old growth management areas are present in the vicinity.

Wildlife Species of Management Concern

Caribou (*Rangifer tarandus*), grizzly bear (*Ursus arctos horribilis*), moose (*Alces alces*), mountain goat (*Oreamnos americanus*), Stone's sheep (*Ovis dalli stonei*), Columbia spotted frog (*Rana luteiventris*), long-toed salamander (*Ambystoma macrodactylum*), western toad (*Anaxyrus boreas*), wood frog (*Rana sylvatica*), and bank swallow (*Riparia riparia*) have been identified as potentially present based on desktop studies.

Geochemical Considerations

Comprehensive geochemical characterization of waste rock, ore, overburden, and tailings is required to assess metal leaching and acid rock drainage potential. This work is critical for mine design, waste management planning, and closure considerations.

Key Environmental Risks

Uncertainties inherent in the current level of available environmental baseline data present potential design risk as follow:

- Species at risk habitat may constrain mine footprint and infrastructure placement.
- Two endangered ecosystems within the claim boundary require avoidance or significant mitigation.
- Remote location and limited baseline data create uncertainty in impact predictions.

25.7.2 Permitting Pathway Conclusions

Major Authorisations

Based on conceptual design and estimated production volumes, which exceed provincial regulatory thresholds, the following authorizations are required:

- Federal *Impact Assessment Act* Decision Statement and/or Provincial Environmental Assessment Certificate. Hat Project is expected to proceed through a substituted or coordinated environmental assessment process, with the government of B.C. and Canada having a shared cooperation agreement to streamline the completion of environmental assessments when required under both jurisdictions.
- Joint *Mines Act / Environmental Management Act* Application for provincial operating permits, through the BC Major Mines Office coordinated process.
- Federal *Fisheries Act* Authorizations under Sections 34 and 35 are anticipated for works contributing deleterious material to fish habitat and harmful alteration, disruption, or destruction of fish habitat. The Hat Project will also be subject to *Metal and Diamond Mines Effluent Regulations*.

Additional Permits

Additional federal and provincial permits, licenses, and approvals will be procured as required, including but not limited to: *Species at Risk Act* Permit, *Migratory Birds Convention Act* Permit, *Explosives Act* licenses, *Environmental Management Act* air and water waste discharge permits, *Water Sustainability Act* approvals, *Forest Act* licenses, and *Heritage Conservation Act* permits.

Permitting Timeline

The environmental assessment and permitting process for a major mining project in BC is complex and highly variable, and can take well over a decade from submission to construction commencement for a greenfield project. The BC and federal governments have committed to reducing major mine permitting timelines; however, until regulatory engagement has been initiated and a clear pathway defined, timelines remain difficult to predict with confidence. Under the 2018 *Environmental Assessment Act*, legislated phase timelines establish a statutory process floor, with the assessment process typically ranging from three to five years under normal circumstances. The subsequent joint *Mines Act* / *Environmental Management Act* application through the BC Major Mines Office does not carry equivalent statutory timelines; review duration is project-specific and ranges from 12 to 24 months for a well-prepared application, though this should not be relied upon as a firm estimate at this stage. The total timeline from EA submission to receipt of key construction authorizations for a project of this nature and complexity should be conservatively estimated at seven to ten or more years. Early initiation of baseline studies is critical to support timely application submission and to minimise overall permitting timelines, and the confidence of the overall timeline will continue to increase as the Hat Project progresses through prefeasibility and baseline programs and permitting applications are instantiated.

Key Permitting Risks

- Environmental assessment timeline uncertainty is inherent to major project reviews.
- First Nations consultation outcomes may affect project design and economics.
- Regulatory framework changes at federal and provincial levels may affect permitting requirements.
- Multiple authorization requirements create interdependencies that may affect project scheduling.

25.7.3 Social and Community Conclusions

First Nations

The Hat Property is located within the Traditional Territories of the Tahltan Nation and the Taku River Tlingit First Nation. Meaningful consultation with both First Nations is required under both the *BC Environmental Assessment Act* and the federal Impact Assessment Act, with the goal of gaining informed consent for the Project.

Key consultation considerations include:

- Identification of valued components and Traditional Use areas.
- Incorporation of Traditional Knowledge into environmental and social risk evaluations.
- Consideration of existing land and resource use plans, including the Taku River Tlingit First Nation and B.C. (2011) shared decision-making framework and the Tahltan Nation Phase 1 Land Use Plan under development with the provincial government.
- Evaluation of the potential to engage in mutually beneficial economic agreements with participating First Nations.

Local Communities

The closest communities are Dease Lake (95 km northeast) and Telegraph Creek (40 km south). Local community and stakeholder engagement will be conducted to comply with assessment and permitting standards and to address potential concerns related to Project development.

Cultural Heritage

Four pre-contact archaeological sites have been recorded within the Project area from previous investigations completed under Permits 2018-219 and 2022-470. Recommendations include avoidance of identified sites, alteration permits and site monitoring as necessary, additional assessments if planned developments are sited within these identified areas, and consultation with First Nations for preferred management approaches.

Trapline Tenures

The Project overlaps three trapline tenures (TR0622T017, TR0626T001, TR0626T003). Formal engagement with tenure holders is recommended with the goal of developing mutual agreements or letters of understanding.

Health and Socioeconomic Effects

A human health and socioeconomic study will likely be required to forecast and describe community health and socioeconomic changes resulting from Project development. Identified issues should be addressed through proposed mitigative measures.

Key Social and Community Risks

- Lack of consent of applicable First Nations will prevent project advancement.
- Community concerns regarding environmental impacts, employment, and local benefits may require additional mitigation not currently contemplated.
- Trapline tenure holder agreements may affect access and operational considerations.

25.8 Economic Analysis

The economic analysis indicates that the Hat Project has the potential to support positive economic performance at the PEA level under all three scenarios evaluated. Using analyst consensus metal price assumptions, the Project returns a post-tax NPV(5%) of C$4.96 billion for Scenario A1, C$6.73 billion for Scenario A2, and C$7.27 billion for Scenario B, with corresponding post-tax IRRs of 19%, 23%, and 19%, respectively. Among the cases evaluated, Scenario A2 provides the strongest overall return profile and is considered the preferred base-case development scenario, while Scenario B offers the highest NPV and cash flow potential but with greater capital intensity, process complexity, and execution risk.

The Project economics are primarily supported by copper and gold, with additional contributions from silver and cobalt, and in Scenario B from scandium oxide. Sensitivity analysis indicates that Project value is most strongly influenced by changes in the overall metal price deck, followed by head grade, exchange rate, capital cost, and operating cost assumptions. Although the results demonstrate robust economic potential at the PEA level, they remain subject to the uncertainties inherent in an early-stage study, including the use of Inferred Mineral Resources, preliminary capital and operating cost estimates, and processing assumptions that will require further refinement in subsequent study stages.

26 RECOMMENDATIONS

The QPs recommend that the next phase of work focus on advancing geological confidence, expanding the Mineral Resource base, and supporting pre-feasibility-level evaluations.

26.1 Mineral Resource Estimate

26.1.1 Drilling Program

Given the high uncertainty in expected outcomes and the need to improve spatial continuity, the next stage of exploration should include approximately 32 new drill holes, ranging from 340 m to 900 m deep, for a total planned metreage of ~17,500 m. Considering operational flexibility, the recommended target range for the program is 17,500 m to 22,000 m.

To maximize geological value, the current fan-drilling approach should be reviewed. Instead, drilling should focus on filling remaining spatial gaps between existing drilling pods, targeting both copper and gold domains. This strategy is expected to improve continuity, reduce classification uncertainty, and support future Mineral Resource estimate upgrades.

26.1.2 Technical Data Collection

Additional geotechnical, hydrogeological, and metallurgical testwork is recommended to support pre-feasibility-level engineering and to further derisk the Project.

26.1.3 Ongoing Model Refinement

New drilling data should be incorporated promptly into the geological and resource models to validate assumptions, refine domain boundaries, and support improved classification confidence.

26.2 Mining Methods

26.2.1 Geotechnical stability

To refine the preliminary geotechnical results developed for this PEA and support advancement to the next stage of study, the following work is recommended:

- Acquire additional oriented core, downhole televiewer data, and detailed structural measurements to improve characterization of discontinuity sets, persistence, spacing, and condition
- Complete additional laboratory testing including uniaxial compressive strength (UCS), triaxial strength, direct shear, and intact tensile strength testing for the principal rock types and geotechnical domains

- Advance the structural interpretation to better constrain potential planar, wedge, and toppling failure mechanisms and improve the kinematic model
- Refine and calibrate domain-specific rock mass parameters, including MRMR adjustment factors such as structure orientation, weathering, and groundwater influence
- Assess overburden, weathered zones, and structurally disturbed domains in greater detail to confirm slope design assumptions in weaker materials
- Complete updated slope stability assessments based on the next-phase pit design, including 2-d numerical modelling where appropriate, once material properties and structural controls are better constrained
- Integrate geotechnical, hydrogeological, and blasting considerations into the next phase of pit slope design to support refinement of inter-ramp angles, bench face angles, berm widths, and catchment performance.

26.2.2 Mine Design and Scheduling

The mine design and production schedule developed for this PEA are appropriate for a preliminary assessment; however, additional engineering is required to advance the Project to the next stage of study. The current mine plan is based primarily on the optimized pit shells and a single final pit design, and does not yet reflect a fully engineered sequence of operational pushbacks or stages. The following work is recommended:

- Develop a phased pit design based on practical mining stages and pushbacks, rather than relying primarily on pit optimization shells for scheduling
- Update the production schedule using the phased design to better reflect operational mining considerations, including mining access, ramp placement, working width, phase interaction, bench development, and equipment productivity
- Review and optimize the mining sequence to confirm the balance between early value extraction, stripping requirements, stockpile management, and practical mining execution
- Complete a more detailed fleet sizing and selection study, including evaluation of loading unit and haul truck matching, equipment availability and utilization, cycle times, queueing, delays, standby requirements, and maintenance assumptions
- Undertake haulage and fleet simulation or equivalent detailed productivity analysis to validate material movement requirements, equipment quantities, and schedule achievability
- Review equipment purchase timing and replacement strategy to optimize the balance between initial and sustaining capital
- Refine waste dump and ore stockpile development schedules in conjunction with the staged mine plan to confirm capacity, haulage requirements, and operational flexibility
- Review dilution and ore loss assumptions relative to the current mineral resource model, bench geometry, and blasting practices to confirm their suitability for the next stage of study

- Review drill-and-blast practices, including wall control requirements, to support both productivity and pit slope performance
- Reassess cut-off strategy, stockpiling approach, and processing schedule integration in conjunction with updated metal prices, recoveries, and processing assumptions
- Review throughput and production rate assumptions together with the process team to confirm the preferred mine-production strategy and overall project value

26.3 Infrastructure

For the pre-feasibility stage, infrastructure work should focus on confirming key design assumptions and advancing engineering to a higher level of confidence. The principal recommendations are as follows:

- Complete preliminary engineering and field investigations for the proposed site access road, including route optimization, geotechnical assessments, hydrological studies, and preliminary bridge design for the Beatty Creek crossing
- Advance electrical supply studies in coordination with BC Hydro to identify a preferred grid interconnection point, evaluate transmission routing alternatives, and develop a preliminary site power distribution concept; initiate early discussions with the power authority regarding transmission requirements and begin the necessary application processes
- Refine the overall site layout through engineering trade-off studies addressing plant location, conveyor alignment, waste storage facility placement, and internal haul road configuration
- Develop preliminary engineering designs for fuel storage facilities, site utilities, camp infrastructure, and other supporting facilities to support pre-feasibility level capital and operating cost estimates

Completion of these activities will allow the infrastructure design to be advanced to a level appropriate for pre-feasibility evaluation and will improve the accuracy of capital and operating cost estimates.

26.4 Mineral Processing and Metallurgical Testing

To support advancement of the Hat Project beyond the PEA stage, a comprehensive metallurgical test program is recommended to better define ore characteristics, process response, and design criteria for the selected processing flowsheet. The recommended work should include the following:

- Confirm the geological character of the deposit so that metallurgical samples are representative of the material expected to be mined, particularly during the first 3 to 5 years of the mine plan, and assess whether material characteristics are likely to vary materially with lithology or over time.
- Expand mineralogical characterization across a broader portion of the deposit to better define the occurrence, distribution, associations, and liberation characteristics of the valuable minerals, including copper-, gold-, and cobalt-bearing phases.

- Define the occurrence, mineral associations, and liberation characteristics of the copper minerals to support selection and optimization of the processing flowsheet.
- Define the occurrence and liberation characteristics of gold, including assessment of whether nugget gold is present and whether a gravity recovery circuit should be incorporated; this should include gravity-recoverable gold testing.
- Determine the crushing work index of the ore to support crushing circuit design, as this information is not currently available.
- Complete a full suite of grinding test work, including Bond Ball Mill Work Index testing and related comminution parameters, together with determination of the Abrasion Index, to support grinding circuit design.
- Undertake systematic flotation test work using appropriate conventional and alternative reagent suites to confirm the suitability of the reagents used in the scoping work and to optimize reagent dosages, conditioning times, grind size, regrind size, flotation kinetics, cleaner stage requirements, and circuit configuration.
- Complete locked-cycle tests to demonstrate process stability and reproducibility, and to establish a reliable mass balance including recycle streams, grades, and recoveries.
- Undertake variability testing using samples from different parts of the deposit and across a range of head grades.
- Evaluate alternative collector and depressant reagents, including investigation of possible replacements for sodium cyanide in the copper flotation circuit, where appropriate.
- Investigate relationships between feed grade and recovery, including any relevant correlations between copper and gold recovery.
- Obtain sufficient concentrate sample material to conduct settling and filtration testing in support of concentrate dewatering design.
- Obtain sufficient concentrate material for submission to potential smelter customers for preliminary market evaluation and assessment of concentrate quality, including any potential penalty elements.
- Investigate the geological occurrence and extent of platinum group elements (PGE) in the deposit, followed by targeted test work to assess whether any economically recoverable value may be present.

The results of the recommended test work should be used to refine the metallurgical design criteria, confirm the preferred process flowsheet, improve recovery and reagent consumption assumptions, evaluate tailings management alternatives, and support a higher-confidence level of engineering in subsequent study phases.

26.5 Tailings Storage Facilities

26.5.1 Geochemistry and Waste Rock Management

- Static and Kinetic Geochemical Testing: Initiate a comprehensive program for both waste rock and tailings to determine their metal leaching and acid rock drainage (ML/ARD) potential.
- ARD Onset and Metal Leaching Assessment: Conduct testing to determine the proportions of potentially acid-generating (PAG) and non-PAG waste, the delay time before the onset of ARD, and the potential for metal leaching under neutral pH conditions.
- Waste Management Planning: Use the geochemical data to develop waste rock management and segregation strategies to guide site layout and closure planning.

26.5.2 Tailings Storage Facility

- Site Investigations: Perform detailed geotechnical, hydrogeological, and geophysical field investigations at the proposed TSF location and potential alternative sites.
- Laboratory Testing: Conduct laboratory tests on tailings and potential foundation materials to characterize their engineering properties, such as settling, consolidation, and strength.
- Alternative Assessment: Carry out a formal TSF technology and siting alternative assessment to optimize the design and satisfy regulatory requirements.
- PFS Level Engineering Studies: Advance the TSF design based on site-specific data, including stability analyses, seepage modelling, tailings deposition, dam sections and materials, and water management for the facility.

26.5.3 Water Management Infrastructure

- Meteorological and Climate Data: Install additional meteorological stations and analyze existing data to refine site-specific climate parameters, including precipitation and evaporation rates, while accounting for climate change.
- Hydrological Monitoring: Expand the surface water and groundwater monitoring network to collect long-term data for baseline characterization and to support infrastructure design.
- Site-Wide Water Balance and Water Quality Model: Develop an integrated water balance and water quality model to simulate water movement and quality throughout the site under various climatic scenarios.
- Infrastructure Sizing and Optimization: Utilize refined hydrological data and modelling to size and optimize water management infrastructure, such as diversion channels, collection ponds, sumps, and the water treatment plant.

