Schaft Creek Preliminary Economic Assessment (PEA), NI 43-101 Technical Report



PRESENTED TO

Copper Fox Metals Inc.

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ACRONYMS & ABBREVIATIONS

Abbreviations	Definition
AA	Atomic Absorption
AAS	Atomic Absorption Spectrophotometer
ABA	Acid-Base Accounting
ADIS	Automated Digital Imaging System
Adj TNPR	Adjusted Total-Sulphur-Based Net Potential Ratios
AEP	Annual Exceedance Probability
AG	Autogenous Grinding
AGL	Associated Geosciences Ltd.
Ai	Abrasion Index
AIA	Archaeological Impact Assessment
AIR	Application Information Requirements
AN	Andesites
AP	Acid Generation Potential
ARD	Acid Rock Drainage
Asarco	American Smelting and Refining Company
BAFAun	Boreal Altai Fescue Alpine Undifferentiated
BCEAA	British Columbia Environmental Assessment Act
BCWQG	British Columbia Water Quality Guidelines
BGC	BGC Engineering Inc.
BMP	Best Management Practices
BQL	Bob Quinn Lake
BQLA	Bob Quinn Lake Airport
BV	Bureau Veritas
BWi	Ball Mill Work Index
$CaCO_3$	Calcium Carbonate
Capex	Capital Cost Estimate
CCME	Canadian Council of Ministers for the Environment
CCTV	Closed Circuit Television
CDA	Canadian Dam Association
CDE	Canadian Development Expense
CEA	Canadian Environmental Assessment

Abbreviations	Definition
CEAA	Canadian Environmental Assessment Act
CEE	Canadian Exploration Expense
CESL	Cominco Engineering Services Limited
CIS	Cassiar Iskut-Stikine
CLRA	Canadian Labour Relations Association
COD	Chemical Oxygen Demand
Copper Fox	Copper Fox Metals Inc.
CRM	Certified Reference Material
CTCA	Cumulative Tax Credit Account
Cu	Copper
CWi	Crushing Work Index
DCIP	Direct Current Induced Polarization
DCS	Distributed Control System
DEM	Digital Elevation Model
DTM	Digital Terrain Model
DWi	Drop-Weight Index
EA	Environmental Assessment
EAO	Environmental Assessment Office
EGL	Effective Grinding Length
EIS	Environmental Impact Statement
EMLI	BC Ministry of Energy, Mines and Low Carbon Innovation
EPCM	Engineering, Procurement, and Construction Management
EPRP	Emergency Preparedness and Response Plan
EPT	Ephemeroptera, Plectoptera, and Trichoptera
ESSFmc	Engelmann Spruce Subalpine Fir Moist Cold
FCF	Free Cash Flow
FO	Fuel Oil
FS	Feasibility Study
G&A	General and Administration
GPS	Global Positioning System
Hazen	Hazen Research Inc.
HAZOP	Hazard and Operability Analysis
Hecla	Hecla Mining Company

Abbreviations	Definition
HEL	HYYPPA Engineering, LLC
HPGR	High-pressure Grinding Rolls
HSE	Health, Safety, and Environmental
HVAC	Heating, Ventilation, and Air Conditioning
I	Indicated Resources
I/O	Input/Output
ICOLD	International Commission on Large Dams
ICP-ES	Inductively Coupled Plasma Emission Spectroscopy
ICP-MS	Inductively Coupled Plasma Mass Spectrometry
IDF	Inflow Design Flood
IP	Induced Polarization
IRR	Internal Rate of Return
JKTech	JKTech Pty Ltd.
K	Potassium
L	Low
LAN	Local Area Network
Liard	Liard Copper Mines Ltd.
LiDAR	Light Detection and Ranging
LOI	Loss on Ignition
LOM	Life of Mine
LRMP	Land and Resource Management Plan
M	Measured Resources
MAC	Mining Association of Canada
MAP	Mean Annual Precipitation
MAPA	Mines Act Permit Application
MCC	Motor Control Centre
MCE	Maximum Credible Earthquake
MDE	Maximum Design Earthquake
MIBC	Methyl Isobutyl Carbinol
ML	Metal Leaching
MSDS	Material Safety Data Sheet
MSE	Mechanically Stabilized Earth
MYAB	Multi-Year Area-Based

Abbreviations	Definition
MT	Magneto-Telluric
N	Normal
NI 43-101	National Instrument 43-101
NN	Nearest Neighbour
NP	Neutralization Potential
NPAG	Non-Potentially Acid Generating
NPI	Net Proceeds Interest
NPV	Net Present Value
NSR	Net Smelter Return
NTS	National Topographic System
NTL	Northwest Transmission Line
OIS	Operator Interface Stations
OK	Ordinary Kriging
OMS	Operation, Maintenance, and Surveillance
Opex	Operating Expense
OSA	Overall Slope Angle
P	Phosphate
P&ID	Piping and Instrumentation Diagram
PAG	Potential Acid Generating
Paramount	Paramount Mining Ltd.
PAX	Potassium Amyl Xanthate
PC	Personal Computer
PEA	Preliminary Economic Assessment
PES	Project Execution Strategy
PEX	Potassium Ethyl Xanthate
PFD	Process and Utility Flow Diagram
PFS	Pre-Feasibility Study
PLC	Programmable Logic Controller
PMF	Probable Maximum Flood
PO	Purchase Order
Polysius	Polysius Research Centre
Project	Schaft Creek Project
QA/QC	Quality Assurance and Quality Control

Abbreviations	Definition
QP	Qualified Professional
QPO	Quantifiable Performance Objectives
QQ	Quantile-Quantile
QMZ	Quartz Monzonite
Rescan	Rescan Environmental Services Ltd.
RMZ	Resource Management Zone
ROM	Run of Mine
RPAS	Remote Piloted Aerial System
RQD	Rock Quality Designation
RSF	Rock Storage Facility
RWi	Rod Mill Work Index
S(-2)	Sulphide Sulphur
S(t)	Total Sulphur
SABC	SAG Mill, Ball Mill, Pebble Crushers
SAG	Semi-autogenous Grinding
SD	Standard Deviation
SEX	Sodium Ethyl Xanthate
SG	Specific Gravity
Silver Standard	Silver Standard Mines Ltd.
SLD	Single Line Diagram
SMA	Standard Major Axis
SMC	SAG Mill Comminution
sPOR	Syn-mineral Porphyry Dikes
SRTM	Shuttle Radar Topography Mission
SWE	Snow Water Equivalent
TAC	Technical Advisory Committee
TAP	Total-Sulphur-Based Acid Potentials
Teck	Teck Resources Limited
THREAT	Tahltan Heritage Resources Environmental Assessment Team
TNDC	Tahltan Nation Development Corporation
TSF	Tailings Storage Facility
TSFA	Terrain Stability Field Assessment
UCS	Unconfined Compressive Strength

Abbreviations	Definition
VoIP	Voice Over Internet Protocol
WAN	Wide Area Network
WBS	Work Breakdown Structure
WHMIS	Workplace Hazardous Materials Information System
WSF	Waste Storage Facility
ZTEM	Z-Axis tipper electromagnetic

UNITS OF MEASURE

above mean sea level	amsl
acre	ac
ampere	A
annum (year)	a
bank cubic metres	bm^3
billion	B
billion tonnes	Bt
billion years ago	Ga
British thermal unit	BTU
centimetre	cm
cubic centimetre	cm^3
cubic feet per minute	cfm
cubic feet per second	ft^3/s
cubic foot	ft^3
cubic inch	in^3
cubic metre	m^3
cubic yard	yd^3
Coefficients of Variation	CVs
day	d
days per week	d/wk
days per year (annum)	d/a
dead weight tonnes	DWT
decibel adjusted	dBA
decibel	dB
degree	°
degrees Celsius	°C
diameter	ø
dollar (United States)	USD$
dollar (Canadian)	CAD$
dry metric ton	dmt
foot	ft
gallon	gal
gallons per minute (US)	gpm
gauge	ga
gigajoule	GJ
gigapascal	GPa
gigawatt	GW
gram	g
grams per litre	g/L
grams per tonne	g/t
greater than	>
hectare (10,000 m^2)	ha
hertz	Hz
horsepower	hp