26.6 Environmental Studies, Permitting, and Social or Community Impact

The desktop environmental baseline studies completed to date for the Hat Project provide a foundational understanding of the existing environmental conditions and will be used to inform the scope of on-site data collection. However, to inform regulatory process and scope, additional field data collection work, permitting applications commencement, and stakeholder engagement are required to advance the Project to the next study phase and support regulatory approval. The following work is recommended:

26.6.1 Environmental Baseline Studies Continuation

Baseline environmental data must be representative of seasonal and interannual variability to support impact assessment, regulatory review, and compliance during operations. The current baseline program, as described in detail in Section 20, should be expanded and continued to meet these three needs. Recommended activities include:

- Install and maintain meteorological station for air quality and climate monitoring.
- Establish hydrological monitoring stations and commence data collection for surface water quantity and quality.
- Complete groundwater well installation and monitoring program to establish baseline conditions.
- Conduct field surveys for aquatic species and fish habitat assessment with emphasis on species at risk or of special concern including cutthroat trout (*Oncorhynchus clarkii*), eulachon (*Thaleichthys pacificus*,), green sturgeon (*Acipenser medirostris*), longfin smelt (*Spirinchus thaleichthys*), and giant threespine stickleback (*Gasterosteus aculeatus*).
- Conduct terrestrial wildlife surveys with emphasis on species at risk (Stone's sheep [*Ovis dalli stonei*], grizzly bear [*Ursus arctos horribilis*], northern caribou [*Rangifer tarandus*])
- Complete vegetation and ecosystem mapping, including endangered ecosystem polygons identified within the claim boundary.
- Conduct soil characterization and potentially acid generating (PAG) material assessment.

Baseline monitoring should continue throughout the environmental assessment application review period and will be defined during early regulatory engagement. Cost estimates for baseline studies are preliminary at this PEA stage and will be refined based on final study scope; a preliminary budget should be developed for the prefeasibility stage.

Timeline: The anticipated timeline for a complete baseline program prior to regulatory application submission is 18 to 24 months.

26.6.2 Permitting Pathway

The Hat Project will require multiple federal and provincial authorizations prior to construction and operation. It is recommended to initiate early engagement with regulators to confirm the permitting pathway, data collection scope, and refine regulatory timelines during the prefeasibility stage.

Early coordination with the federal and provincial agencies to determine the pathway under the Canada–British Columbia Impact Assessment Cooperation Agreement (Cooperation Agreement) is recommended to facilitate substitution or coordination. The coordinated process between the provincial and federal impact assessment runs concurrently with both agencies maintaining process with a common information requirements document to guide data collection. Under a substituted process, the federal government defers to the BC government process and timelines.

Note the 2024 amendments to the *Federal Impact Assessment Ac*t and ongoing regulatory streamlining efforts may affect assessment timelines and requirements. The federal and provincial Major Projects Offices remit is evolving quickly, and it is recommended to inquire if the Project is eligible to be included in either of these offices to significantly streamline approvals and support with logistics for Project construction, which could include ancillary project support (i.e., road upgrades) or supply chain support.

The following permitting pathway is recommended based on the regulatory framework in place at the time of writing:

- Federal *Impact Assessment Act* process: It is recommended that an Initial Project Description be submitted to the Impact Assessment Agency of Canada (IAAC), with designation as a reviewable project anticipated based on production thresholds.
  - Timeline: under coordination in the Cooperation Agreement expect 24 to 36 months from submission to Decision Statement; under substitution, the provincial process and timelines govern.
- Provincial *Environmental Assessment Act* process: It is recommended that an application for an Environmental Assessment Certificate be submitted to the BC Environmental Assessment Office (BC EAO) under the Cooperation Agreement to confirm a substituted or coordinated process.
  - Timeline: 18 to 30 months from submission to a BC Environmental Assessment Certificate issuance; under substitution, the EAO conducts the assessment on behalf of IAAC and this provincial timeline governs the federal and provincial processes.
- BC Major Mines Office (MMO) Joint *Mines Act* / *Environmental Management Act* application: It is recommended that this application be submitted in the later stages of the IAIA/BC EAO process. The application will establish conditions for construction, operation, monitoring, and closure.
  - Timeline: Unlike the EA process, there are no legislated statutory timelines for the joint application review; timeline is project-specific and determined through the Mine Review Committee process. For a well-prepared greenfield application, the review phase ranges from 12 to 24 months from submission to permit issuance, though this may extend

depending on application complexity, information requests, and the depth of First Nations consultation required.

- Additional federal authorizations (concurrent with the IAIA process): *Fisheries Act* section 34/35 authorizations (DFO), *Species at Risk Act* permits (if applicable), *Migratory Birds Convention Act* compliance, and *Metal and Diamond Mines Effluent Regulations* compliance. Other permits or approvals may be identified through early engagement with regulators.
  - Timeline: these will be done concurrently with the federal EA process or per the substituted provincial process.
- Additional provincial authorizations: Water Use Licences (*Water Sustainability Act*), Waste Discharge Permits (*Environmental Management Act*), Occupant Licence to Cut (vegetation clearing), and Mining Right of Way Permits (access). Other permits or approvals may be identified through early engagement with regulators.
  - Timeline: these will be done concurrently with the provincial EA process.

Permit application costs and associated consultant fees are preliminary at this PEA stage and will be refined during the prefeasibility stage once regulatory pathways are confirmed and application requirements are finalised.

First Nations Consultation

The Project is located within the asserted Traditional Territories of the Tahltan Nation and the Taku River Tlingit First Nation. Both Nations have established consultation protocols and land use planning interests that must be respected in advance of, throughout, and following the EA and permitting process and to incorporate over the project lifecycle. Consultation should be prioritised as the Hat Project progresses to the prefeasibility stage to support both regulatory requirements and inform the scope of baseline environmental studies. Recommended activities include:

- Formalise consultation with the Tahltan Nation and Taku River Tlingit First Nation under prescribed procedures of the *Environmental Assessment Act* and *Impact Assessment Act.*
- Engage on identification of valued components and Traditional Use areas.
- Incorporate Traditional Knowledge into baseline study design and impact assessment.
- Work toward benefits agreements or letters of understanding with both Nations.
- Review and incorporate the Tahltan Nation Phase 1 Land Use Plan, announced in June 2025 under co-development with the Province of BC, once enacted, as it may affect conservation and development objectives relevant to the Project.
- Consider the Taku River Tlingit First Nation shared decision-making framework established with BC.

Cost estimates for First Nations engagement, capacity funding, and potential impact benefit agreements are to be determined through negotiation; budget provision should be made during the prefeasibility stage. Timeline: ongoing throughout the assessment and permitting process; key agreements targeted prior to Environmental Assessment Certificate issuance.

Timeline: Engagement with First Nations is underway at an introductory level and will continue through the life of the project. Engagement for a regulatory purpose commences with the submission of the regulatory applications and is run by the regulatory body.

Trapline Tenure Holder Engagement

Three registered trapline tenures overlap the Project area, and tenure holders may be affected by Project development and associated access requirements. Early engagement with trapline tenure holders is recommended to identify potential conflicts and develop mutually acceptable agreements. Recommended activities include:

- Formalise engagement with tenure holders TR0622T017, TR0626T001, and TR0626T003
- Develop agreements or letters of understanding for access
- Address compensation and indemnity terms as applicable.

Agreements should be negotiated prior to commencement of field activities that may affect trapline operations. Cost estimates for tenure holder engagement are preliminary at this PEA stage and should be developed during the prefeasibility stage.

Archaeological and Cultural Heritage

Four pre-contact archaeological sites have been identified within the Project area through previous assessments conducted by Taiga Heritage Consulting Ltd. (2024). These sites are protected under the *Heritage Conservation Act*, and avoidance measures must be implemented to prevent disturbance unless otherwise authorized. Recommended activities include:

- Continue to implement avoidance measures for the four identified pre-contact archaeological sites
- Obtain alteration permits and conduct monitoring as necessary for any ground disturbance near sites
- Complete additional assessments if planned developments are sited in sensitive areas
- Consult with First Nations on preferred management approaches for sites not protected under the *Heritage Conservation Act*.

Cost estimates for archaeological management are preliminary at this PEA stage and will be refined based on final site layout; preliminary provision should be made during the prefeasibility stage.

Human Health and Socioeconomic Assessment

Project development may result in community health or socioeconomic changes in the region, including effects on local employment, infrastructure, services, and Traditional Land Uses. A human health and socioeconomic assessment are recommended to forecast potential changes and develop appropriate mitigative measures. Recommended activities include:

- Complete an impact study on community health and socioeconomic changes resulting from Project development
- Identify issues and develop mitigative measures in consultation with affected communities.

Cost estimates for this assessment are preliminary at this PEA stage and will be determined during the prefeasibility stage.

Environmental Effects Monitoring Program

An Environmental Effects Monitoring (EEM) program will be required during Project operation under the *Metal Mining Effluent Regulations* and permit conditions established through the environmental assessment and permitting process. The EEM program should be designed during the pre-feasibility stage and implemented prior to construction to establish baseline conditions. Recommended activities include:

- Develop a monitoring program to be implemented during operation in accordance with *Metal Mining Effluent Regulations* requirements
- Include water quality, water quantity, air quality, and biological species and/or habitat monitoring
- Establish baseline conditions prior to construction for comparison against operational data
- Enter all baseline environmental data into provincial database systems (EMS, WIDM, FDIS) as applicable.

Operating costs for the EEM program are preliminary at this PEA stage and will be quantified in operating cost estimates at the pre-feasibility stage based on final permit conditions, which are subject to the outcomes of the environmental assessment process and First Nations consultation.

Closure and Post-Closure Planning

Closure planning should advance concurrently with mine design to ensure that reclamation objectives, criteria, and cost estimates are integrated into the Project economics. Financial security will be required under the *Mines Act* and *Environmental Management Act* to ensure that reclamation obligations can be fulfilled. Recommended activities include:

- Develop a preliminary closure plan with reclamation objectives and criteria
- Complete reclamation cost estimation for financial security requirements
- Plan for long-term post-closure monitoring, with duration informed by geochemical characterisation results
- Quantify financial provision for closure and post-closure at the prefeasibility stage, noting that security amounts will be determined based on the approved closure plan.

Reclamation cost estimates and financial security amounts are preliminary at this PEA stage and will be quantified at the prefeasibility stage based on approved closure plans and regulatory requirements.

26.6.3 Regulatory Compliance Framework

Upon issuance of a federal Decision Statement under the Impact Assessment Act, an Environmental Assessment Certificate under the *BC Environmental Assessment Act*, a BC Mines Act permit, and ancillary

federal and provincial authorizations (e.g. *Fisheries Act Authorization*), the Project will be subject to ongoing compliance and reporting obligations under multiple federal and provincial statutes. A compliance tracking system should be established to ensure that all permit conditions are met and reporting requirements fulfilled. Recommended activities:

- Establish a compliance tracking system for all permit conditions.
- Prepare for annual environmental quality review reporting during the assessment process.
- Plan for data submission to provincial or other required databases (Environmental Monitoring System, Species Inventory Database, Fish Inventory Data Queries).
- Obtain environmental impairment liability insurance prior to construction.
- Obtain directors and officers insurance covering environmental liabilities.
- Include environmental liability provisions in all contractor agreements.

Cost estimates for compliance programs are preliminary at this PEA stage and will be refined and included in operating cost estimates at the prefeasibility stage. The regulatory framework should be confirmed during the prefeasibility stage to ensure that permitting pathways and requirements reflect current legislation and agency expectations, including the 2024 amendments to the *Impact Assessment Act* and the anticipated enactment of the Tahltan Nation Phase 1 Land Use Plan

26.7 Economic Analysis

The economic analysis presented herein reflects a Preliminary Economic Assessment-level evaluation based on current assumptions and available data. As the Project advances, refinement of the economic model is warranted to improve the accuracy and reliability of the projected financial outcomes and to support subsequent phases of study.

- Update the economic model to reflect current and supportable assumptions for commodity prices, foreign exchange, and commercial terms, including concentrate treatment charges, payabilities, and pricing assumptions for secondary products such as cobalt and scandium oxide, where market transparency remains limited.
- Refine the fiscal framework applied in the model, including royalties, British Columbia mineral tax, federal and provincial income taxes, depreciation, and available tax pools, to improve the accuracy of after-tax cash flow projections and valuation metrics.
- Progressively update capital and operating cost inputs as engineering definition advances, with particular focus on key cost drivers including mining cost progression with depth, processing and reagent consumption, hydrometallurgical and scandium processing assumptions, sustaining capital timing, and closure cost provisions.
- Expand scenario and sensitivity analysis to further evaluate the Project's exposure to key economic variables, including commodity prices, exchange rates, capital and operating costs, metallurgical performance, and the timing and integration of the scandium recovery circuit.

- The current economic analysis incorporates Inferred Mineral Resources within the mine plan and cash flow model. As additional drilling is completed and resource classification improves, conversion of Inferred material to Indicated or Measured categories is expected to influence mine scheduling, production profiles, and the timing of cash flows, and should be incorporated in future updates to the economic model.

26.8 Summary

A phased program of technical studies and engineering work is recommended to advance the Hat Project to the next stage of evaluation. The proposed program includes additional drilling, Mineral Resource update, metallurgical and hydrometallurgical testwork, geotechnical and hydrogeological investigations, tailings and infrastructure studies, environmental baseline and assessment work, and completion of a pre-feasibility study. The estimated cost of the recommended program is $85.02 million, including contingency, as summarized in Table 26-1.

Table 26-1: Estimated Cost for Proposed Recommendations

Description	Cost ($M)
Exploration and Resource Upgrading	
PFS Infill and Geotechnical Drilling	14.00
Regional Exploration & Target Testing	15.00
Resource Estimate Update (MRE)	0.45
Metallurgy and Process Design	
Comminution and Ore Property Testing	1.50
Metallurgical PFS (Flotation & Cobalt)	2.50
Hydrometallurgical PFS (Scandium Recovery & SX)	2.25
Engineering and Infrastructure	
Geotechnical, Hydrogeological, and Terrain studies.	7.00
Geochemical Testing and Acid-Rock Drainage/ML Modelling	1.20
Site investigations (TSF, WSF, Borrow)	4.50
Hydrological, Meteorological & Glacier.	1.30
Infrastructure studies (Bridge, Road, Power)	6.50
Environmental, Social, and Permitting	
Social, Archaeological, and Heritage	1.80
Indigenous Participation Negotiations	1.20
Environmental baseline studies	5.00
Environmental assessment studies	2.50
Project Management and Studies	
Pre-feasibility Study Report & QPs	3.50
Product Marketing & Logistics Study	0.40
Tailings Trade-off Study (Dry-stack)	0.25
Subtotal	**70.85**
20% Contingency	14.17
Total	**85.02**

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28 CERTIFICATE OF QUALIFIED PERSONS

28.1 Shervin Teymouri, P.Eng.

I, Shervin Teymouri, of Vancouver, British Columbia, Canada, as an author of this report titled *National Instrument 43-101 Technical Report, Preliminary Economic Assessment and Updated Mineral Resource Estimate for the Hat Project, Northwest British Columbia, Canada*, with an MRE effective date of February 4, 2026, PEA effective date of March 22, 2026, and dated April 10, 2026 (this "Technical Report"), prepared for Doubleview Gold Corp. (Doubleview or the Issuer) do hereby certify that:

- I am a Principal Mining and Geotechnical Engineer with Mineit Consulting Inc., with a business address at Suite 1188–1095 West Pender Street, Vancouver, British Columbia, V6E 2 M6, Canada.
- I am a Geological Engineering graduate from the University of British Columbia (2005) and Master of Mining Engineering from the University of British Columbia (2007). I have been involved or associated with the mining industry since 2005, in China, Mexico, and Canada, in production roles for four years and 17 years in consulting. I have practiced my profession continuously since graduation.
- I am a Professional Engineer and member in good standing of the Engineers and Geoscientists BC (License # 35469).
- 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 fulfill the requirements to be a qualified person for the purpose of NI 43-101.
- I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
- My relevant experience includes more than 17 years in mining engineering, project development, capital and operating cost estimation, and economic evaluation for mineral projects in British Columbia and internationally, including technical studies prepared under NI 43-101.
- I am responsible for the preparation of Sections 1 through 5, 16, 19, and 21 through 27 of this Technical Report.
- I have not visited the Hat Project site.
- I have no prior involvement with the property that is the subject of the Technical Report.
- I am independent of the Issuer within the meaning of Section 1.5 of NI 43-101.
- I have read the News Releases dated March 2, 2026, and confirm these news releases are a fair and accurate summary of my sections of this Technical Report.
- As of the effective date of the Technical Report, and as of the date of this certificate, to the best of my knowledge, information, and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.
- I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication of the Technical Report in the public Company files on their websites.