hour	h
hours per day	h/d
hours per week	h/wk
hours per year	h/a
inch	"
kilo (thousand)	k
kilogram	kg
kilograms per cubic metre	kg/m^3
kilograms per hour	kg/h
kilograms per square metre	kg/m^2
kilometre	km
kilometres per hour	km/h
kilopascal	kPa
kilotonne	kt
kilovolt	kV
kilovolt-ampere	kVA
kilowatt	kW
kilowatt hour	kWh
kilowatt hours per tonne	kWh/t
kilowatt hours per year	kWh/a
less than	<
litre	L
litres per minute	L/m
megabytes per second	Mb/s
megapascal	MPa
megavolt-ampere	MVA
megawatt	MW
metre	m
metres above sea level	masl
metres Baltic sea level	mbsl
metres per minute	m/min
metres per second	m/s
metric ton (tonne)	t
microns	µm
milligram	mg
milligrams per litre	mg/L
millilitre	mL
millimetre	mm
million	M
million bank cubic metres	Mbm^3
million bank cubic metres per annum	Mbm^3/a
million tonnes	Mt
minute (plane angle)	'
minute (time)	min
month	mo
Neutron	N

ounce	oz
pascal	Pa
centipoise	mPa·s
parts per million	ppm
parts per billion	ppb
percent	%
pound(s)	lb
pounds per square inch	psi
revolutions per minute	rpm
second (plane angle)	"
second (time)	s
specific gravity	SG
square centimetre	cm^2
square foot	ft^2
square inch	in^2
square kilometre	km^2
square metre	m^2
thousand tonnes	kt
Three Dimensional	3D
Three Dimensional Model	3DM
tonne (1,000 kg)	t
tonnes per day	t/d
tonnes per hour	t/h
tonnes per year	t/a
tonnes seconds per hour metre cubed	ts/hm^3
troy ounce	t oz
volt	V
week	wk
weight/weight	w/w
wet metric ton	wmt
year (annum)	a

GLOSSARY

Throughout the code, certain words are used in a general sense when a more specific meaning may be attached to them by particular commodity groups within the industry. In order to avoid unnecessary duplication, a non-exclusive list of generic terms is tabulated below, together with other terms that may be regarded as synonymous for the purposes of this technical report.

Generic Term	Synonyms and Similar Terms	Intended Generalized Meaning
Tonnage	Quantity, Volume	An expression of the amount of material of interest irrespective of the units of measurement (which should be stated when figures are reported).
Grade	Quality, Assay, Analysis (Value)	Any physical or chemical measurement of the characteristics of the material of interest in samples or product. Note that the term quality has special meaning for diamonds and other gemstones. The units of measurement should be stated when figures are reported.
Metallurgy	Processing, Beneficiation, Preparation, Concentration	Physical and/or chemical separation of constituents of interest from a larger mass of material. Methods employed to prepare a final marketable product from material as mined. Examples include screening, flotation, magnetic separation, leaching, washing, roasting, etc.
Recovery	Yield	The percentage of material of initial interest that is extracted during mining and/or processing. A measure of mining or processing efficiency.
Mineralization	Type of Deposit, Orebody, Style of Mineralization	Any single mineral or combination of minerals occurring in a mass or deposit, of economic interest. The term is intended to cover all forms in which mineralization may occur, whether by class of deposit, mode of occurrence, genesis, or composition.
Mineral Reserves	Ore Reserves	“Ore Reserves” is preferred under the JORC Code but “Mineral Reserves” is the recommended term under CIM guidelines and NI43-101 rules.
Cut-off Grade	Product Specifications	The lowest grade or quality of mineralized material that qualifies as economically mineable and available in a given deposit. May be defined on the basis of economic evaluation, or on physical or chemical attributes that define an acceptable product specification.

TETRA TECH

1.0 SUMMARY

In 2020, Copper Fox commissioned a team of engineering consultants to complete this Preliminary Economic Assessment (PEA), in accordance with NI 43-101 Standards of Disclosure for Mineral Projects.

Components of this PEA were completed by the following consultants:

- Tetra Tech Canada Inc. (Tetra Tech): overall project management, mining methods, metallurgical testing, mineral processing and recovery methods, project infrastructure, capital and operating cost estimates, and economic analysis.
- Red Pennant Communications Corp. (Red Pennant): project description and location, accessibility, history, geological setting, deposit types, exploration, drilling, Mineral Resource estimate and adjacent properties.
- Knight Piésold Ltd. (Knight Piésold): tailings and waste rock management and power transmission (including capital costs).
- Greenwood Environmental Ltd. (Greenwood) – environmental studies, permitting, and social or community impact.
- Ruskin Construction Ltd./Allnorth Consultants Ltd. (Ruskin/Allnorth) – Galore Creek Access Road and More Canyon Bridge.
- McElhanney Consulting Services Ltd. (McElhanney) – Mess Creek Access Road.

A summary of the consultants responsible for each section of this report is detailed in Table 2-1.

All dollar figures expressed in this report are in United States Dollar (USD) unless otherwise noted.

1.1 Project Description

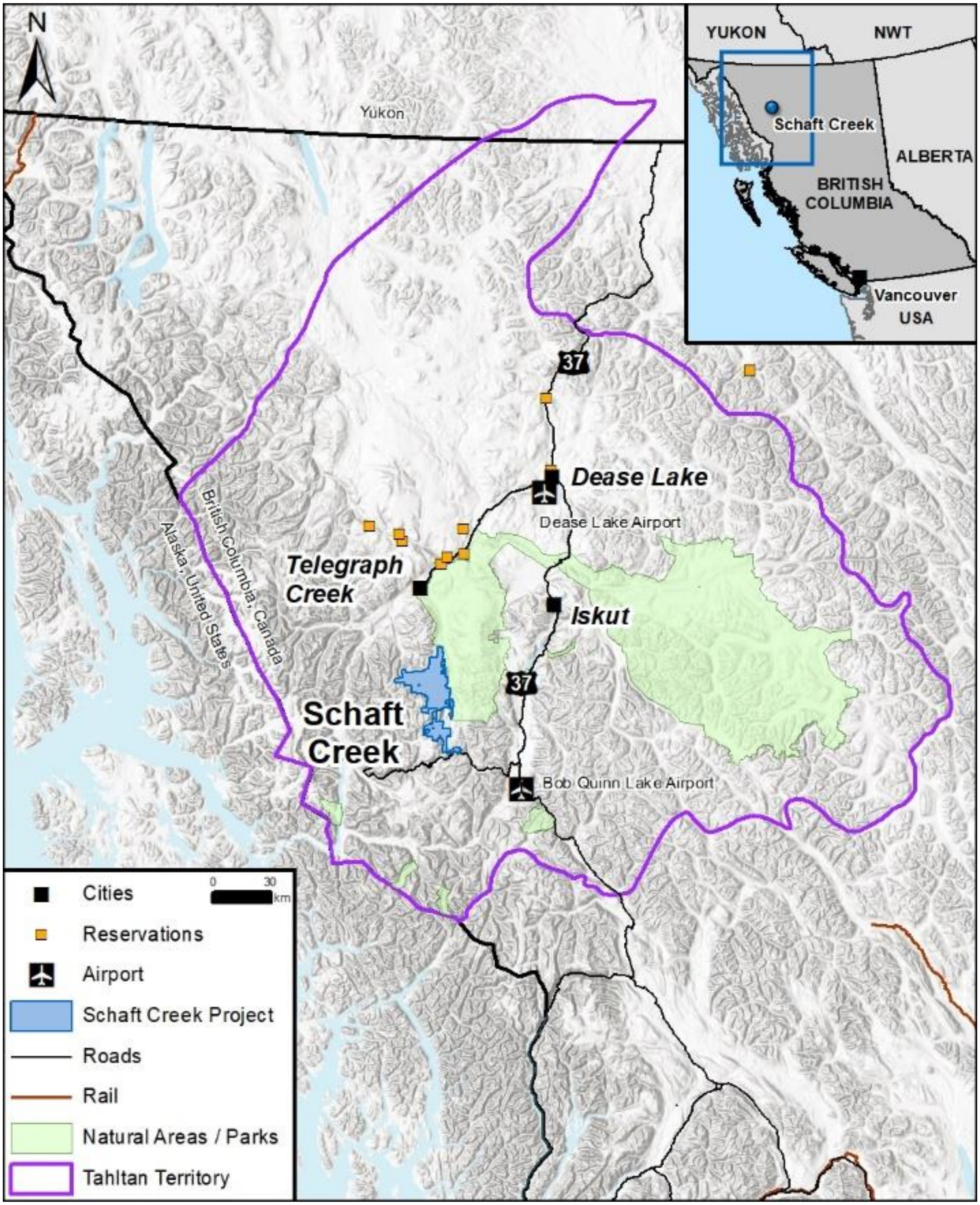
The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the results of the PEA will be realized.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

Schaft Creek is a large copper-molybdenum-gold porphyry deposit located in Tahltan territory in northwestern British Columbia, approximately 60 kilometres south of Telegraph Creek and 37 kilometres northeast of the Galore Creek property.