Signed and dated this 10th day of April 2026 at Vancouver, British Columbia, Canada.

Original Signed and Sealed

Shervin Teymouri, P.Eng.

28.2 Tomasz Wawruch, FAusIMM.

I, Tomasz Wawruch, FAusIMM, do hereby certify that I am retained by Mineit Consulting Inc., with an office at 1188–1095 West Pender Street, Vancouver, British Columbia, V6E 2M6, Canada.

- This certificate applies to the technical report titled *National Instrument 43-101 Technical Report, Preliminary Economic Assessment and Updated Mineral Resource Estimate for the Hat Project, Northwest British Columbia, Canada*, with an MRE effective date of February 4, 2026, PEA effective date of March 22, 2026, and dated April 10, 2026 (this "Technical Report"), prepared for Doubleview (the "Issuer").
- I am a Fellow of the Australasian Institute of Mining and Metallurgy (FAusIMM), in good standing, membership number 221993. I hold a Master of Science degree in Applied Geology in Civil Engineering, awarded by the University of Warsaw in 1991. I also successfully completed postgraduate studies in Mining Geology for Open Pit Operations at the École des Mines de Paris during the 1992–1993 academic year.
- I have 32 years of professional experience in geology, including 23 years with the Anglo-American Group in Chile and South Africa, 2 years with Endeavour Silver in Mexico, and 7 years with Geo4M Consulting Services. My background combines academic training and extensive operational experience in mining, economic geology, marketable minerals, resource exploration and development, geological analysis and interpretation, mineral resource evaluation, risk quantification in mineral resource management, technical reporting, and the preparation of annual mineral resource statements.
- Over the course of my career, I have worked on a wide spectrum of mineralized systems, including: iron-oxide and hydrothermal systems, sedimentary, stratiform, and basin-hosted systems, supergene and oxide systems nickel and orogenic systems, skarn and carbonate-replacement systems, structurally controlled and alteration-dominated systems, in a variety of countries across Africa and South America.
- I have read the definition of "qualified person" set out in National Instrument 43-101 - Standards of Disclosure for Mineral Projects (NI 43-101) and certify that, by reason of my education, professional affiliation, and relevant experience, I satisfy the requirements to act as a Qualified Person for the purposes of NI 43-101.
- I conducted a personal inspection of the Hat Project property from 1 to 3 October 2025.
- I am responsible for Sections 1.3, 1.4, 1.5, 6, 7, 8, 9, 10, 11, 12, and 14 of the Technical Report. I am also responsible for the portions of Sections 25 and 26 that relate to exploration, drilling, and the estimation and classification of Mineral Resources.
- I am independent of the Issuer within the meaning of Section 1.5 of NI 43-101.
- I have had no prior involvement with the property that is the subject of the Technical Report, other than work completed in connection with the Technical Report.
- I have read NI 43-101 and Form 43-101F1, and the portions of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the effective date of the Technical Report, and as of the date of this certificate, to the best of my knowledge, information, and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

Signed and dated this 10th day of April 2026 at Vancouver, British Columbia, Canada.

Original Signed and Sealed

Tomasz Wawruch, FAusIMM

28.3 Marinus André de Ruijter, M.Sc., P.Eng.

I, Marinus André De Ruijter, M.Sc. (Met Eng.), P.Eng., do hereby certify that I am retained by Mineit Consulting Inc. as Principal Metallurgical Engineer, with an office at 1188–1095 West Pender Street, Vancouver, British Columbia, V6E 2M6, Canada.

- This certificate applies to the technical report titled *National Instrument 43-101 Technical Report, Preliminary Economic Assessment and Updated Mineral Resource Estimate for the Hat Project, Northwest British Columbia, Canada*, with an MRE effective date of February 4, 2026, PEA effective date of March 22, 2026, and dated April 10, 2026 (this "Technical Report"), prepared for Doubleview (the "Issuer").
- I am a registered Professional Engineer in good standing with Engineers and Geoscientists British Columbia (Registration No. 31031) and the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (Registration No. L4672). I hold a Master of Science in Metallurgical Engineering and a Bachelor of Science in Metallurgical Engineering from the University of the Witwatersrand, Johannesburg, South Africa, and a Bachelor of Science in Physics and Mathematics. I have practiced my profession continuously since 1979. My experience includes metallurgical testwork programs, interpretation of metallurgical data, process design, and engineering of base and precious metal projects in Canada, the United States, Mexico, Chile, and Peru.
- I have read the definition of "qualified person" set out in National Instrument 43-101—Standards of Disclosure for Mineral Projects (NI 43-101) and certify that, by reason of my education, professional affiliation, and relevant experience, I satisfy the requirements to act as a Qualified Person for the purposes of NI 43-101.
- I have not conducted a personal inspection of the Hat Project property.
- I am responsible for Sections 13.1 to 13.2, 17.1 to 17.6 and subsections in 1, 21, 25, 26, 27 of this Technical Report.
- I am independent of the Issuer within the meaning of Section 1.5 of NI 43-101.
- I have had no prior involvement with the property that is the subject of the Technical Report, other than work completed in connection with the Technical Report.
- I have read NI 43-101 and Form 43-101F1, and the portions of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101 and Form 43-101F1.
- I have read the News Releases dated March 2, 2026, and confirm these news releases are a fair and accurate summary of my sections of this Technical Report.
- As of the effective date of the Technical Report, and as of the date of this certificate, to the best of my knowledge, information, and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

Signed and dated this 10th day of April 2026 at Vancouver, British Columbia, Canada.

Original Signed and Sealed

Marinus André De Ruijter, P.Eng.

28.4 Andrew Carter, C.Eng., MIMMM, QMR, MSAIMM

I, EUR ING Andrew Carter, of Salisbury, Wiltshire, United Kingdom, as an author of this report titled *National Instrument 43-101 Technical Report, Preliminary Economic Assessment and Updated Mineral Resource Estimate for the Hat Project, Northwest British Columbia, Canada*, with an MRE effective date of February 4, 2026, PEA effective date of March 22, 2026, and dated April 10, 2026 (this "Technical Report"), prepared for Doubleview (the "Issuer") do hereby certify that:

- I am retained by Mineit Consulting Inc., with an office at 1188–1095 West Pender Street, Vancouver, British Columbia, V6E 2M6, Canada.
- I am a Chartered Engineer (C.Eng.), European Engineer (EUR ING) and a member of relevant professional organizations including MIMMM, MSAIMM, and SME. I hold the following degrees: B.Sc Minerals Processing, The University of Leeds 1980. I have 46 years of experience in Extractive Metallurgy, including Minerals Processing, Pyrometallurgy, Hydrometallurgy, and Bio-hydrometallurgy.
- I have read the definition of "qualified person" set out in National Instrument 43-101 - Standards of Disclosure for Mineral Projects (NI 43-101) and certify that, by reason of my education, professional affiliation, and relevant experience, I satisfy the requirements to act as a Qualified Person for the purposes of NI 43-101.
- I am responsible for Sections 13.3 and 17.7, and subsections in 1, 21, 25, 26, 27 of this Technical Report.
- I have had prior involvement with the property that is subject to this Technical Report.
- I have provided technical support to Sepro Laboratories in relation to preliminary metallurgical testwork on Hat samples, and participated in related discussions with Sepro and Doubleview.
- I am independent of Issuer within the meaning of Section 1.5 of NI 43-101.
- I have not conducted a site visited nor been engaged in any specific work at the Hat Project site.
- I have read NI 43-101 and Form 43-101F1 and the parts of the Technical Report for which I am responsible, and these sections have been prepared in compliance with that instrument.
- I have read the News Release dated March 2, 2026, and confirm this news release is a fair and accurate summary of my sections of this Technical Report.
- As of the effective date of the Technical Report, and as of the date of this certificate, to the best of my knowledge, information, and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.
- I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication of the Technical Report in the public Company files on their websites.

Signed and dated this 10th of April 2026 in Vancouver, British Columbia, Canada.

Original Signed and Sealed

Andrew Carter, P.Eng.

28.5 Franky Li, P.Eng.

I, Franky Li, of Victoria, British Columbia, Canada, as an author of this report titled *National Instrument 43-101 Technical Report, Preliminary Economic Assessment and Updated Mineral Resource Estimate for the Hat Project, Northwest British Columbia, Canada*, with an MRE effective date of February 4, 2026, PEA effective date of March 22, 2026, and dated April 10, 2026, (this "Technical Report"), prepared for Doubleview (the "Issuer") do hereby certify that:

- I am an Associate Director with EMM Consulting Pty Limited, with a business address at Suite 300-2345 Yonge Street, Toronto, Ontario, M4P 2E5, Canada.
- I graduated with Bachelor of Civil Engineering from the University of British Columbia in 1996 and a Master of Applied Science in Geoenvironmental Engineering in 1999 from the University of British Columbia. I have 25 years of continuous experience in geotechnical and water resources engineering, in the areas of soil mechanics, dam design, tailings management and water management, and worked on multiple mining projects in Canada, Mongolia, Indonesia, and Papua New Guinea.
- I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- I am responsible for Sections 18.2.8 to 18.3.6, and 20, and subsections in 1, 21, 25, 26, 27 of this Technical Report.
- I have had no prior involvement with the property that is subject to this Technical Report.
- I am independent of the Issuer within the meaning of Section 1.5 of NI 43-101.
- I have not visited the Hat Project site.
- I have read NI 43-101 and Form 43-101F1 and the parts of the Technical Report for which I am responsible, and these sections have been prepared in compliance with that instrument.
- I have read the News Releases dated March 2, 2026, and confirm these news releases are a fair and accurate summary of my sections of this Technical Report.
- I am not aware of any material fact or material change with respect to the subject matter of the report, which is not reflected in the report and by which, the omission to disclose would make the Technical Report misleading.
- I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication of the Technical Report in the public Company files on their websites.

Signed and dated this 10th of April 2026 in Vancouver, British Columbia, Canada.

Original Signed and Sealed

Franky Li, P.Eng.

28.6 Jayeshkumar Rami, P.Eng.

I, Jayeshkumar Rami, P.Eng., do hereby certify that I am employed by Sacre-Davey Engineering, and retained by Mineit Consulting Inc., with an office at 1188–1095 West Pender Street, Vancouver, British Columbia, V6E 2M6, Canada.

- This certificate applies to the technical report titled *National Instrument 43-101 Technical Report, Preliminary Economic Assessment and Updated Mineral Resource Estimate for the Hat Project, Northwest British Columbia, Canada*, with an MRE effective date of February 4, 2026, PEA effective date of March 22, 2026, and dated April 10, 2026 (the "Technical Report"), prepared for Doubleview (the "Issuer").
- I am a registered Professional Engineer. I hold Mechanical Engineering degree by profession (BVM Engineering College, Sardar Patel University, Bachelor of Engineering, 2001), registered with Canadian Professional Engineering Associations in British Columbia and Nunavut/Northwest Territories. I have practiced my profession for more than 20 years. I have been directly involved in mining, and mineral processing, projects in Canada and India.
- I have read the definition of "qualified person" set out in National Instrument 43-101—Standards of Disclosure for Mineral Projects (NI 43-101) and certify that, by reason of my education, professional affiliation, and relevant experience, I satisfy the requirements to act as a Qualified Person for the purposes of NI 43-101.
- I have not conducted a personal inspection of the Hat Project property.
- I am responsible for Sections 18.1.1, 18.1.2, 18.2.1 to 18.2.7 and subsections in 1, 25, 26, 27 of this Technical Report.
- I am independent of the Issuer within the meaning of Section 1.5 of NI 43-101.
- I have had no prior involvement with the property that is the subject of the Technical Report, other than work completed in connection with the Technical Report.
- I have read NI 43-101 and Form 43-101F1, and the portions of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the effective date of the Technical Report, and as of the date of this certificate, to the best of my knowledge, information, and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

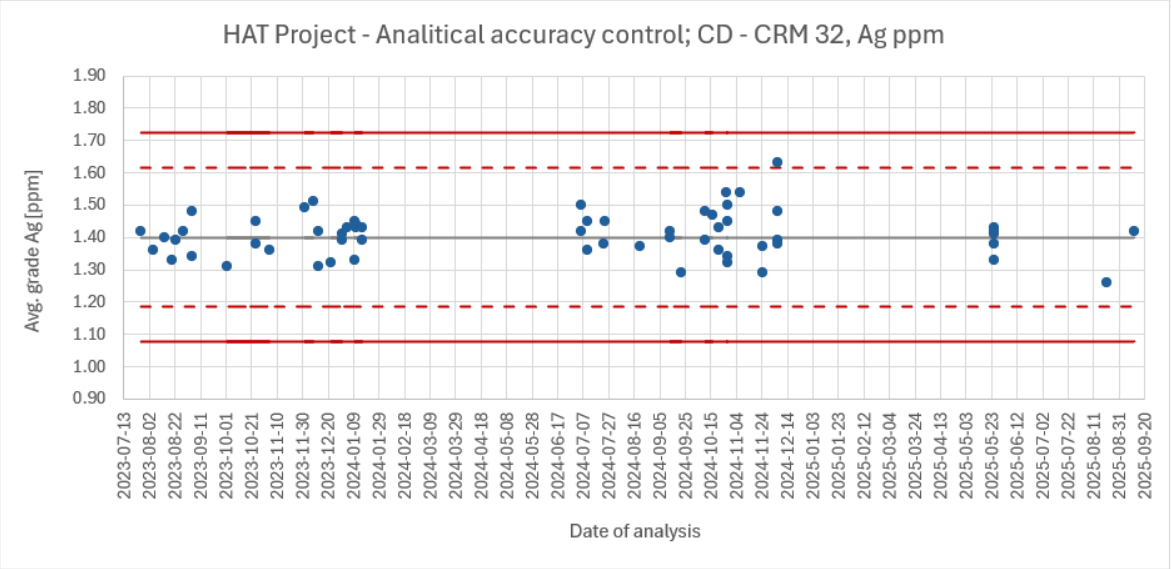
Signed and dated this 10th day of April 2026 at Vancouver, British Columbia, Canada.