The Schaft Creek Project is managed through the Schaft Creek Joint Venture (Schaft Creek JV) formed in 2013 between Teck Resources Limited (Teck) (75%) and Copper Fox Metals Inc. (Copper Fox) (25%) with Teck being the Operator.

As illustrated in Figure 1-1, the Project is located approximately 61 km south of the village of Telegraph Creek, 120 km southwest of Dease Lake, 45 km due west of Highway 37, 70 km west northwest of Bob Quinn Lake (BQL), and approximately 375 km northwest of Smithers. The Project comprises approximately 55,779.56 ha encompassing portions of the Schaft Creek and Mess Creek Valleys, and Mount LaCasse situated in the Cassiar/Liard Mining Division of northwestern BC, Canada. Access to the Project is via helicopter and fixed wing aircraft from either Dease Lake, BQL, or Smithers.

Figure 1-1: Location Map of the Schaft Creek Project



Source: Schaft Creek JV, 2020

1.2 Geology, Mineralization, Status of Exploration, and Mineral Resource Estimate

In 2021 Copper Fox completed a Resource Model update taking into account 6,087 metres of new drilling completed from 2013 to 2015; 42,888 metres of re-logging completed between 2013 and 2015; 1:5000 scale Anaconda-style geological mapping completed over the deposit in 2014; as well as improvements made to the database through a life of project Quality Assurance and Quality Control (QA/QC) review. The updated Mineral Resource statement was released as part of Copper Fox's Resource Estimate and it is presented in Table 1-1.

Table 1-1: 2021 Mineral Resource Statement

		Average Value				Material Content			
	Mass	Cu	Au	Mo	Ag	Cu	Au	Mo	Ag
Category	Mt	%	g/t	%	g/t	million lb.	million t. oz	million lb.	million t. oz
Measured	176	0.32	0.22	0.018	1.46	1,262	1.28	71	8.26
Indicated	1,169	0.25	0.15	0.017	1.22	6,503	5.69	440	46.00
Total M&I	**1,346**	**0.26**	**0.16**	**0.017**	**1.25**	**7,764**	**6.97**	**511**	**54.25**
Inferred	344	0.17	0.11	0.013	0.84	1,303	1.18	96	9.28

Notes:

Mt=millions of tonnes, Cu=copper, Au=gold, Mo=molybdenum, Ag=silver, lb.= pounds, t.oz.= troy ounces.

1. Mineral Resources are reported using the 2014 CIM Definition Standards.
2. The QP for the estimate is Mr. Michael F. O'Brien, P.Geo., Red Pennant Resources Geoscience. Mineral Resources have an effective date of 15 January 2021.
3. Mineral Resources are reported within a conceptual constraining pit shell that includes the following input parameters: $3/lb Cu, $10/lb Mo, $1,200/oz Au, $20/oz Ag, mining cost of CAD$1.95/t mined, processing cost of CAD$4.94/t processed and pit slope angles that vary from 40–44º. Metal recoveries; Cu 86.6%, Au 73%, Mo 58.8%, Ag 48.3%.
4. Mineral Resources are reported using a net smelter return cut-off of USD$4.31/t, and a CAD$ to USD$ exchange rate of 1.20.
5. Metal prices are in USD$.
6. Tonnes (t) are metric tonnes, with copper and molybdenum grades as percentages, and gold and silver grades as gram per tonne units. Copper and molybdenum metal content is reported in lbs and gold and silver content is reported in troy ounces.
7. Totals may not sum due to rounding.

1.3 Mining

The updated mine plan, used a 20 m x 20 m x 15 m block model for pit shell generation, pit design, and production scheduling. Pit optimizations were performed using the Whittle® software package. The metal prices used in pit optimizations are $3.15/lb Cu, $1,300/oz Au, $10.00/lb Mo, and $20.00/oz Ag.

Mining recovery and dilution were assumed to be 97.7% and 0.3%, respectively (Tetra Tech, 2013). Process recoveries used in pit optimizations are variable, based on metal grades as discussed in Section 13.0 of this report.

The mining study for this PEA was based on a variable throughput rate for processing based on the updated geometallurgical work and material hardness outlined in Table 17-2 in Section 17.0 of this report.

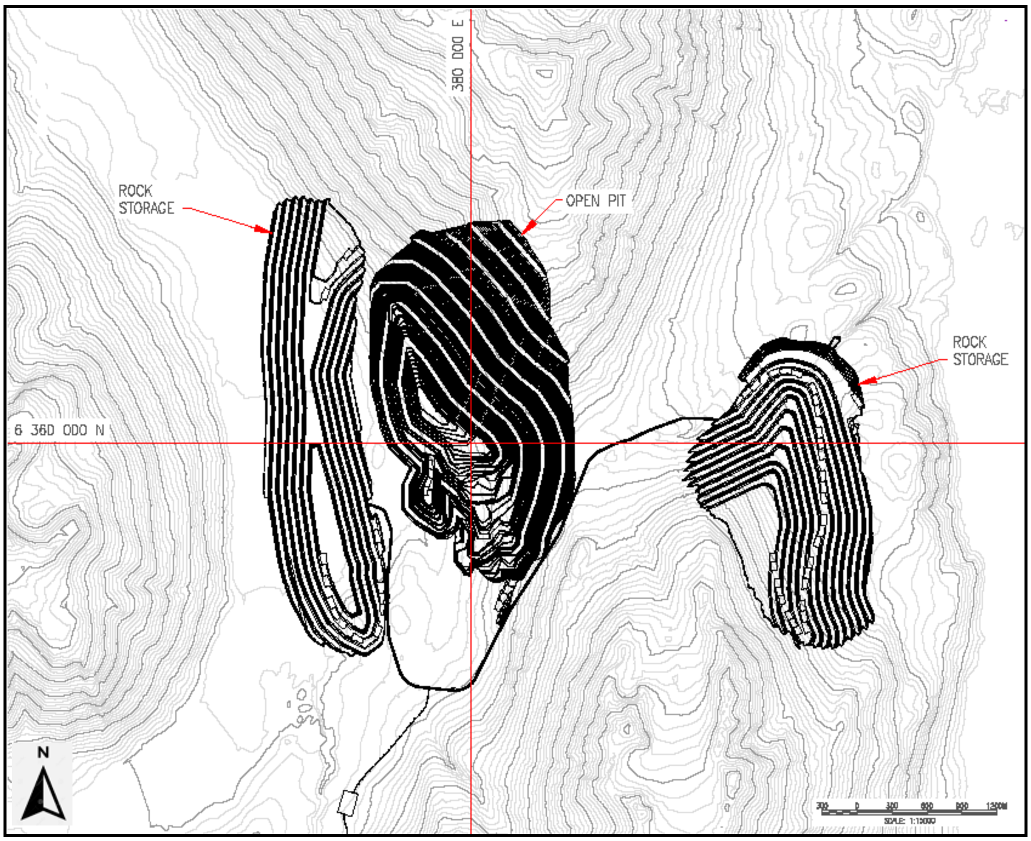
The mine plan is based on a conventional open pit truck-and-shovel operation. The primary loading units will be the electric rope shovels with 45 m^3 buckets. Hauling will be performed using 360 t haul trucks. A stockpiling strategy has been completed to allow the mine to give priority to higher-value material for processing and to ensure that the required mill feed is maintained. A variable cut-off grade was applied on a year-by-year basis, based on the available mineralized material for the period.

Waste rock from the pit will be stored in the east and west rock storage facilities (RSFs). The total capacity for rock storage is 571.1 million m^3. Some of the mined-out waste rock from the pit will be used for embankment construction.

Over the 21-year life of mine (LOM), the open pit will be producing 1.03 billion tonnes (Bt) of waste rock and 1.03 Bt of mill feed with average grades of 0.26% Cu, 0.16 g/t Au, 0.017% Mo, and 1.23 g/t Ag. The overall LOM strip ratio is approximately 1.

Final pit and the waste rock storage facilities are shown in Figure 1-2.

Figure 1-2: General Layout of Open Pit Area



1.4 Metallurgy

The Schaft Creek deposit is a calc-alkalic polymetallic (copper-molybdenum-gold-silver) porphyry deposit, with a low-sulphidation state, and overlapping mineralized zones. Historically, the deposit has been divided into three mineralization zones: the Main (or Liard) Zone, the Paramount Zone, and the West Breccia Zone. The Liard Zone is the largest zone of mineralization, and the Paramount Zone and the West Breccia Zone were combined into one zone at the end of the 2011 field season. In 2015, the mineralization was classified into four rock types: Volcanic, Intrusive, Porphyry, and Breccias to verify their grindability and comminution circuit design (the Schaft Creek 2015 GeoMet Program). The main focus of the 2015 GeoMet program was to evaluate the primary comminution circuits proposed by the 2013 study and investigate the comminution variability for updating the throughput projection.

Between 2004 and 2015, G&T/ALS, PRA (Inspectorate), Hazen, Polysius Research Centre (Polysius), and Cominco Engineering Services Ltd. (CESL) conducted extensive metallurgical tests on samples from the various GeoMet units and mineralization zones of the Schaft Creek deposit to support various studies.

Metallurgical test programs conducted included mineralogy, grindability, flotation, and dewatering tests. The most recent test work was the 2015 Schaft Creek JV GeoMet Program. The metallurgical test program focused on comminution test work, including SAG (semi-autogenous grinding) Mill Comminution (SMC) tests and Bond ball mill work index (BWi) determination, and was conducted by ALS. After the 2015 comminution test program, the primary grinding circuit was further assessed by SimSAG Pty. Ltd. using JK simulations for blasting, crushing, and grinding processes.

The mineralogical and metallurgical test studies showed:

- Chalcopyrite was the dominant copper sulphide mineral together with ancillary bornite and chalcocite.
- The other main sulphide mineral in the mineralization was pyrite. The pyrite contents of the samples used in the test programs ranged from 0.04% to 1.0%. The 2012 test program showed 0.1% to 0.8% pyrite content, averaging at approximately 0.3%.
- Comminution characteristics indicate that the mineralized zones can be classified as hard with respect to SAG mill and ball mill grinding. The average A x b values for the breccia, intrusive and porphyry are functionally equivalent at approximately 34, while the volcanic lithology represents a distinctly harder mineralization type with an A x b value of 31. The BMWi value also varies distinctly with lithology ranging from 16.6 kWh/t to 22.4 kWh/t, indicative of a very hard mineralized material. The average Ai is 0.25 g, fluctuating from 0.17 g to 0.57 g.
- Test work to date supports a process primary grind size of 80% passing 150 µm.
- The copper and molybdenum bulk flotation locked cycle test results showed that the mineral samples responded well to a simple, conventional process. Recovery is predominantly feed grade dependent, with some performance influence from copper mineralization.
- Bulk rougher regrind size requirements of 80% passing 25 to 30 µm were determined in preparation for the subsequent three stage cleaner flotation process.
- At an average primary grind size of 80% passing 151 µm, G&T test results show that on average, 86.2% of the copper was recovered from the head sample containing approximately 0.37%

copper. The other associated metal recoveries were 73.3% for gold, 55.7% for silver, and 71.9% for molybdenum. The average feed grades of the samples were approximately 0.27 g/t gold, 2.7 g/t silver, and 0.019% molybdenum. On average, the concentrate produced contained 30.9% copper. The average data from G&T and PRA show that at the primary grind size of 80% passing 146 µm, 86.7% of the copper reported to the copper concentrate at a grade of 29.9% copper. The gold, silver, and molybdenum recoveries to the concentrate were 74.7%, 56.9%, and 73.7%, respectively.

- The required grind size of the molybdenum rougher concentrate for molybdenum separation process was determined to be approximately 20 µm or finer. The inclusion of a leach facility for processing out of specification final molybdenum concentrate may be necessary. Five copper-molybdenum separation locked cycle tests were performed on the bulk concentrates generated from the pilot plant tests. The molybdenum recovery to molybdenum concentrate ranged from 67% to 88%. On average, 73.1% of the molybdenum was recovered to the molybdenum concentrates. The molybdenum concentrate grades ranged from 44% to 50% molybdenum.
- Multi-element assays on the bulk concentrates generated from the locked cycle tests showed, on average, that the impurities of the copper concentrates produced from the mineralization should be below smelting penalty thresholds set forth by most smelters.

1.5 Mineral Processing

The proposed processing plant is designed to process the Schaft Creek mineralization at a nominal throughput of 133,000 t per day (t/d) (with an availability of 92%) to produce market-grade copper and molybdenum concentrates.

The LOM average mill feed grades are estimated to be 0.26% Cu, 0.16 g/t Au, 1.23 g/t Ag, and 0.017% Mo. The estimated metal recoveries are 83.1% for copper, 71.0% for gold, and 40.3% for silver in copper concentrate and 60.1% of molybdenum in molybdenum concentrate. The LOM average annual production is estimated to be approximately 385,000 t/a of copper concentrate, which contains 28% Cu, 14.1 g/t Au, 63.1 g/t Ag, and 9,780 t/a of molybdenum concentrate at 50% Mo.

A conventional flotation process is proposed for the Project. The processing plant will consist of:

- primary crushing at the mine site
- a crushed mill feed stockpile
- a main processing plant, including:
  - two primary grinding circuits, consisting of two SAG mills, four ball mills, and three pebble crushers (SABC circuits)
  - two copper-molybdenum bulk flotation circuits
  - one copper/molybdenum separation circuit, including molybdenum concentrate leaching to reduce copper/lead contents in the final molybdenum concentrate
  - concentrate dewatering
  - tailings disposal

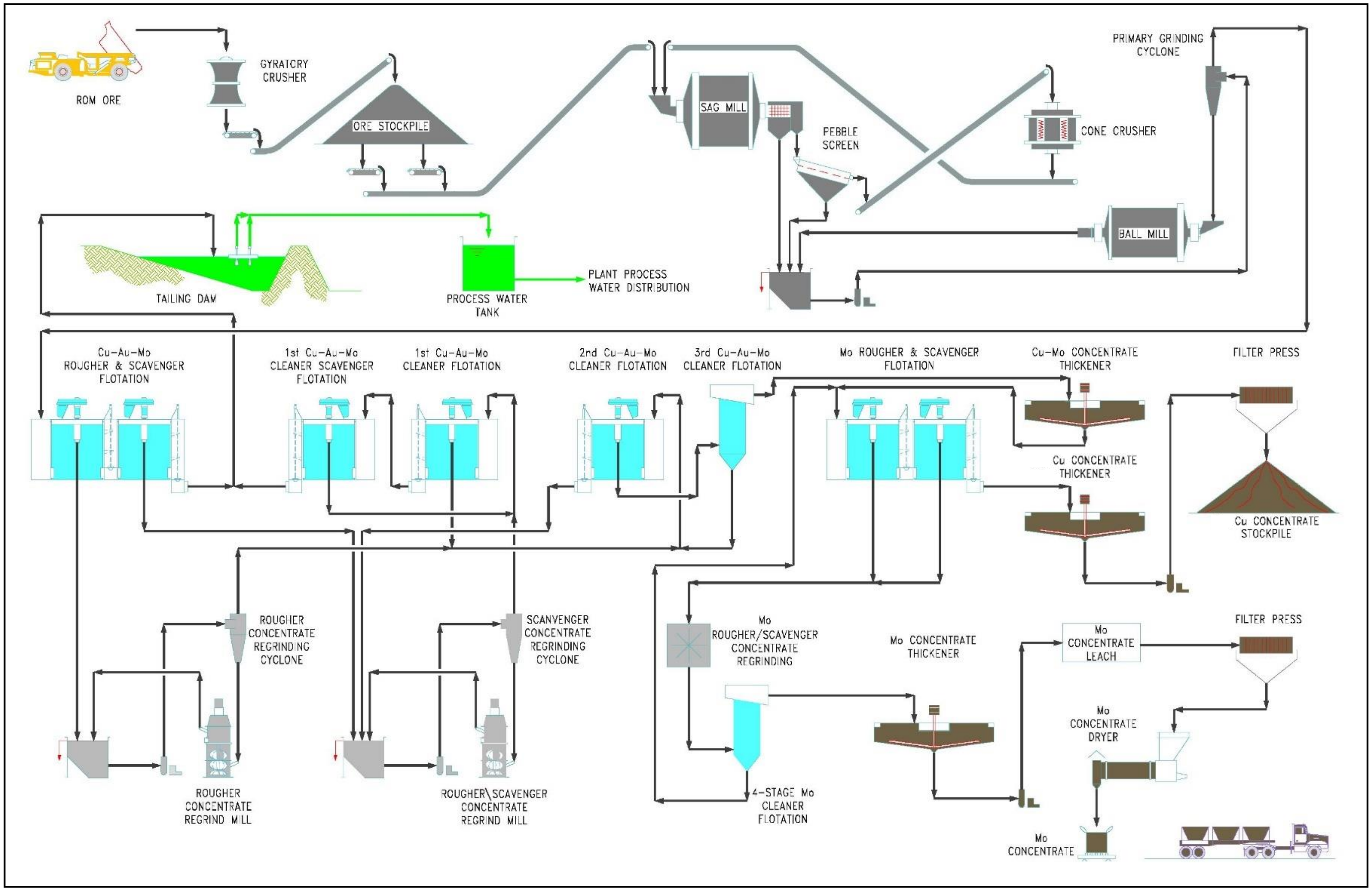
Two gyratory crushers, operating as the primary crushing units, will reduce the Run of Mine (ROM) particle size to approximately 80% passing 120 mm or finer. The crushed mill feed will be conveyed to a stockpile with a live capacity of 120,000 t. The mill feed will then be reclaimed in two parallel lines to two SABC circuits to further reduce particle size down to 80% passing 150 µm.