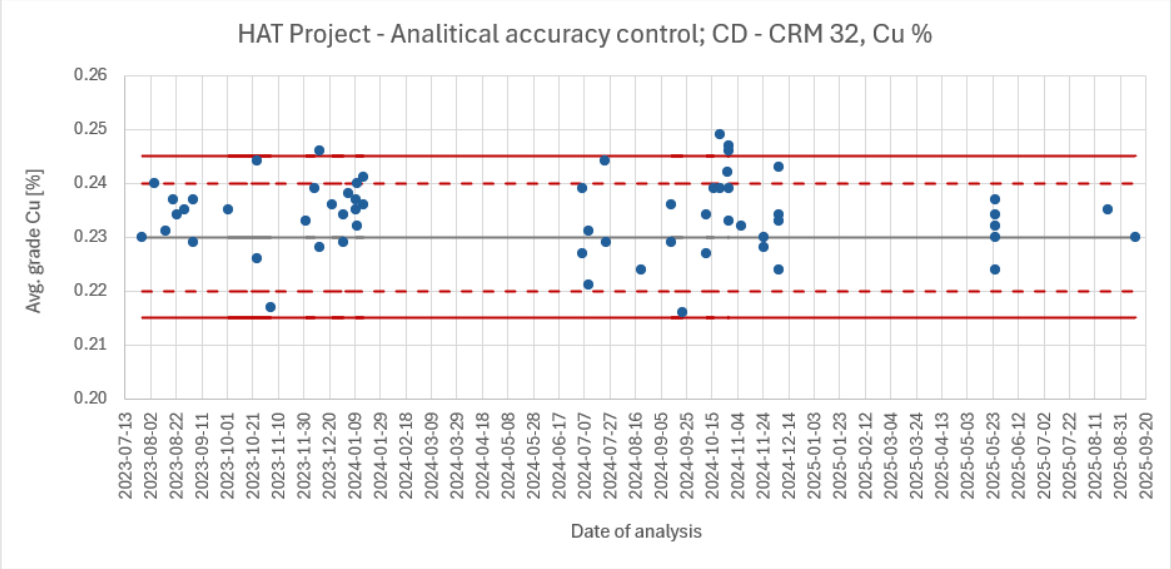
Original Signed and Sealed

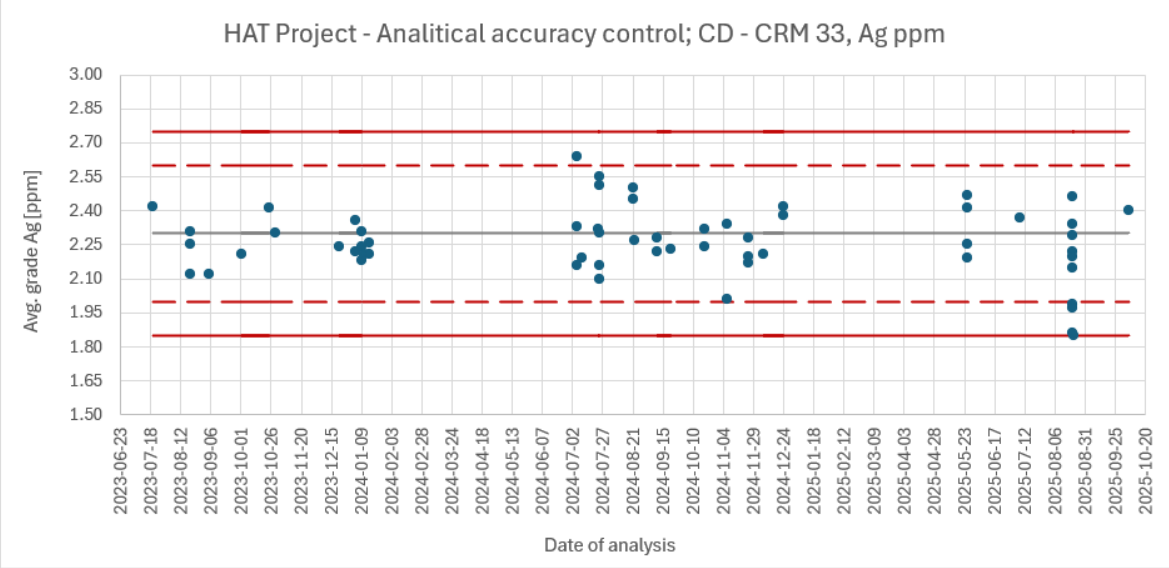
Jayeshkumar Rami, P.Eng.

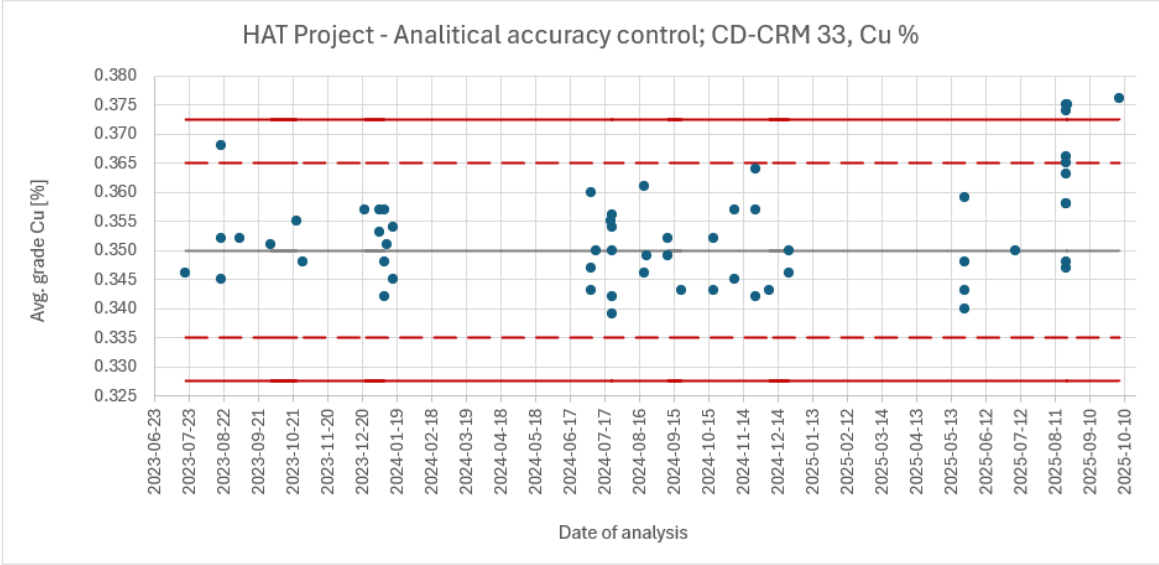
Appendix 11-1

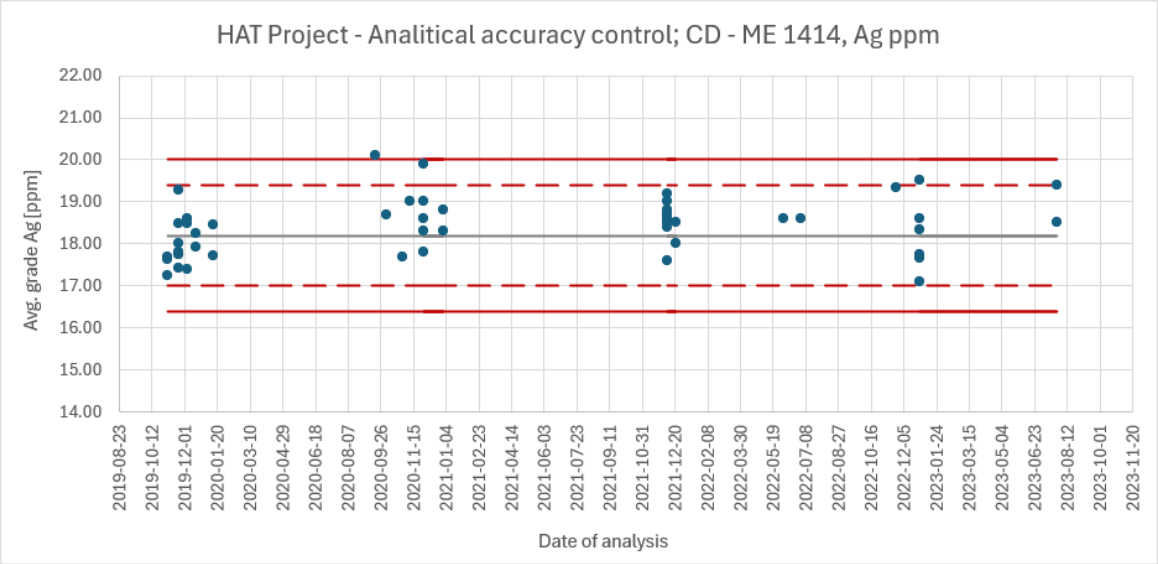
QC/QC CRM performance

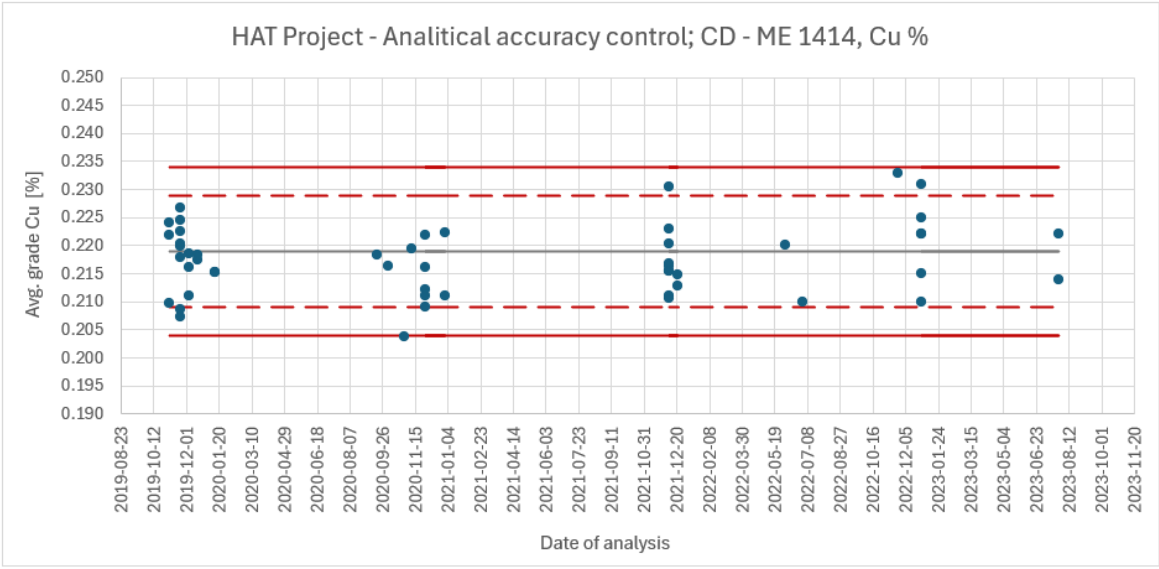


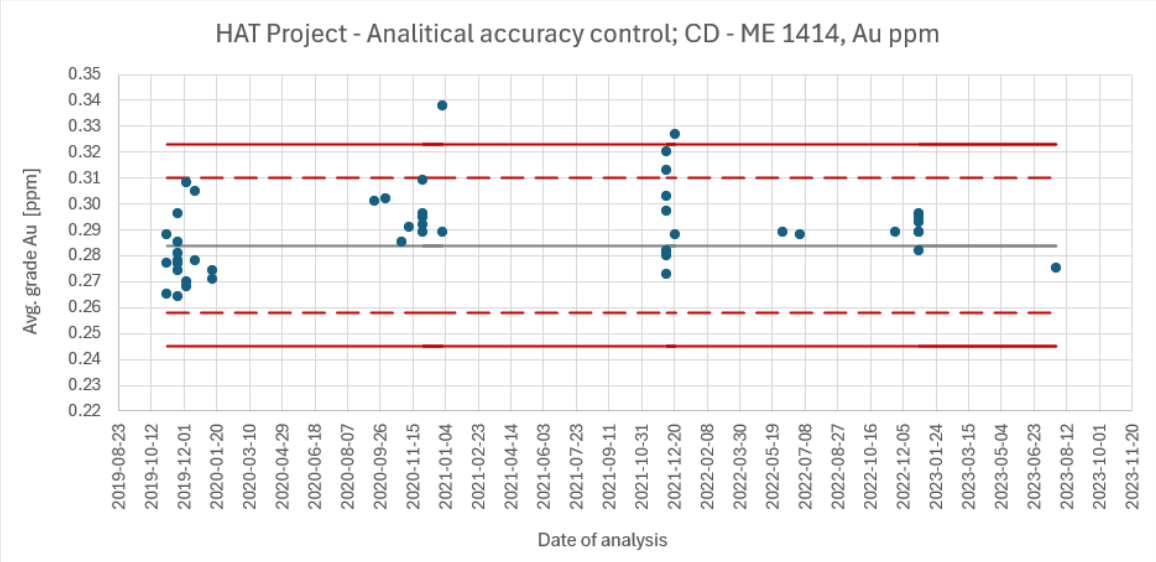


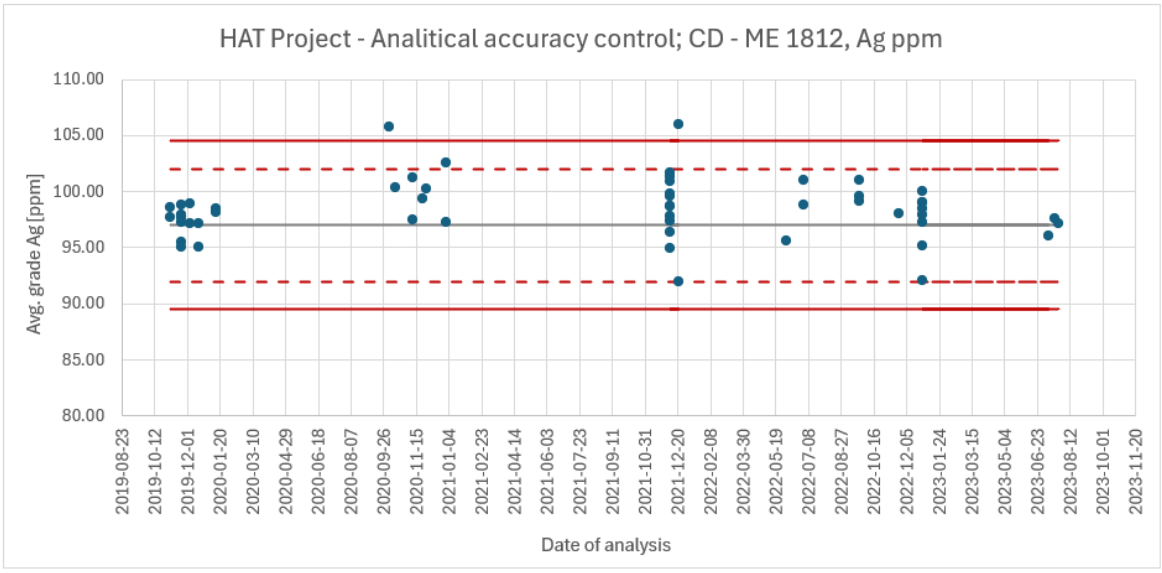

HAT Project - Analitical accuracy control; CD - CRM 33, Ag ppm
Avg. grade Ag [ppm]
3.00
2.85
2.70
2.55
2.40
2.25
2.10
1.95
1.80
1.65
1.50
2023-06-23
2023-07-18
2023-08-12
2023-09-06
2023-10-01
2023-10-26
2023-11-20
2023-12-15
2024-01-09
2024-02-03
2024-02-28
2024-03-24
2024-04-18
2024-05-13
2024-06-07
2024-07-02
2024-07-27
2024-08-21
2024-09-15
2024-10-10
2024-11-04
2024-11-29
2024-12-24
2025-01-18
2025-02-12
2025-03-09
2025-04-03
2025-04-28
2025-05-23
2025-06-17
2025-07-12
2025-08-06
2025-08-31
2025-09-25
2025-10-20
Date of analysis

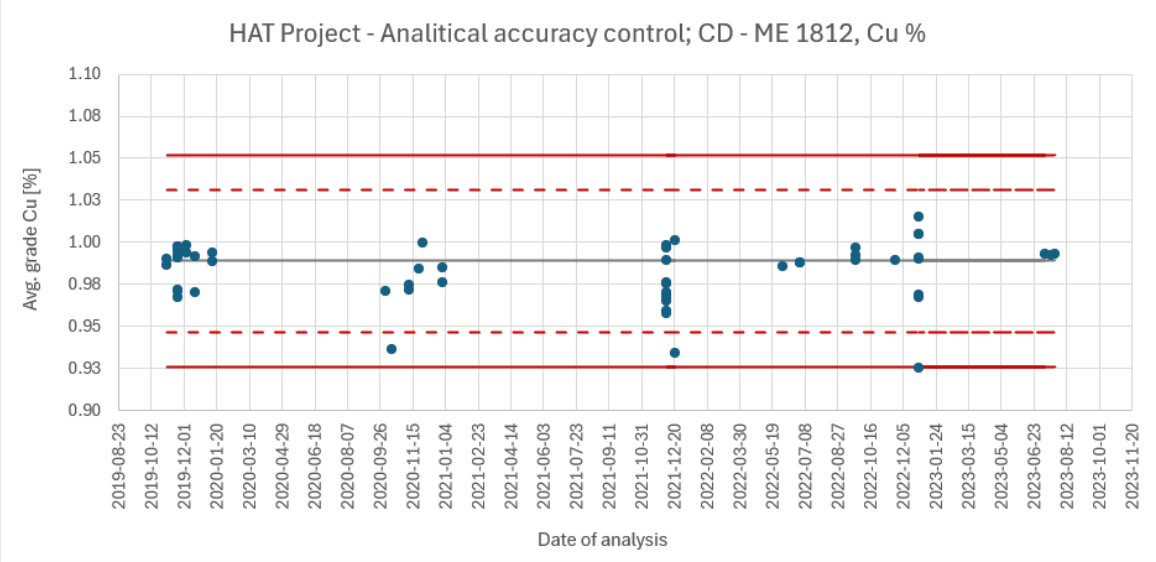

HAT Project - Analitical accuracy control; CD-CRM 33, Cu %
Avg. grade Cu [%]
0.380
0.375
0.370
0.365
0.360
0.355
0.350
0.345
0.340
0.335
0.330
0.325
2023-06-23
2023-07-23
2023-08-22
2023-09-21
2023-10-21
2023-11-20
2023-12-20
2024-01-19
2024-02-18
2024-03-19
2024-04-18
2024-05-18
2024-06-17
2024-07-17
2024-08-16
2024-09-15
2024-10-15
2024-11-14
2024-12-14
2025-01-13
2025-02-12
2025-03-14
2025-04-13
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2025-06-12
2025-07-12
2025-08-11
2025-09-10
2025-10-10
Date of analysis

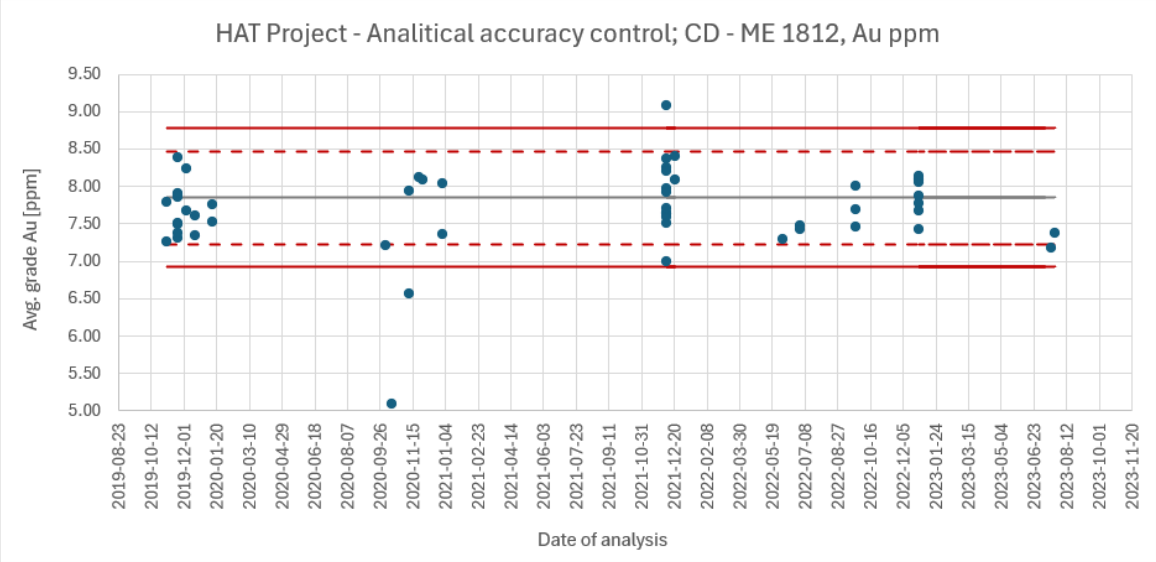

HAT Project - Analitical accuracy control; CD - ME 1414, Ag ppm
Avg. grade Ag [ppm]
22.00
21.00
20.00
19.00
18.00
17.00
16.00
15.00
14.00
2019-08-23
2019-10-12
2019-12-01
2020-01-20
2020-03-10
2020-04-29
2020-06-18
2020-08-07
2020-09-26
2020-11-15
2021-01-04
2021-02-23
2021-04-14
2021-06-03
2021-07-23
2021-09-11
2021-10-31
2021-12-20
2022-02-08
2022-03-30
2022-05-19
2022-07-08
2022-08-27
2022-10-16
2022-12-05
2023-01-24
2023-03-15
2023-05-04
2023-06-23
2023-08-12
2023-10-01
2023-11-20
Date of analysis

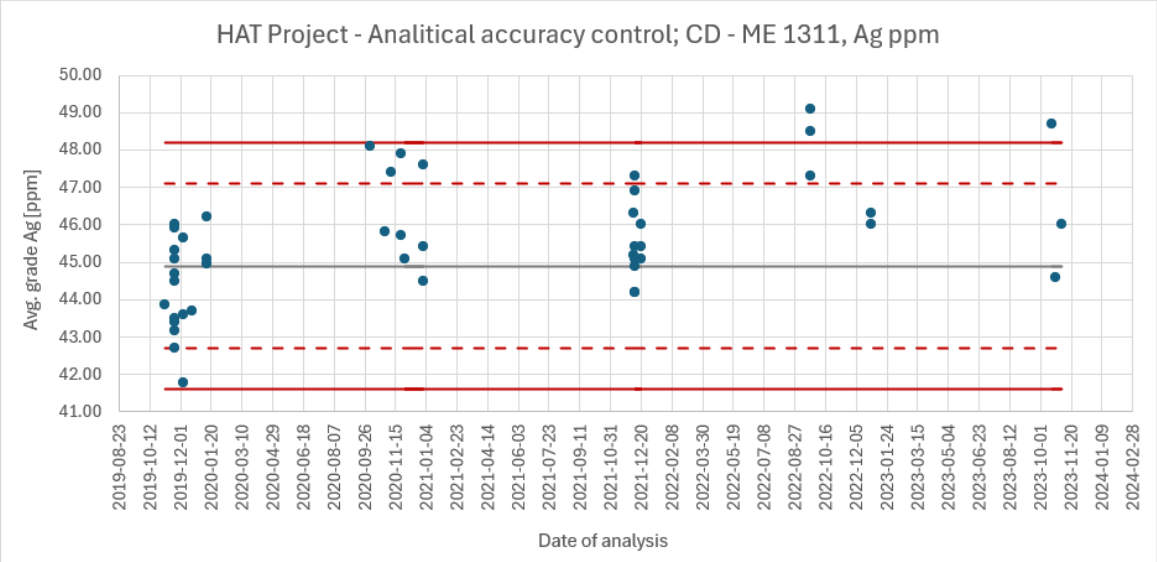

HAT Project - Analitical accuracy control; CD - ME 1414, Cu %
Avg. grade Cu [%]
0.250
0.245
0.240
0.235
0.230
0.225
0.220
0.215
0.210
0.205
0.200
0.195
0.190
2019-08-23
2019-10-12
2019-12-01
2020-01-20
2020-03-10
2020-04-29
2020-06-18
2020-08-07
2020-09-26
2020-11-15
2021-01-04
2021-02-23
2021-04-14
2021-06-03
2021-07-23
2021-09-11
2021-10-31
2021-12-20
2022-02-08
2022-03-30
2022-05-19
2022-07-08
2022-08-27
2022-10-16
2022-12-05
2023-01-24
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2023-08-12
2023-10-01
2023-11-20
Date of analysis

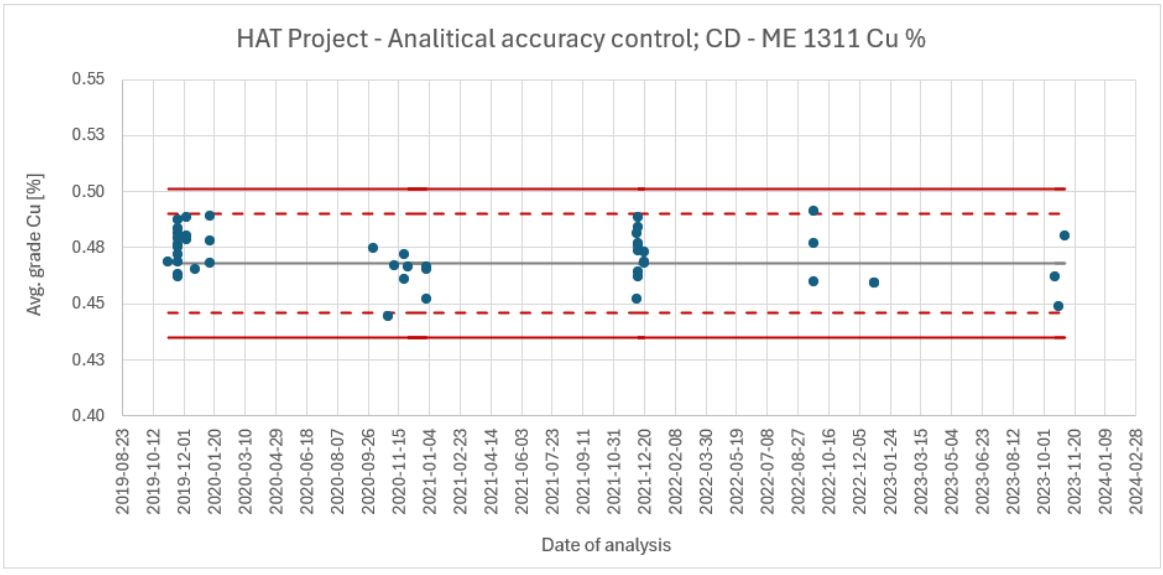

HAT Project - Analitical accuracy control; CD - ME 1414, Au ppm
Avg. grade Au [ppm]
0.35
0.34
0.33
0.32
0.31
0.30
0.29
0.28
0.27
0.26
0.25
0.24
0.23
0.22
2019-08-23
2019-10-12
2019-12-01
2020-01-20
2020-03-10
2020-04-29
2020-06-18
2020-08-07
2020-09-26
2020-11-15
2021-01-04
2021-02-23
2021-04-14
2021-06-03
2021-07-23
2021-09-11
2021-10-31
2021-12-20
2022-02-08
2022-03-30
2022-05-19
2022-07-08
2022-08-27
2022-10-16
2022-12-05
2023-01-24
2023-03-15
2023-05-04
2023-06-23
2023-08-12
2023-10-01
2023-11-20
Date of analysis

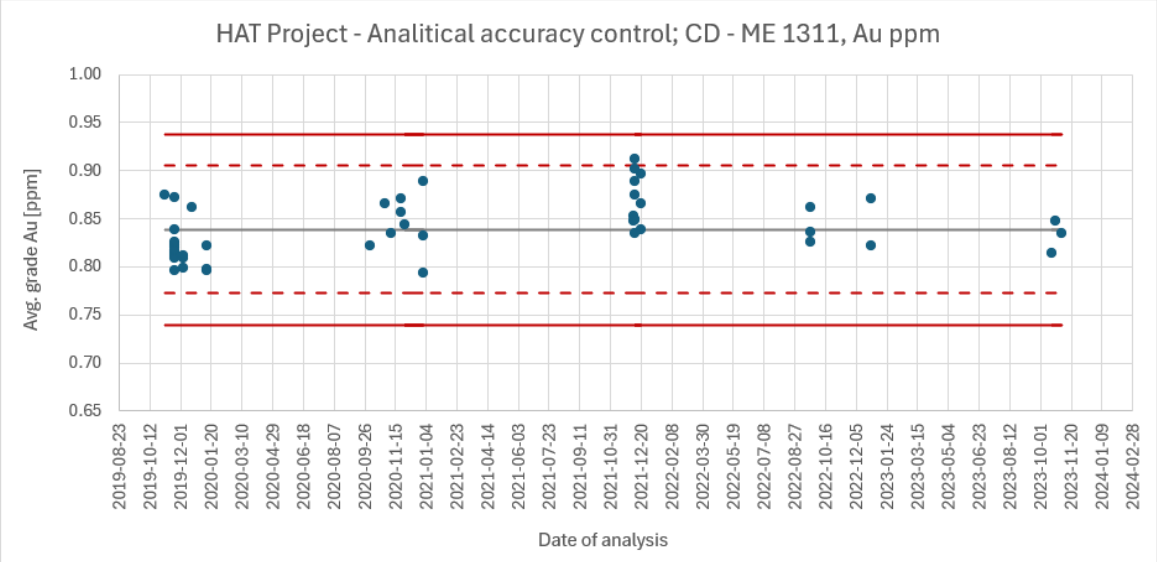

HAT Project - Analitical accuracy control; CD - ME 1812, Ag ppm
Avg. grade Ag [ppm]
110.00
105.00
100.00
95.00
90.00
85.00
80.00
2019-08-23
2019-10-12
2019-12-01
2020-01-20
2020-03-10
2020-04-29
2020-06-18
2020-08-07
2020-09-26
2020-11-15
2021-01-04
2021-02-23
2021-04-14
2021-06-03
2021-07-23
2021-09-11
2021-10-31
2021-12-20
2022-02-08
2022-03-30
2022-05-19
2022-07-08
2022-08-27
2022-10-16
2022-12-05
2023-01-24
2023-03-15
2023-05-04
2023-06-23
2023-08-12
2023-10-01
2023-11-20
Date of analysis

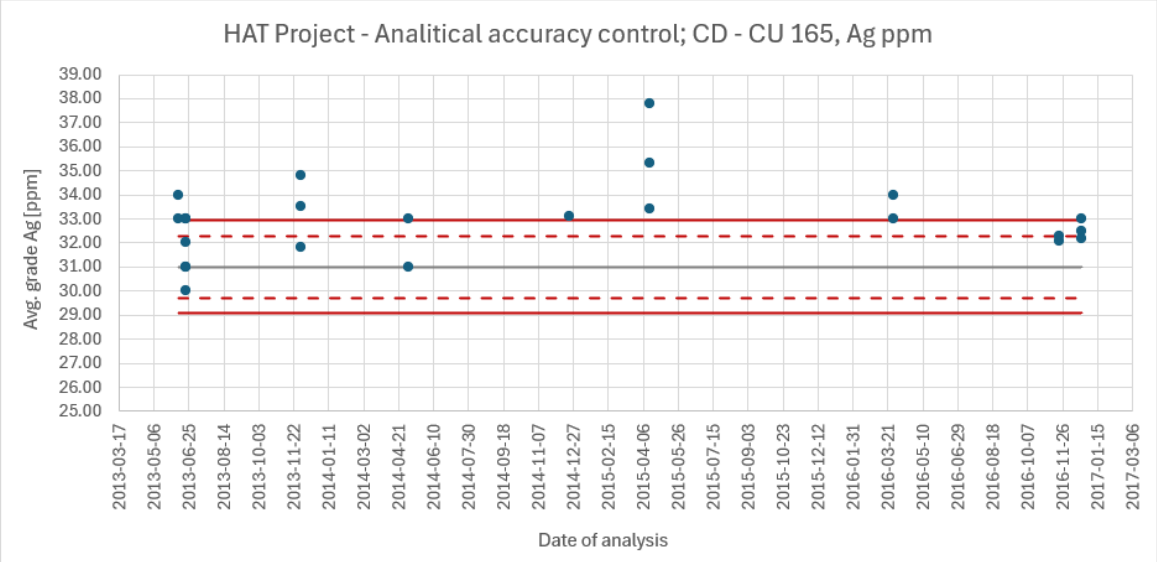

HAT Project - Analitical accuracy control; CD - ME 1812, Cu %
Avg. grade Cu [%]
1.10
1.08
1.05
1.03
1.00
0.98
0.95
0.93
0.90
2019-08-23
2019-10-12
2019-12-01
2020-01-20
2020-03-10
2020-04-29
2020-06-18
2020-08-07
2020-09-26
2020-11-15
2021-01-04
2021-02-23
2021-04-14
2021-06-03
2021-07-23
2021-09-11
2021-10-31
2021-12-20
2022-02-08
2022-03-30
2022-05-19
2022-07-08
2022-08-27
2022-10-16
2022-12-05
2023-01-24
2023-03-15
2023-05-04
2023-06-23
2023-08-12
2023-10-01
2023-11-20
Date of analysis