There will be two trains of copper-molybdenum rougher/scavenger flotation. The products from the primary grinding circuits will feed to the rougher/scavenger flotation circuits, which will produce a high-grade rougher concentrate and a lower-grade rougher/scavenger concentrate. The two concentrates will be separately reground, then upgraded in three stages of cleaner flotation to produce a copper-molybdenum bulk flotation concentrate. The bulk concentrate will be further treated by flotation to produce a molybdenum concentrate and a copper concentrate containing gold and silver. The copper concentrate is estimated to contain approximately 28% copper. The molybdenum concentrate will contain approximately 50% molybdenum after the flotation concentrate is leached using the chloride leaching procedure to reduce copper and lead contents.

The final flotation concentrates will be thickened and then pressure-filtered to a moisture content of approximately 9%, while the molybdenum concentrate will be further dewatered by drying to a moisture content of approximately 4% to 5%. The copper concentrate will be stockpiled and then bulk trucked via Highway 37 to the Port of Stewart for storage and loading for export of the concentrate to foreign markets. The dried molybdenum concentrate will be bagged prior to being trucked to the Fairview Terminals in Prince Rupert for international shipment to the smelter.

The simplified process flowsheet is shown in Figure 1-3.

Figure 1-3: Simplified Processing Flowsheet



1.6 Project Infrastructure

In comparison to the 2013 FS, numerous infrastructure design improvements have been incorporated into the 2021 PEA, including major facilities on and off site, such as waste storage facility (WSF), tailings storage facility (TSF), process plant, ancillary buildings, and airstrip.

The new mine plan has reduced the waste mined by approximately 1.0 Bt, which allows elimination of the south WSF that was included in the 2013 FS.

The revised TSF layout has reduced the number of embankments from three to two and reduced the overall TSF footprint. The TSF has also been relocated closer to the mine and process plant. In addition, the number of tailings lines has been reduced from three to two. The revised TSF design helps reduce the start dam construction volumes, material movement, tailings and reclaim water piping and pumping, construction schedule, and construction and operating costs.

The process plant, truck shop, camp, and most ancillary buildings are closer to the mine site and to a flatter terrain. This helps reduce material moving equipment, access roads, and building pad cut and fill quantities. The length of the overland crusher mill feed conveyor has been reduced significantly. Although the process plant has been moved away from the TSF and towards the mine, the total tailings pipe length remains about the same.

The 2021 PEA proposes upgrading and utilizing an existing airfield near the Project. This eliminates the need of building a new airstrip and terminal at the Project site.

The 100-km fuel delivery pipeline between the plant site and Highway 37 junction in the 2013 FS has been eliminated. Fuel will be delivered to site by inbound freight trucks similar to other operating mines in British Columbia.

Inbound delivery of material and equipment will be directed from a marshalling yard near the highway 37 junction to Project site. The Tahltan Transfer Depot proposed in the 2013 FS has become obsolete.

The site preparation, pad and road cut and fill quantities have been reduced due to infrastructure pad size reductions, flatter terrains, and shortening of roads. Consequently, the capital and operating costs have also been reduced. Table 1-2 shows the differences of selected Project key metrics between the 2013 FS and 2021 PEA.

Table 1-2: Key Infrastructure Metrics Summary

Project Metrics	Unit	2013 FS	2021 PEA	Difference ∆	Comment
Overland Conveyor	m	7,150	2,650	**-4,500**	Pumping slurry is more cost efficient than conveying materials. At about CAD$10,000 per metre, the estimated Capex reduction by shortening the overland conveyor is CAD$45M. The Opex reduction in power consumption is estimated CAD$1.8M per year.
Tailings Line (Total Length)	m	11,250	11,900	**+650**	The third tailings line in 2013 FS has been eliminated, as there were already two tailings lines so that one of them could stay in operation while the other line is temporarily shut down, with a reduced throughput. This practice is common among other mine operations such as Kemess Mine. Despite the increased distance between the process plant and TSF, the additional tailings pipe length is mostly offset by the elimination of the redundant pipe. The total tailings pipe length remains about the same.
Distance between Mine and Crushed Mill Feed Stockpile	m	5,600	1,400	**-4,200**	The shortened distances between the site facilities and the open pit mine have resulted in Capex reductions in site access road construction and Opex reductions in material haulage (e.g. fuel).
Distance between Mine and Process Plant	m	6,500	2,100	**-4,400**	
Distance between Mine and TSF	m	8,000	6,200	**-1,800**	
Distance between Mine and Truck Shop	m	4,400	1,000	**-3,400**	
Distance between Mine and Camp	m	5,000	3,200	**-1,800**	

table continues…

Project Metrics	Unit	2013 FS	2021 PEA	Difference Δ	Comment
Site Airstrip	m	1,800	0	**-1,800**	By eliminating the new airstrip on site and upgrading/using the existing BQLA nearby instead, the CAD$57.4M airstrip Capex has been eliminated, partially offset by the BQLA upgrade Capex and incremental staff transportation Opex. The estimated NPV of the savings is CAD$16.1M over LOM.
Fuel Delivery Pipeline	km	100	0	**-100**	By eliminating the 100-km fuel delivery pipeline from Highway 37 to site, the CAD$54.7M Capex has been eliminated. The incremental fuel haulage cost to cover the distance between Highway 37 and project site is estimated CAD$120K per year.
Site Preparation, Cut and Fill and Roads	Labour Hours	1,007,000	566,000	**-441,000**	By relocating the process plant and most ancillary facilities to flatter locations, the estimated site preparation, pads cut and fill and site access road Capex allowance has been reduced by approximately 43% in terms of construction labour hours.
Capital Cost Before/After Process Plant Relocation[1]	CAD$ M	127.6	72.1	**-55.5**	This is the estimated initial Capex savings from relocating the site facilities.
Operating Cost Before/After Process Plant Relocation[1]	CAD$ M/yr	15.7	8.3	**-7.4**	This is the estimated Opex savings per annum from relocating the site facilities.
TSF Initial Capital Cost	CAD$ M	212.2	178.4	**-33.8**	This is the estimated initial Capex savings from relocating the TSF north embankment 2 km to the south.

Note: BQLA = Bob Quinn Lake Airport; NPV = Net Present Value; Capex = Capital Cost Estimate; Opex = Operating Expense
[1] Excludes TSF and Mining.

Locations of project facilities and other infrastructure items were selected to take advantage of local topography, accommodate environmental considerations, and provide efficient and convenient operation of the mine haul fleet.

Project infrastructure will include the following:

- Completion of the Galore Creek access road to the Schaft Creek turnoff

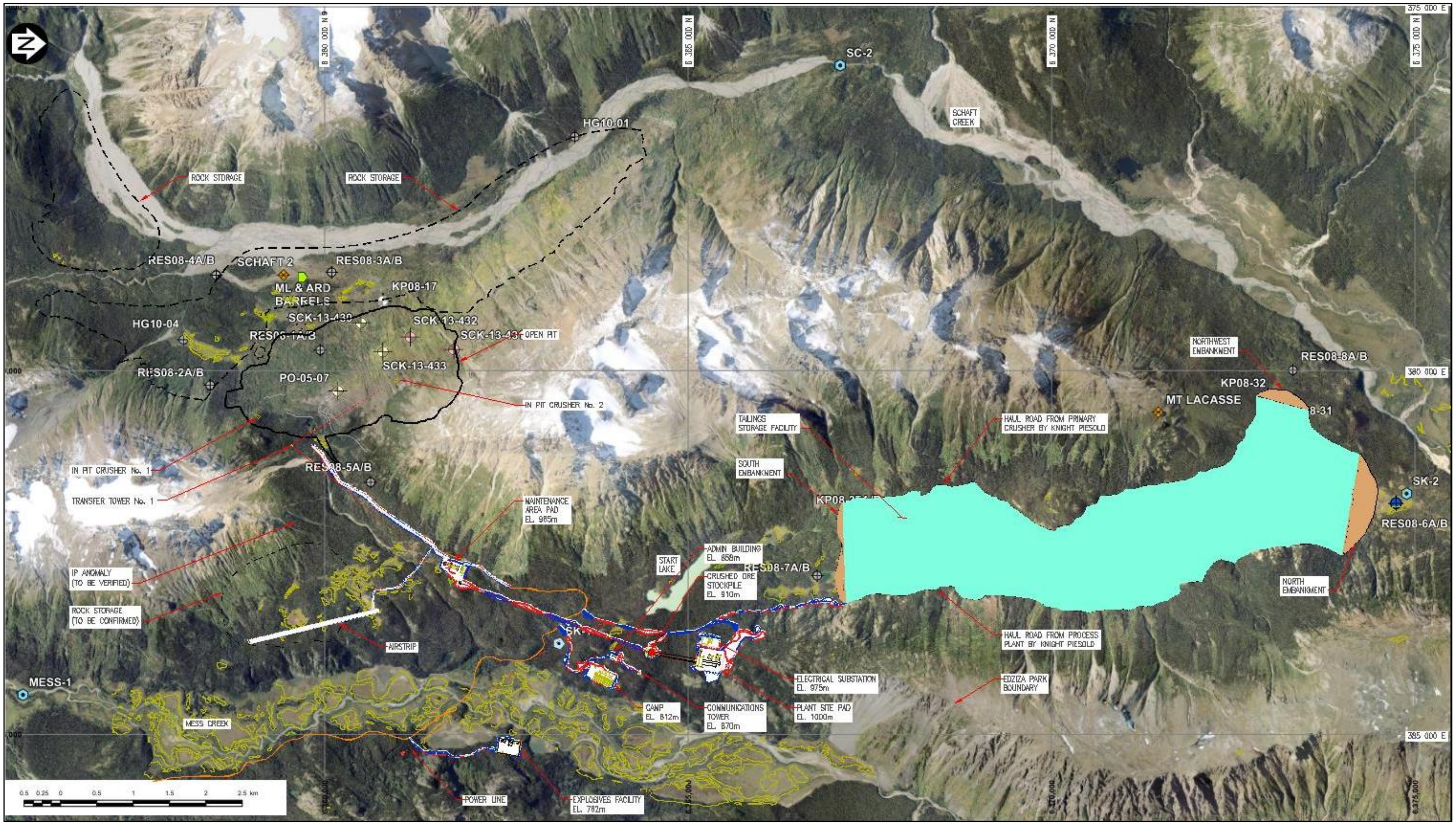
- More Canyon Bridge
- Access roads, including 40 km of new road, from the Schaft Creek turnoff north through the Mess Creek Valley to the mine site
- A TSF to safely manage the tailings and water associated with mill feed processing
- A network of site haul roads
- A complete water supply and distribution system
- A sewage disposal plant
- Process and ancillary facilities, including:
  - A primary crushing facility
  - A mine site crushed mill feed stockpile
  - A crushed mill feed stockpile
  - A pebble crushing building
  - A mill building
  - Reagent storage
  - A warehouse/truck shop/mine dry
  - A cold storage warehouse
  - Facilities for administration and an assay laboratory
  - Facilities for the storage of fuel, explosives, and concentrate
  - An emulsion plant
  - An operation and construction camp
  - A power supply and distribution network
  - Communications infrastructure
  - Diesel fuel storage and distribution

Off-site infrastructure include the following:

- Upgrade of BQLA, with installation of navigation/instrument landing system and a terminal.
- Concentrate storage facility at the port facility in Stewart, British Columbia. Concentrate unloading, handling, and ship loading services will be provided by the port operator).

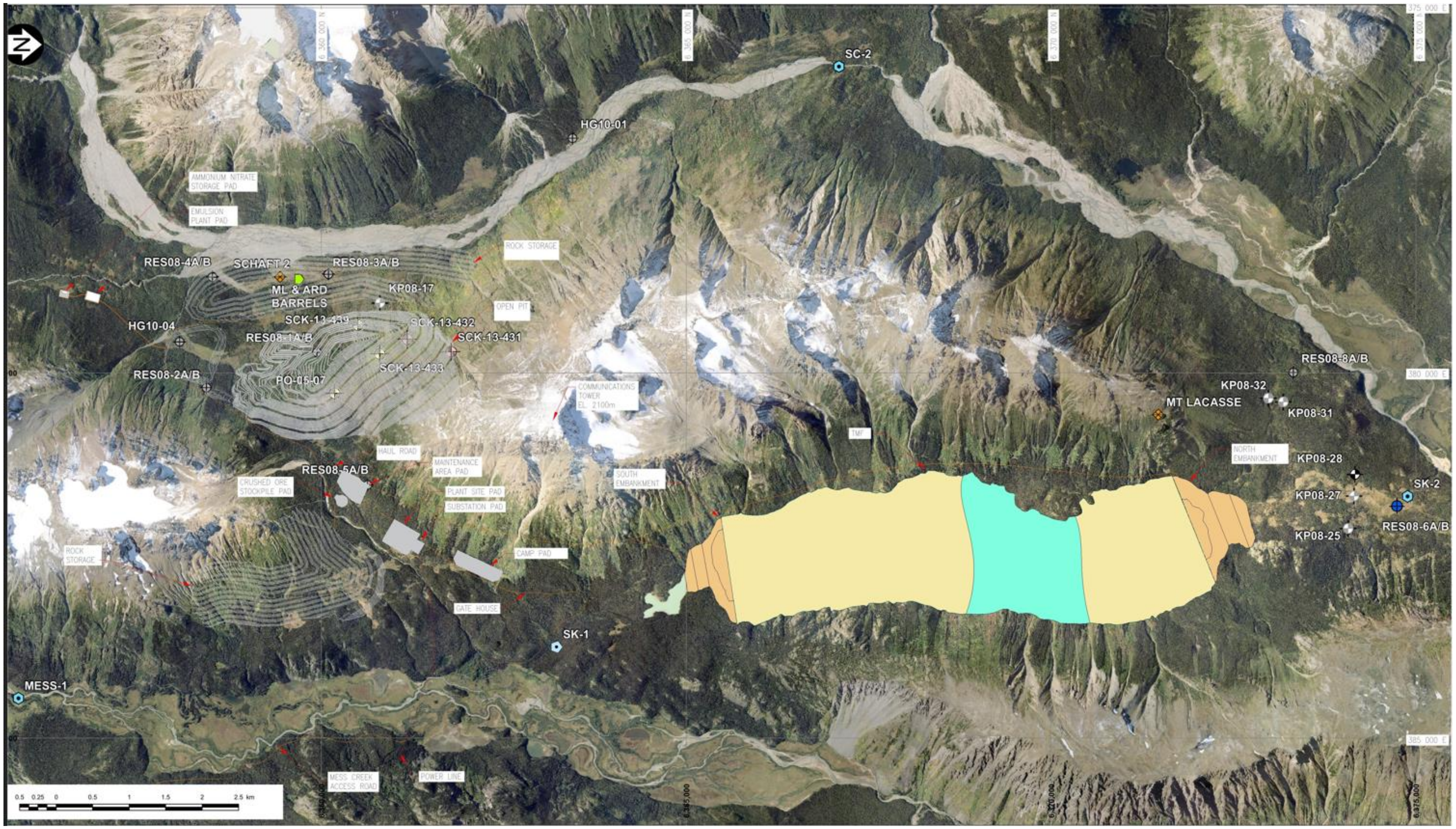
Figure 1-4 and Figure 1-5 show the 2013 FS and 2021 PEA Site Layouts, respectively, such that the reader can see and compare the changes between these studies.

Figure 1-4: 2013 FS Site Layout



Source: Tetra Tech (2020)

Figure 1-5: 2021 PEA Site Layout



Source: Tetra Tech (2020)

1.6.1 Road Access

McElhanney analyzed the project site access road requirements and developed a preliminary road design and associated construction cost estimate. A single lane resource access road with pull outs will be built to support the construction and operation of the Project. This road will utilize the Galore Creek access road for the first 65.2 km from Highway 37, then travel north through the Mess Creek valley for approximately 40 km to the mine site. This proposed access road would require an access and road use agreement that goes 65.2 km from Highway 37 to the Mess Creek Valley acceptable to existing road users, the BC Government and the Tahltan.