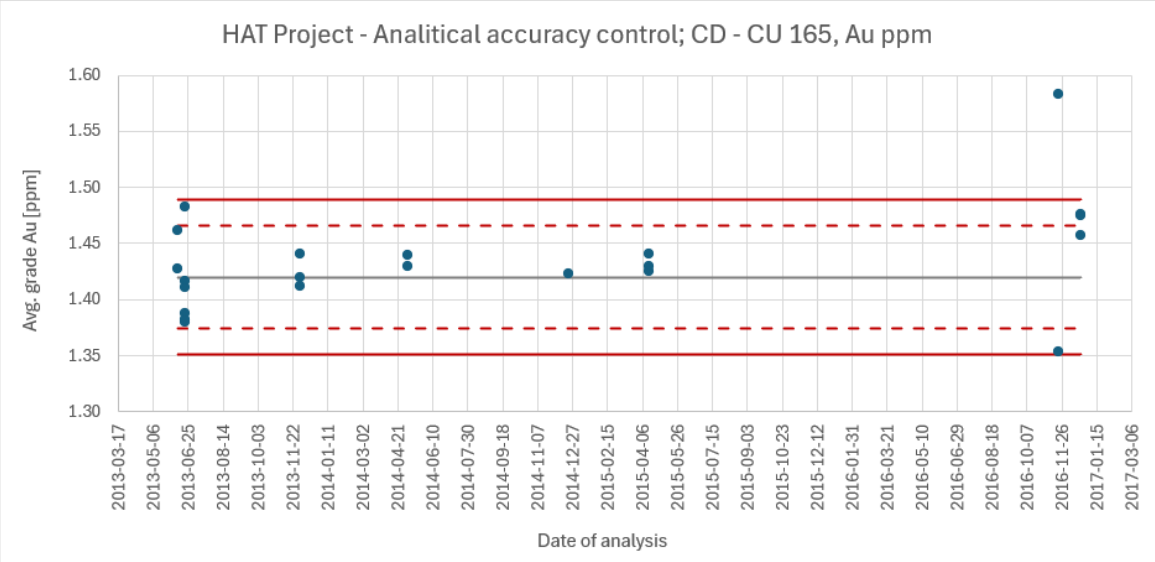

HAT Project - Analitical accuracy control; CD - ME 1812, Au ppm
Avg. grade Au [ppm]
9.50
9.00
8.50
8.00
7.50
7.00
6.50
6.00
5.50
5.00
2019-08-23
2019-10-12
2019-12-01
2020-01-20
2020-03-10
2020-04-29
2020-06-18
2020-08-07
2020-09-26
2020-11-15
2021-01-04
2021-02-23
2021-04-14
2021-06-03
2021-07-23
2021-09-11
2021-10-31
2021-12-20
2022-02-08
2022-03-30
2022-05-19
2022-07-08
2022-08-27
2022-10-16
2022-12-05
2023-01-24
2023-03-15
2023-05-04
2023-06-23
2023-08-12
2023-10-01
2023-11-20
Date of analysis

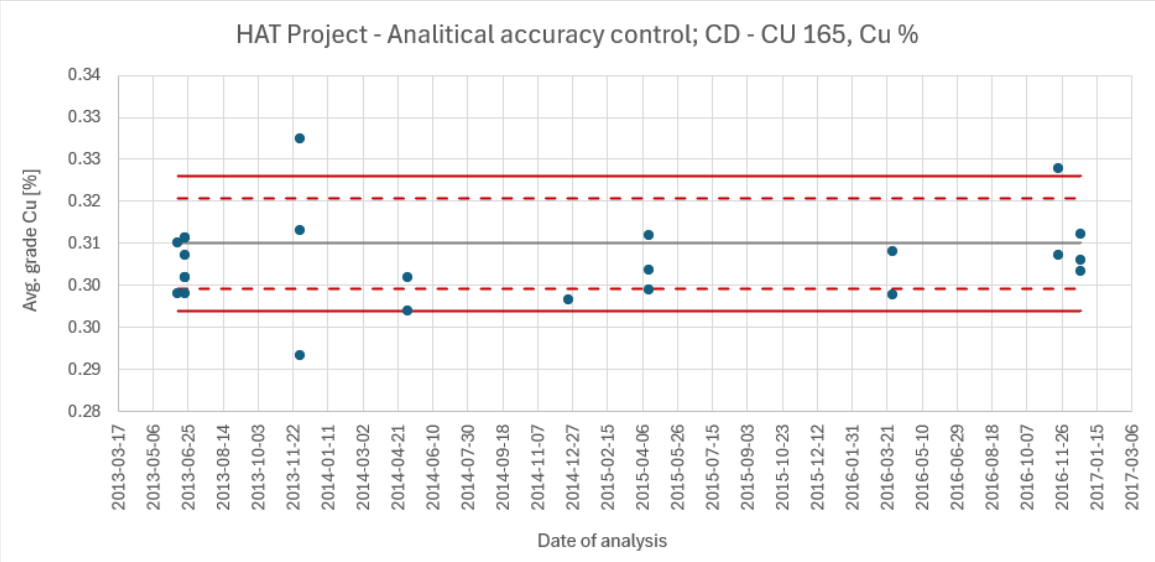

HAT Project - Analitical accuracy control; CD - ME 1311, Ag ppm
Avg. grade Ag [ppm]
50.00
49.00
48.00
47.00
46.00
45.00
44.00
43.00
42.00
41.00
2019-08-23
2019-10-12
2019-12-01
2020-01-20
2020-03-10
2020-04-29
2020-06-18
2020-08-07
2020-09-26
2020-11-15
2021-01-04
2021-02-23
2021-04-14
2021-06-03
2021-07-23
2021-09-11
2021-10-31
2021-12-20
2022-02-08
2022-03-30
2022-05-19
2022-07-08
2022-08-27
2022-10-16
2022-12-05
2023-01-24
2023-03-15
2023-05-04
2023-06-23
2023-08-12
2023-10-01
2023-11-20
2024-01-09
2024-02-28
Date of analysis

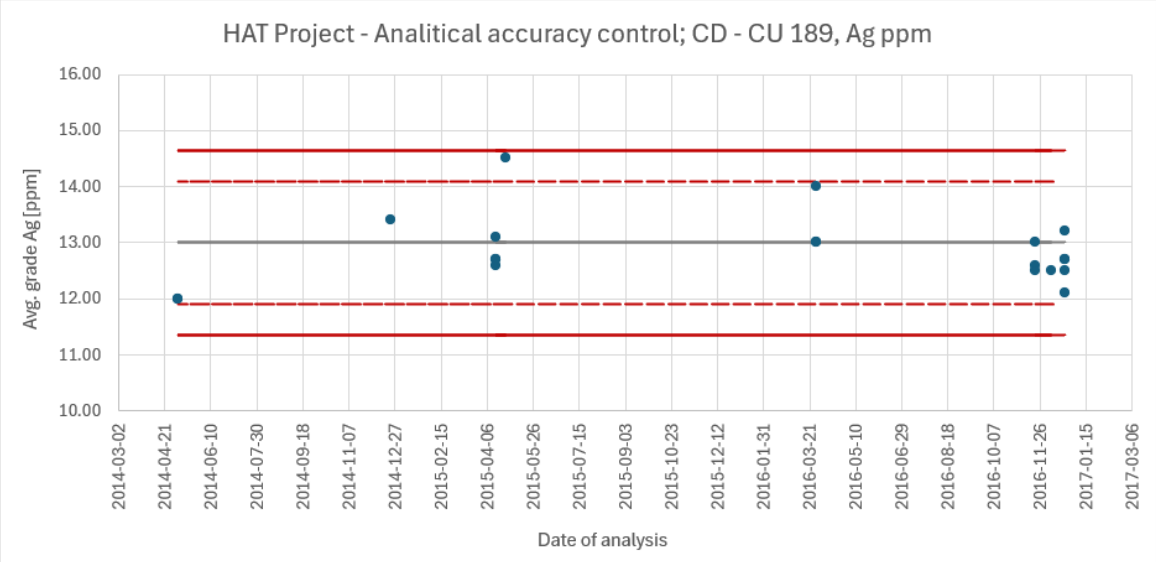

HAT Project - Analitical accuracy control; CD - ME 1311 Cu %
Avg. grade Cu [%]
0.55
0.53
0.50
0.48
0.45
0.43
0.40
2019-08-23
2019-10-12
2019-12-01
2020-01-20
2020-03-10
2020-04-29
2020-06-18
2020-08-07
2020-09-26
2020-11-15
2021-01-04
2021-02-23
2021-04-14
2021-06-03
2021-07-23
2021-09-11
2021-10-31
2021-12-20
2022-02-08
2022-03-30
2022-05-19
2022-07-08
2022-08-27
2022-10-16
2022-12-05
2023-01-24
2023-03-15
2023-05-04
2023-06-23
2023-08-12
2023-10-01
2023-11-20
2024-01-09
2024-02-28
Date of analysis

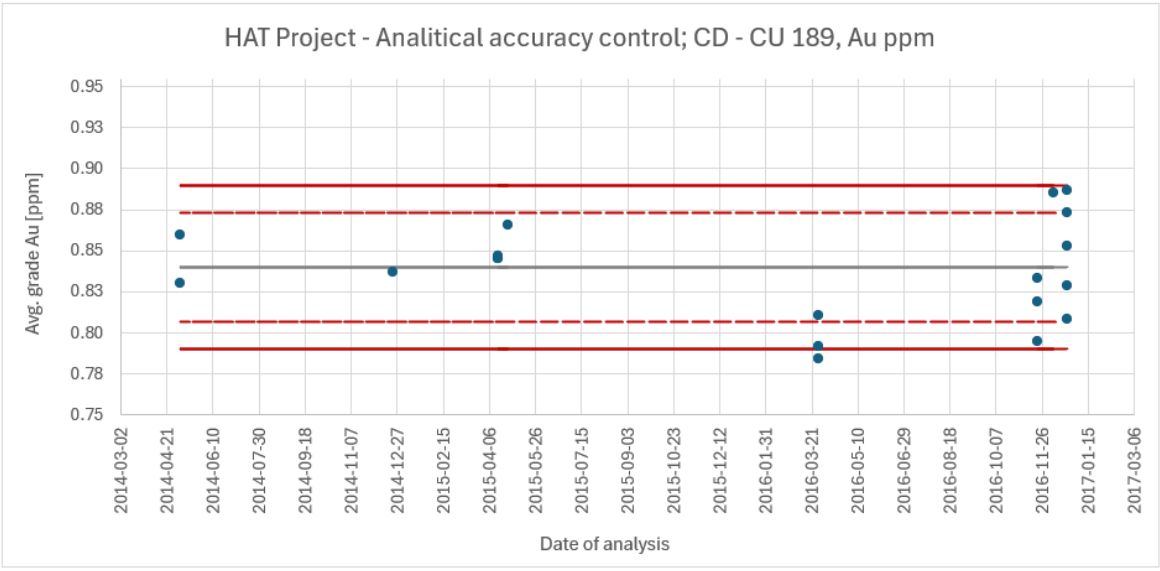

HAT Project - Analitical accuracy control; CD - ME 1311, Au ppm
Avg. grade Au [ppm]
1.00
0.95
0.90
0.85
0.80
0.75
0.70
0.65
2019-08-23
2019-10-12
2019-12-01
2020-01-20
2020-03-10
2020-04-29
2020-06-18
2020-08-07
2020-09-26
2020-11-15
2021-01-04
2021-02-23
2021-04-14
2021-06-03
2021-07-23
2021-09-11
2021-10-31
2021-12-20
2022-02-08
2022-03-30
2022-05-19
2022-07-08
2022-08-27
2022-10-16
2022-12-05
2023-01-24
2023-03-15
2023-05-04
2023-06-23
2023-08-12
2023-10-01
2023-11-20
2024-01-09
2024-02-28
Date of analysis

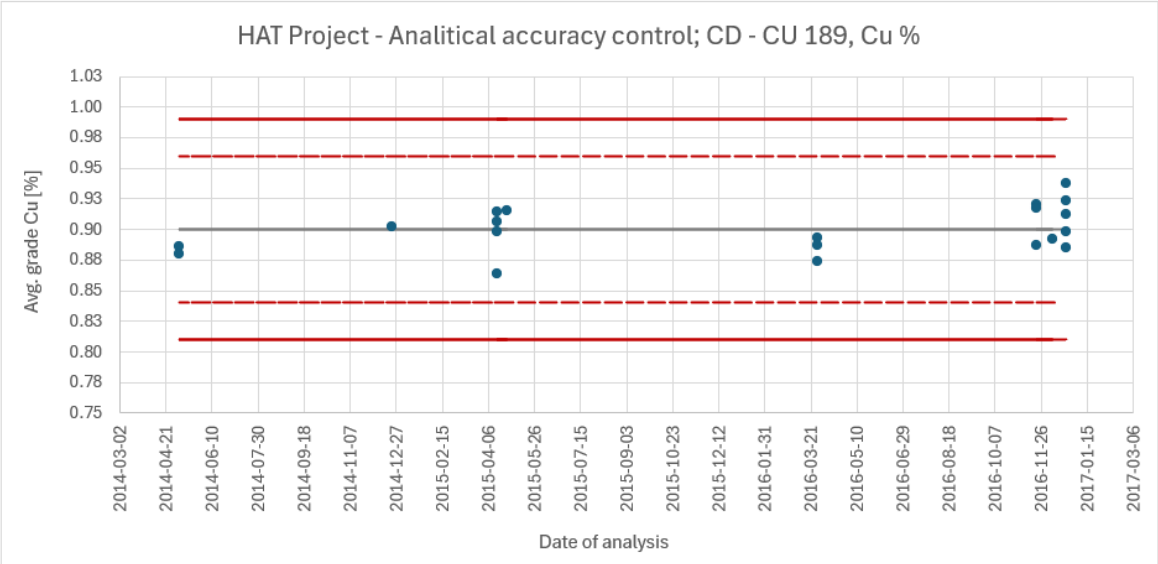

HAT Project - Analitical accuracy control; CD - CU 165, Ag ppm
Avg. grade Ag [ppm]
39.00
38.00
37.00
36.00
35.00
34.00
33.00
32.00
31.00
30.00
29.00
28.00
27.00
26.00
25.00
2013-03-17
2013-05-06
2013-06-25
2013-08-14
2013-10-03
2013-11-22
2014-01-11
2014-03-02
2014-04-21
2014-06-10
2014-07-30
2014-09-18
2014-11-07
2014-12-27
2015-02-15
2015-04-06
2015-05-26
2015-07-15
2015-09-03
2015-10-23
2015-12-12
2016-01-31
2016-03-21
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2016-06-29
2016-08-18
2016-10-07
2016-11-26
2017-01-15
2017-03-06
Date of analysis