As the Project is being considered as a standalone project, the completion of the Galore Creek access road and More Canyon Bridge have been reviewed by Ruskin/Allnorth. The updated design and costs have been included in the overall scope of the Project.

1.6.2 Bob Quinn Lake Airport Upgrade

Mine personnel will be transported to the site by regular daily charter flights to BQLA, located approximately 120 km by road from site, and the scheduled bus service between BQLA and site. The BQL Airport will require runway extension, navigation instrument upgrades, and a new terminal.

1.6.3 Tailings Storage Facility

The TSF is designed to provide secure and permanent storage of approximately 1,000 Mt of tailings for a 133,000 t/d mining operation with a 21-year mine life. Tailings will be impounded in a TSF in the Start Creek and Skeeter Creek Valley, located to the northwest of the open pit, and due north of the plant site. The TSF includes the following:

- Two zoned embankments constructed using cyclone sand
- Upslope surface water diversion channels
- Seepage and embankment runoff collection systems
- Tailings transport and deposition system
- Reclaim water system
- Surplus water removal system
- Tailings beaches
- Supernatant water pond

The TSF design has been optimized from the 2013 FS design to reduce the TSF footprint, dam construction volumes, material movement, tailings and reclaim water piping and pumping, construction schedule, and construction and operating costs.

The north embankment has been relocated approximately 2 km south of the 2013 FS location to minimize initial construction requirements and the starter embankment construction volumes. The south embankment has been relocated by 750 m to the south, relative to the 2013 FS location. This arrangement also allows for any future expansion of the Project beyond the currently proposed

21 years LOM for up to 2 Bt without significantly changing the footprint. The number of embankments has been reduced from 3 to 2.

The conceptual layouts were completed using 5H:1V downstream slopes for the embankment shells and a centre-line method of construction.

The north and south embankments will be constructed as cyclone sand structures. The north starter embankment will be constructed using rockfill from a local quarry and the south starter embankment will use non-potentially acid generating pit-run waste material. The starter embankments will include an upstream geosynthetic liner; the raised portions will include a lower permeability central zone, selective tailings deposition, and downstream filter and transition zones to manage seepage.

The ongoing embankment raises will incorporate a filter zone and a transition zone that will be supported by the upstream and downstream shell zones. The filter and transition zones will prevent the downstream migration of the tailings. The tailings beach, which will have a relatively low permeability, will provide confinement for the supernatant pond and mitigate drainage into the embankment. The filter and transition zones will be constructed using suitable granular, free-draining material.

Construction water management at the TSF will commence approximately two years prior to mill start-up and coincide with initial construction of the facility. This phase is characterized by extensive clearing, grubbing and stripping, development of access roads and haul roads, and establishment of water management and sediment control systems.

Non-contact water will be diverted around the TSF during operations to the extent practical. Undiverted runoff will be stored within the TSF. Process water will be discharged into the TSF with the tailings slurry and supernatant water will be reclaimed back to the mill for use in mill feed processing. Surplus water will be removed from the facility to prevent the accumulation of water and discharged to Schaft Creek. Seepage from the TSF will be collected in the seepage collection ponds downstream of the embankments and recycled to the TSF supernatant pond.

1.7 Environmental

A number of project-specific baseline studies were completed for the Schaft Creek Project between 2006 and 2012, in support for a potential Environmental Assessment Application. This includes baseline collection of dust, noise, meteorological, groundwater, and surface water monitoring studies; aquatic and fisheries studies; collection of physical, chemical, and biological marine data; sediment quality; wetland, flora, and fauna surveys; species at risk surveys; site metal analysis; archaeological assessments; land use reviews; and, cultural and socio-economic studies.

Schaft Creek JV has continued to collect environmental data intermittently since 2013 as part of a long-term data collection effort for hydrology, climate, and hydrogeology.

Section 9.0 provides a summary of the environmental setting of biophysical aspects of the Project area, including a summary of baseline study results of the following key topics:

- Climate and Atmospheric Conditions
- Topography and Glacial History

- Geology, Surficial Geology and Soils
- Geohazards
- Metal Leaching (ML) and Acid Rock Drainage (ARD) Hydrology and Watershed Characterizations
- Surface Water
- Groundwater
- Fisheries and Aquatic Habitat
- Terrestrial Ecosystems
- Wildlife and Wildlife Habitat

Environmental management plans for the Project would need to be developed to support environmental assessment and permitting. Environmental management are plans developed to be site-specific and to ensure that necessary measures are identified and implemented in order to protect the environment and comply with environmental legislation. They include legislative requirements, policies, best management practices, committed mitigation measures, and monitoring and reporting commitments. Environmental management plans may include, but are not limited to:

- Surface Water Management and Monitoring Plan
- Groundwater Management and Monitoring Plan
- RSF Management and Monitoring Plan
- TSF Management and Monitoring Plan
- Waste Management Plan
- Dangerous Goods and Hazardous Materials Management Plan
- ML and ARD Management Plan
- Fish and Aquatic Habitat Management Plan
- Noise Management Plan
- Traffic and Access Management Plan
- Sediment and Erosion Control Plan
- Air Quality Management and Monitoring Plan
- Emergency and Spill Response Plan
- Wildlife Management and Monitoring Plan
- Vegetation and Wetland Management Plan
- Hazardous Materials Management Plan
- Archaeological and Cultural Resources Management Plan
- Transportation Management Plan

- Contingency Plans

The Environmental Assessment (EA) process was initiated for the Schaft Creek Project by Copper Fox, entering into the pre application phase in 2006, and subsequently withdrawn upon Schaft Creek JV's request in March 2016.

The Schaft Creek Application Information Requirements / Environmental Impact Statement (AIR/EIS) Guidelines was issued by the Environmental Assessment Office (EAO) and the Canadian Environmental Assessment (CEA) Agency on February 7, 2011. Development of new application information requirements will be required for the Schaft Creek Project for the EA process under the new Environmental Assessment Act (2018). However, much of the information gathered during the development of the Schaft Creek 2011 AIR/EIS guidelines can be used toward the development of the required documents of the EA process. Early in the pre-application stage, Copper Fox held open houses in Telegraph Creek, Dease Lake, Iskut, Terrace, and Stewart, and with the EAO's Schaft Creek advisory working group to identify issues and concerns to be identified and addressed in the EA process documents and Application. The EAO's advisory Schaft Creek Working Group formed in 2011 was comprised of representatives from federal, provincial, local governments, Tahltan Nation, the State of Alaska, and the US federal government.

The federal and provincial authorizations, licences, and permits that are anticipated to be required for the construction and / or operation of the Schaft Creek Project are presented and discussed in Section 20.

1.8 Capital Cost Estimate

Tetra Tech compiled a Capex for this PEA study. The total estimated pre-production capital cost for the design, construction, installation, and commissioning for all facilities and equipment is USD$2,653.2 million. The LOM sustaining capital cost is estimated at USD$848.7 million, inclusive of USD$154 million of closure costs. A summary of Capex is presented in Table 21-1.

This estimate has been prepared in accordance with the Class 5 Cost Estimate standards of the AACE. The accuracy of the estimate is ±30%. A detailed cost breakdown is presented in Table 1-3.

Table 1-3: Pre-production Capital Cost Summary and Comparison

Area Code	Description	Initial Capex (USD$ million)
Direct Costs		
10	Overall Site	137.3
20	Mining	188.8
30	Primary Crushing	50.3
35	Stockpile and Reclaim	41.8
50	Grinding, Flotation and Regrind	500.0
60	TSF	137.2
70	Environmental (included in sustaining capital)	-
80	Site Services and Site Utilities	29.5
85	Ancillary Buildings	117.8
86	Plant Mobile Fleet	6.8
87	Temporary Services	4.3
88	Off-site Infrastructure and Facilities	82.2
	Subtotal – Direct Costs	**1,296.1**
Indirect Costs		
90	Project Indirect Costs (including Owner's costs)	771.0
	Subtotal – Direct + Indirect Costs	**2,067.1**
99	Contingencies (~25%) + Provisions	586.1
	Total	**2,653.2**

Note: Figures are rounded to hundred thousands.

This estimate includes direct field costs required to execute the Project, plus indirect costs associated with design, construction, and commissioning. This estimate is based on pricing as of Q4 2020, with no allowances for inflation or escalation. All currency in this Capex is expressed in United States dollars, unless otherwise noted.