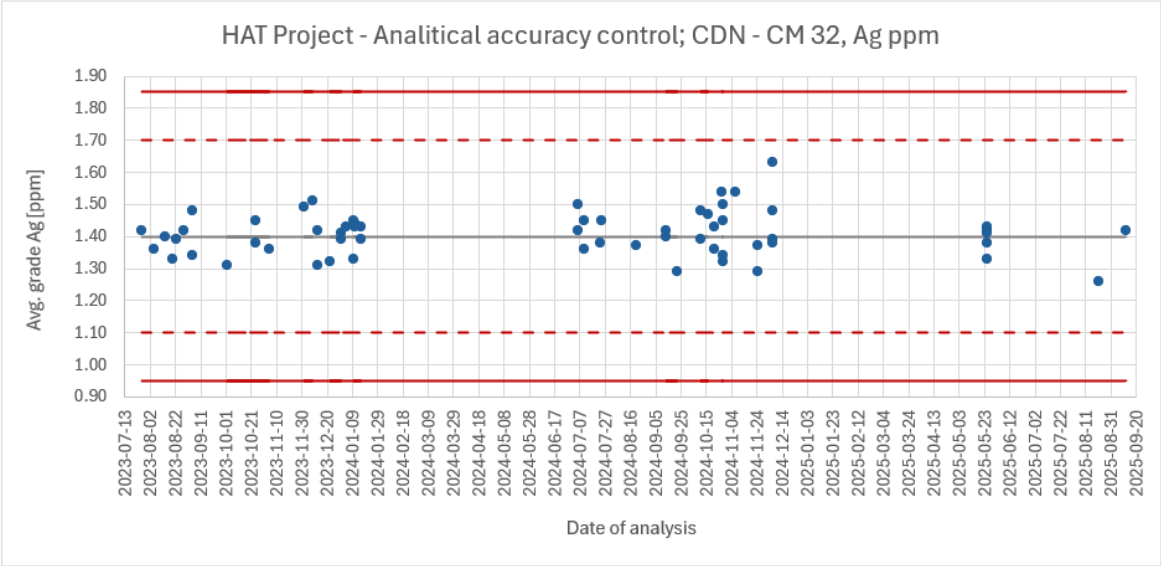

HAT Project - Analitical accuracy control; CD - CU 165, Au ppm
Avg. grade Au [ppm]
1.60
1.55
1.50
1.45
1.40
1.35
1.30
2013-03-17
2013-05-06
2013-06-25
2013-08-14
2013-10-03
2013-11-22
2014-01-11
2014-03-02
2014-04-21
2014-06-10
2014-07-30
2014-09-18
2014-11-07
2014-12-27
2015-02-15
2015-04-06
2015-05-26
2015-07-15
2015-09-03
2015-10-23
2015-12-12
2016-01-31
2016-03-21
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2016-06-29
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2016-11-26
2017-01-15
2017-03-06
Date of analysis

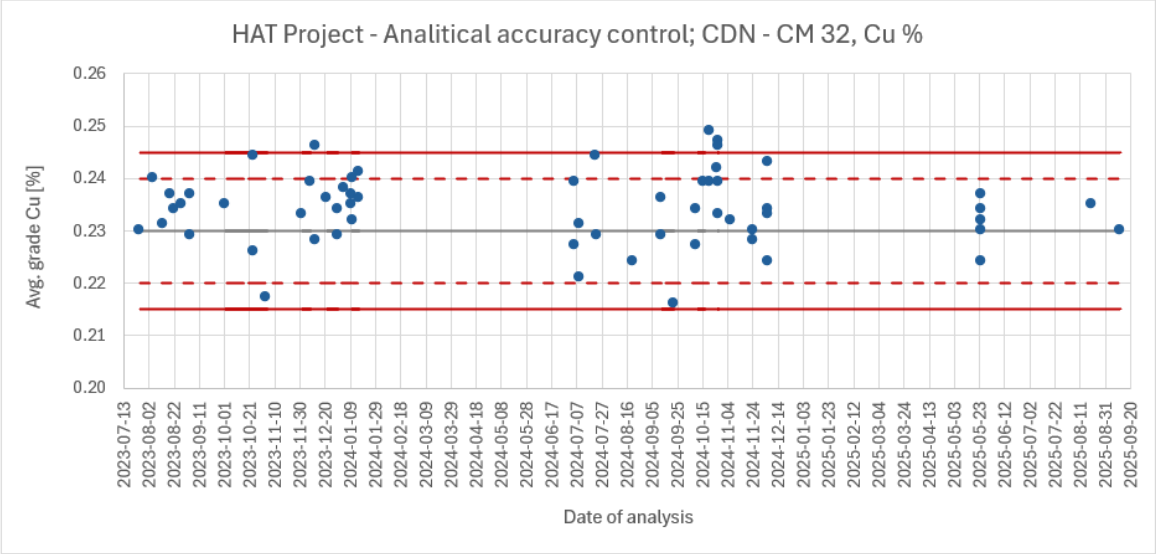

HAT Project - Analitical accuracy control; CD - CU 165, Cu %
Avg. grade Cu [%]
0.34
0.33
0.33
0.32
0.31
0.30
0.30
0.29
0.28
2013-03-17
2013-05-06
2013-06-25
2013-08-14
2013-10-03
2013-11-22
2014-01-11
2014-03-02
2014-04-21
2014-06-10
2014-07-30
2014-09-18
2014-11-07
2014-12-27
2015-02-15
2015-04-06
2015-05-26
2015-07-15
2015-09-03
2015-10-23
2015-12-12
2016-01-31
2016-03-21
2016-05-10
2016-06-29
2016-08-18
2016-10-07
2016-11-26
2017-01-15
2017-03-06
Date of analysis

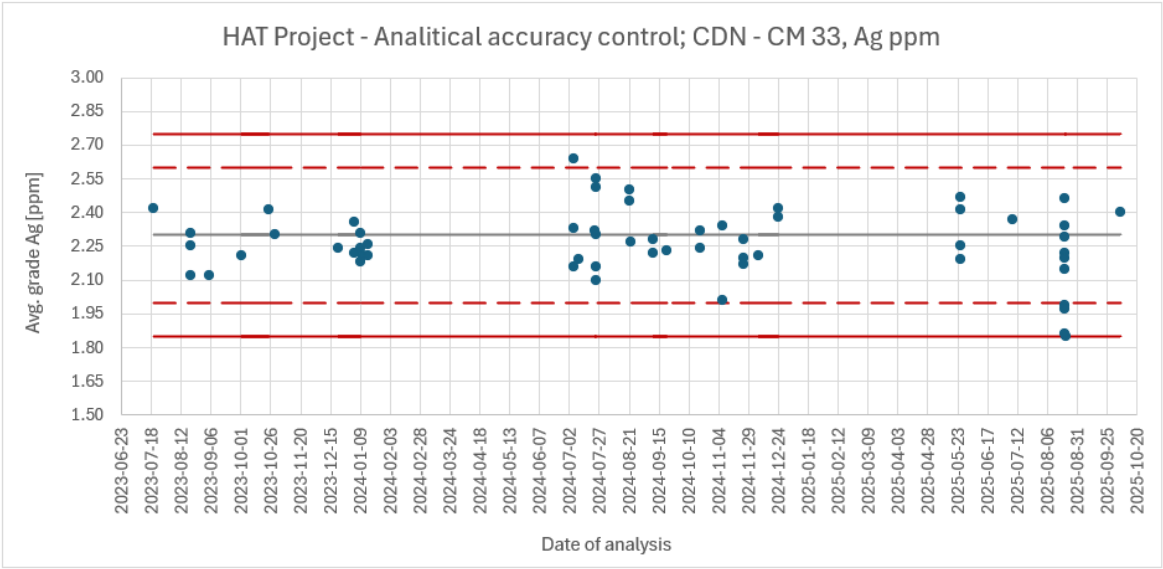

HAT Project - Analitical accuracy control; CD - CU 189, Ag ppm
Avg. grade Ag [ppm]
16.00
15.00
14.00
13.00
12.00
11.00
10.00
2014-03-02
2014-04-21
2014-06-10
2014-07-30
2014-09-18
2014-11-07
2014-12-27
2015-02-15
2015-04-06
2015-05-26
2015-07-15
2015-09-03
2015-10-23
2015-12-12
2016-01-31
2016-03-21
2016-05-10
2016-06-29
2016-08-18
2016-10-07
2016-11-26
2017-01-15
2017-03-06
Date of analysis

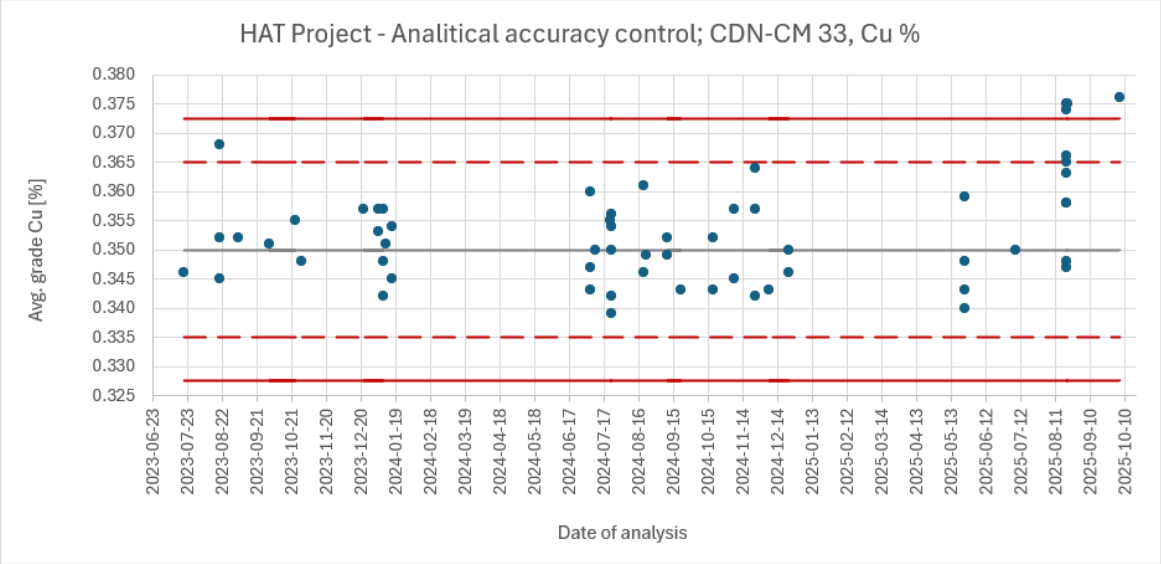

HAT Project - Analitical accuracy control; CD - CU 189, Au ppm
Avg. grade Au [ppm]
0.95
0.93
0.90
0.88
0.85
0.83
0.80
0.78
0.75
2014-03-02
2014-04-21
2014-06-10
2014-07-30
2014-09-18
2014-11-07
2014-12-27
2015-02-15
2015-04-06
2015-05-26
2015-07-15
2015-09-03
2015-10-23
2015-12-12
2016-01-31
2016-03-21
2016-05-10
2016-06-29
2016-08-18
2016-10-07
2016-11-26
2017-01-15
2017-03-06
Date of analysis

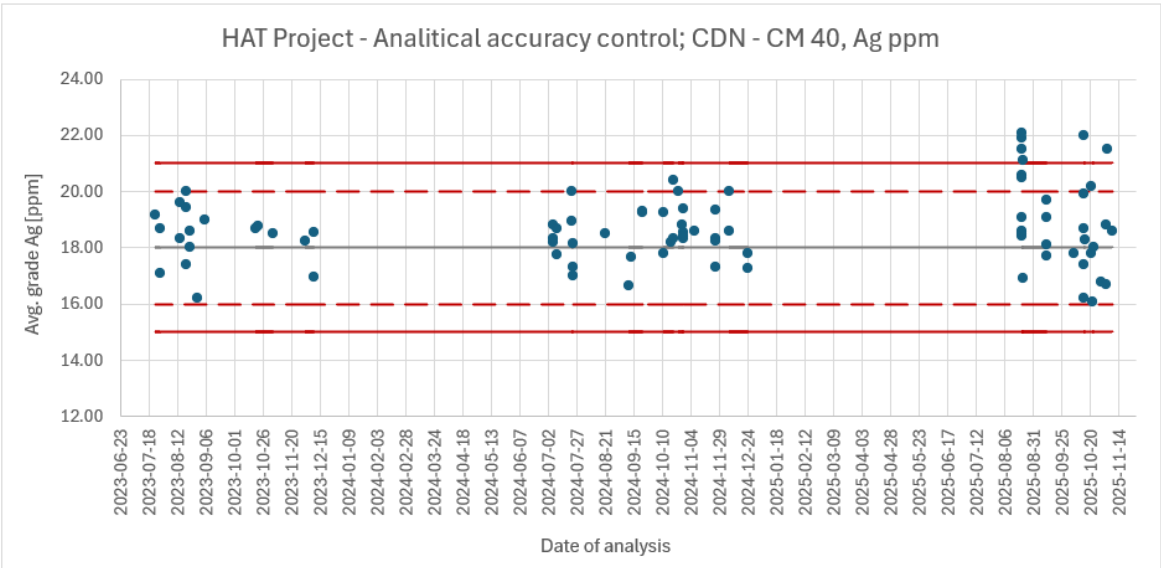

HAT Project - Analitical accuracy control; CD - CU 189, Cu %
Avg. grade Cu [%]
1.03
1.00
0.98
0.95
0.93
0.90
0.88
0.85
0.83
0.80
0.78
0.75
2014-03-02
2014-04-21
2014-06-10
2014-07-30
2014-09-18
2014-11-07
2014-12-27
2015-02-15
2015-04-06
2015-05-26
2015-07-15
2015-09-03
2015-10-23
2015-12-12
2016-01-31
2016-03-21
2016-05-10
2016-06-29
2016-08-18
2016-10-07
2016-11-26
2017-01-15
2017-03-06
Date of analysis

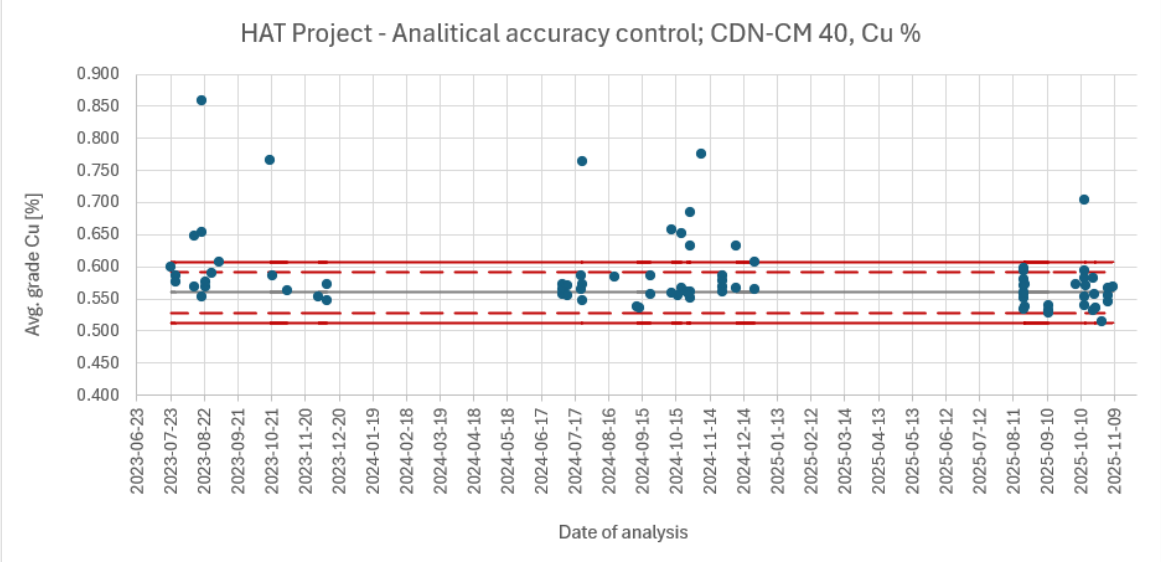

HAT Project - Analitical accuracy control; CDN - CM 32, Ag ppm
Avg. grade Ag [ppm]
1.90
1.80
1.70
1.60
1.50
1.40
1.30
1.20
1.10
1.00
0.90
2023-07-13
2023-08-02
2023-08-22
2023-09-11
2023-10-01
2023-10-21
2023-11-10
2023-11-30
2023-12-20
2024-01-09
2024-01-29
2024-02-18
2024-03-09
2024-03-29
2024-04-18
2024-05-08
2024-05-28
2024-06-17
2024-07-07
2024-07-27
2024-08-16
2024-09-05
2024-09-25
2024-10-15
2024-11-04
2024-11-24
2024-12-14
2025-01-03
2025-01-23
2025-02-12
2025-03-04
2025-03-24
2025-04-13
2025-05-03
2025-05-23
2025-06-12
2025-07-02
2025-07-22
2025-08-11
2025-08-31
2025-09-20
Date of analysis

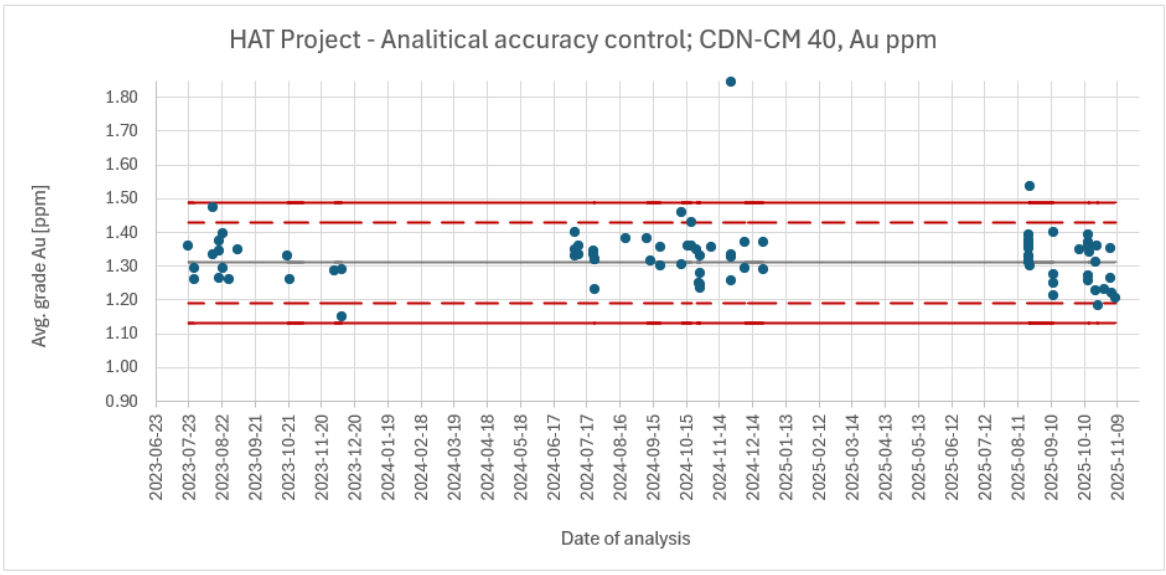

HAT Project - Analitical accuracy control; CDN - CM 32, Cu %
Avg. grade Cu [%]
0.26
0.25
0.24
0.23
0.22
0.21
0.20
2023-07-13
2023-08-02
2023-08-22
2023-09-11
2023-10-01
2023-10-21
2023-11-10
2023-11-30
2023-12-20
2024-01-09
2024-01-29
2024-02-18
2024-03-09
2024-03-29
2024-04-18
2024-05-08
2024-05-28
2024-06-17
2024-07-07
2024-07-27
2024-08-16
2024-09-05
2024-09-25
2024-10-15
2024-11-04
2024-11-24
2024-12-14
2025-01-03
2025-01-23
2025-02-12
2025-03-04
2025-03-24
2025-04-13
2025-05-03
2025-05-23
2025-06-12
2025-07-02
2025-07-22
2025-08-11
2025-08-31
2025-09-20
Date of analysis

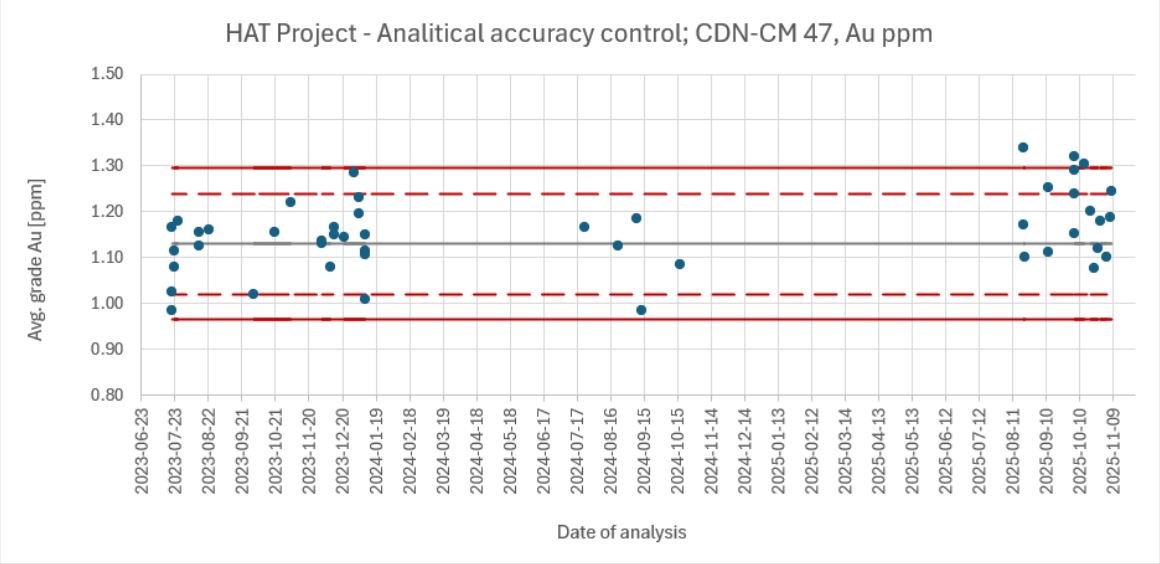

HAT Project - Analitical accuracy control; CDN - CM 33, Ag ppm
Avg. grade Ag [ppm]
3.00
2.85
2.70
2.55
2.40
2.25
2.10
1.95
1.80
1.65
1.50
2023-06-23
2023-07-18
2023-08-12
2023-09-06
2023-10-01
2023-10-26
2023-11-20
2023-12-15
2024-01-09
2024-02-03
2024-02-28
2024-03-24
2024-04-18
2024-05-13
2024-06-07
2024-07-02
2024-07-27
2024-08-21
2024-09-15
2024-10-10
2024-11-04
2024-11-29
2024-12-24
2025-01-18
2025-02-12
2025-03-09
2025-04-03
2025-04-28
2025-05-23
2025-06-17
2025-07-12
2025-08-06
2025-08-31
2025-09-25
2025-10-20
Date of analysis

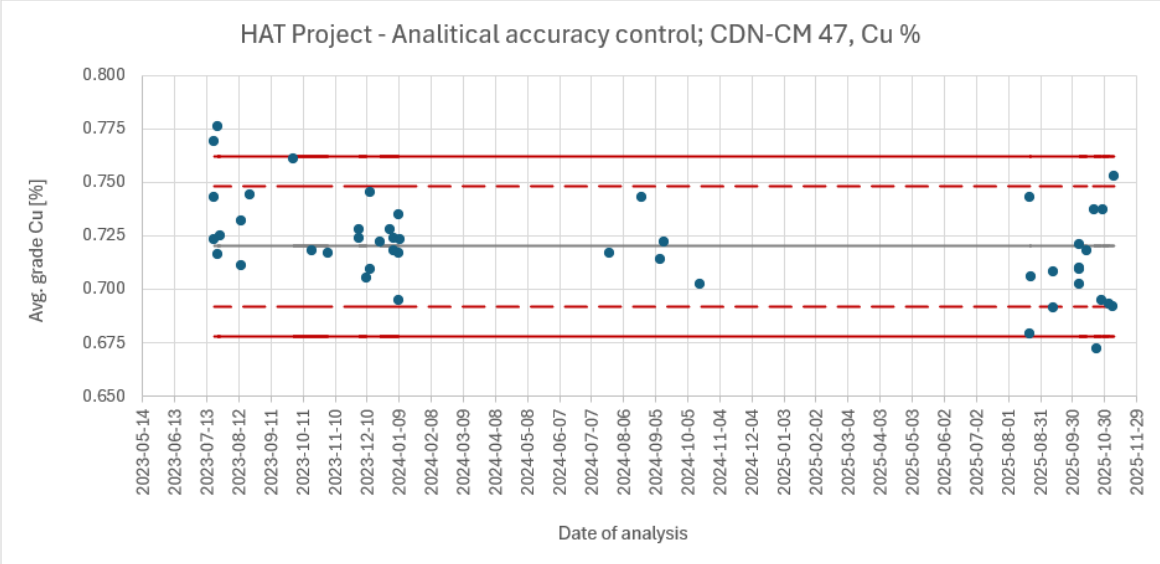

HAT Project - Analitical accuracy control; CDN-CM 33, Cu %
Avg. grade Cu [%]
0.380
0.375
0.370
0.365
0.360
0.355
0.350
0.345
0.340
0.335
0.330
0.325
2023-06-23
2023-07-23
2023-08-22
2023-09-21
2023-10-21
2023-11-20
2023-12-20
2024-01-19
2024-02-18
2024-03-19
2024-04-18
2024-05-18
2024-06-17
2024-07-17
2024-08-16
2024-09-15
2024-10-15
2024-11-14
2024-12-14
2025-01-13
2025-02-12
2025-03-14
2025-04-13
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2025-09-10
2025-10-10
Date of analysis

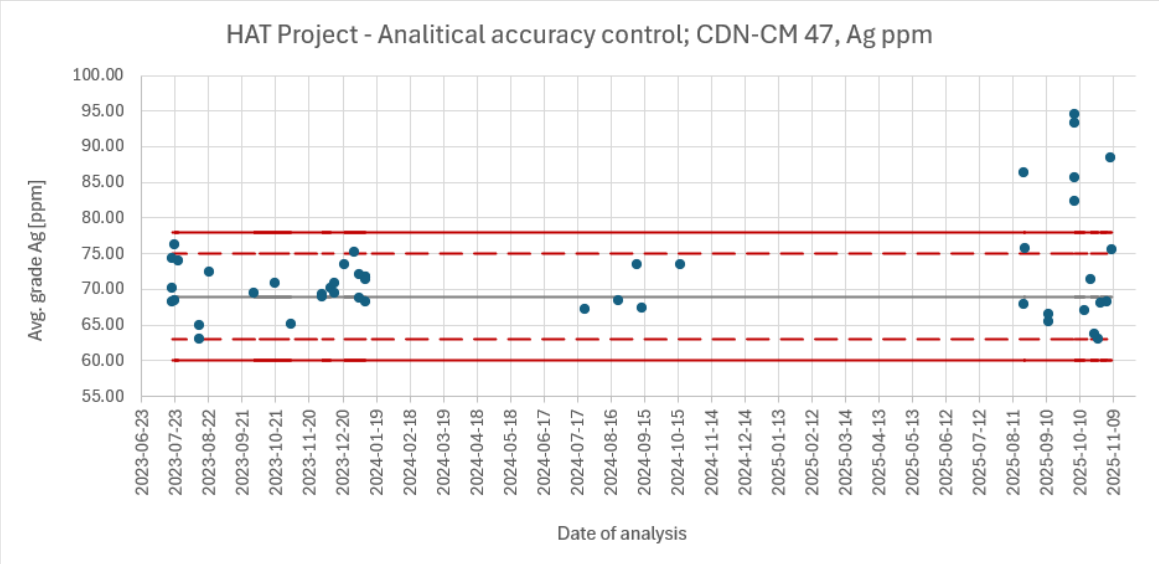

HAT Project - Analitical accuracy control; CDN - CM 40, Ag ppm
Avg. grade Ag [ppm]
24.00
22.00
20.00
18.00
16.00
14.00
12.00
2023-06-23
2023-07-18
2023-08-12
2023-09-06
2023-10-01
2023-10-26
2023-11-20
2023-12-15
2024-01-09
2024-02-03
2024-02-28
2024-03-24
2024-04-18
2024-05-13
2024-06-07
2024-07-02
2024-07-27
2024-08-21
2024-09-15
2024-10-10
2024-11-04
2024-11-29
2024-12-24
2025-01-18
2025-02-12
2025-03-09
2025-04-03
2025-04-28
2025-05-23
2025-06-17
2025-07-12
2025-08-06
2025-08-31
2025-09-25
2025-10-20
2025-11-14
Date of analysis


HAT Project - Analitical accuracy control; CDN-CM 40, Cu %
Avg. grade Cu [%]
0.900
0.850
0.800
0.750
0.700
0.650
0.600
0.550
0.500
0.450
0.400
2023-06-23
2023-07-23
2023-08-22
2023-09-21
2023-10-21
2023-11-20
2023-12-20
2024-01-19
2024-02-18
2024-03-19
2024-04-18
2024-05-18
2024-06-17
2024-07-17
2024-08-16
2024-09-15
2024-10-15
2024-11-14
2024-12-14
2025-01-13
2025-02-12
2025-03-14
2025-04-13
2025-05-13
2025-06-12
2025-07-12
2025-08-11
2025-09-10
2025-10-10
2025-11-09
Date of analysis


HAT Project - Analitical accuracy control; CDN-CM 40, Au ppm
Avg. grade Au [ppm]
1.80
1.70
1.60
1.50
1.40
1.30
1.20
1.10
1.00
0.90
2023-06-23
2023-07-23
2023-08-22
2023-09-21
2023-10-21
2023-11-20
2023-12-20
2024-01-19
2024-02-18
2024-03-19
2024-04-18
2024-05-18
2024-06-17
2024-07-17
2024-08-16
2024-09-15
2024-10-15
2024-11-14
2024-12-14
2025-01-13
2025-02-12
2025-03-14
2025-04-13
2025-05-13
2025-06-12
2025-07-12
2025-08-11
2025-09-10
2025-10-10
2025-11-09
Date of analysis


HAT Project - Analitical accuracy control; CDN-CM 47, Au ppm
Avg. grade Au [ppm]
1.50
1.40
1.30
1.20
1.10
1.00
0.90
0.80
2023-06-23
2023-07-23
2023-08-22
2023-09-21
2023-10-21
2023-11-20
2023-12-20
2024-01-19
2024-02-18
2024-03-19
2024-04-18
2024-05-18
2024-06-17
2024-07-17
2024-08-16
2024-09-15
2024-10-15
2024-11-14
2024-12-14
2025-01-13
2025-02-12
2025-03-14
2025-04-13
2025-05-13
2025-06-12
2025-07-12
2025-08-11
2025-09-10
2025-10-10
2025-11-09
Date of analysis


HAT Project - Analitical accuracy control; CDN-CM 47, Cu %
Avg. grade Cu [%]
0.800
0.775
0.750
0.725
0.700
0.675
0.650
2023-05-14
2023-06-13
2023-07-13
2023-08-12
2023-09-11
2023-10-11
2023-11-10
2023-12-10
2024-01-09
2024-02-08
2024-03-09
2024-04-08
2024-05-08
2024-06-07
2024-07-07
2024-08-06
2024-09-05
2024-10-05
2024-11-04
2024-12-04
2025-01-03
2025-02-02
2025-03-04
2025-04-03
2025-05-03
2025-06-02
2025-07-02
2025-08-01
2025-08-31
2025-09-30
2025-10-30
2025-11-29
Date of analysis


HAT Project - Analitical accuracy control; CDN-CM 47, Ag ppm
Avg. grade Ag [ppm]
100.00
95.00
90.00
85.00
80.00
75.00
70.00
65.00
60.00
55.00
2023-06-23
2023-07-23
2023-08-22
2023-09-21
2023-10-21
2023-11-20
2023-12-20
2024-01-19
2024-02-18
2024-03-19
2024-04-18
2024-05-18
2024-06-17
2024-07-17
2024-08-16
2024-09-15
2024-10-15
2024-11-14
2024-12-14
2025-01-13
2025-02-12
2025-03-14
2025-04-13
2025-05-13
2025-06-12
2025-07-12
2025-08-11
2025-09-10
2025-10-10
2025-11-09
Date of analysis