1.9 Operating Cost Estimate

The operating cost estimate for the Project consists of mining, processing, general and administration (G&A), surface services, gate services (gate control, internal road maintenance and concentrate hauling), and tailings and site water management costs. The LOM average operating costs are summarized in Table 1-4. The total LOM average operating cost is estimated to be USD$8.66/t mill feed processed. The operating cost estimate is expressed in US dollars, unless specified.

Table 1-4: LOM Average Operating Cost Summary*

Function	Operating Cost (USD$/t processed)
Mining	3.11
Processing	4.08
Tailings and Site Water Management	0.11
Gate Control and Road Services**	0.32
Surface Services	0.25
G&A	0.79
Total Cost	**8.66**

Note: *LOM average operating, which is slightly different from the operating cost at the full process rate of 133,000 t/d.
**Gate control and road services include concentrate hauling from the site to the gate.

1.10 Economic Analysis

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

The pre-tax and post-tax project economic analysis of the Project is based on payable metal and was prepared on a 100% basis using revenues and costs projected into the future on an annual basis and then discounted using mid year discounting at a rate of 8% per annum to yield the NPV and IRR. Net smelter return, capital, operating, sustaining and closure costs, net proceeds interests (NPI) payments, BC mineral tax, and federal and provincial income taxes are included in the financial analysis. Metal prices are based on long term consensus metal prices.

For the 21-year mine life and 1.03 Bt mill feed tonnage and the metal prices and foreign exchange rate shown in Table 1-5 (base case), the following financial parameters were calculated:

- Pre-tax IRR of 15.2%
- Post-tax IRR of 12.9%
- Pre-tax NPV of USD$1,383 million at an 8% discount rate
- Post-tax NPV of USD$842 million at an 8% discount rate
- 4.4-year payback period for pre-tax estimate and 4.8-year payback period for post-tax estimate

Table 1-5: Metal Pricing and USD/CAD Exchange Rate Inputs

Copper	USD$/lb.	3.25
Gold	USD$/oz	1,500
Molybdenum	USD$/lb	10.00
Silver	USD$/oz	20.00
CAD/USD Exchange	--	1.30

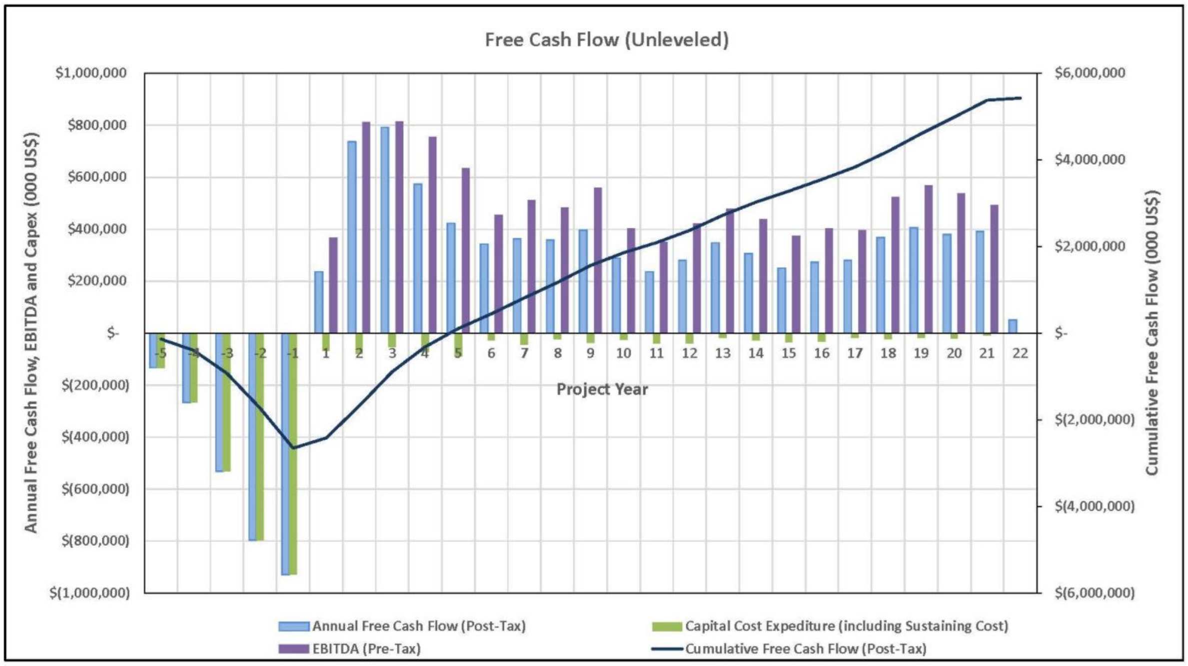
Table 1-6 presents the post-tax financial analysis summary.

Table 1-6: Financial Analysis Summary (Post-Tax)

Description	Value	Units
Financial Analysis Summary		
LOM	21	years
Tonnes Mined Including Waste Rock	2,073,623	kt
Tonnes Processed	1,030,207	kt
Annual Tonnage Processed	49,057	kt
Tonnes Concentrate Produced (Dry Mass) Copper	8,091	kt
Copper Recovered to Concentrate	4,994,616	klb
Gold Recovered to Concentrate	3,695	koz
Silver Recovered to Concentrate	16,412	koz
Tonnes Concentrate Produced (Dry Mass) Molybdenum	205	kt
Molybdenum Recovered To Concentrate	226,457	klb
Net Revenue from Sales	$21,250	USD$ million
LOM Unit Cash Costs		
Before By-Product Credits	2.56	USD$/lb Copper
After By-Product Credits	1.00	USD$/lb Copper
All-in Sustaining Costs	1.18	USD$/lb Copper
Cash Flow		
Post-Tax Operating Cash Flow	$5,395	USD$ millions
Post-Tax NPV at a Discount Rate of 8%	$842	USD$ millions
Post-Tax IRR	12.9%	--
Payback Years	4.8	years

Unlevered and undiscounted free cash flow (FCF) projection are shown in Figure 1-6.

Figure 1-6: PEA Post-Tax Annual and Cumulative FCFs, EBITDA, and Capex



Note: EBITDA = Earnings Before Interest, Taxes, Depreciation, and Amortization

Federal, Provincial and BC Mineral Tax payables based on the PEA financial model are calculated and shown in Table 1-7. The BC Mineral Tax (Provincial Resource Tax) is deductible from Federal and Provincial taxes payable.

Table 1-7: Estimated Taxes Payable

Tax Component	LOM Amount (CAD$M)
Corporate Tax (Federal)	1,432
Corporate Tax (Provincial)	1,145
BC Mineral Tax	1,198
Total Taxes	3,775

Sensitivity analysis for post-tax financial parameters for both NPV at a discount rate of 8% and IRR were conducted. The project post-tax NPV at a discount rate of 8% is more sensitive to copper pricing and USD/CAD exchange rate, followed by changes in gold pricing and operating and capital costs. The project post-tax IRR shows a similar sensitivity pattern to these variables, except that it is most sensitive to USD/CAD exchange rate, followed by copper pricing.

1.11 Conclusions and Recommendations

The Schaft Creek Project is considered to be technically and economically viable based on the results of the work presented in this Technical Report. It is recommended to advance the Project to the next stage. Section 26.0 outlines detailed recommendations for the Schaft Creek Project.

2.0 INTRODUCTION

In 2020, Copper Fox commissioned a team of engineering consultants to complete this PEA, in accordance with NI 43-101 Standards of Disclosure for Mineral Projects.

Components of this PEA were completed by the following consultants:

- Tetra Tech Canada Inc. (Tetra Tech): overall project management, mining methods, metallurgical testing, mineral processing and recovery methods, project infrastructure, capital and operating cost estimates, and economic analysis.
- Red Pennant Communications Corp. (Red Pennant): project description and location, accessibility, history, geological setting, deposit types, exploration, drilling, Mineral Resource estimate and adjacent properties.
- Knight Piésold Ltd. (Knight Piésold): tailings and waste rock management and power transmission (including capital costs).
- Greenwood Environmental Ltd. (Greenwood) – environmental studies, permitting, and social or community impact.
- Ruskin Construction Ltd./ Allnorth Consultants Ltd. (Ruskin/Allnorth) – Galore Creek Access Road and More Canyon Bridge.
- McElhanney Consulting Services Ltd. (MlElhanney) – Mess Creek Access Road.

A summary of the consultants responsible for each section of this report is detailed in Table 2-1.

2.1 Qualified Persons

The name of the Qualified Persons (QPs) of this report and their QP certificates are included in Section 28.0.

The following QPs conducted a site visit of the Property:

- Mr. Hassan Ghaffari, P.Eng. of Tetra Tech, visited the site on September 22, 2010 and conducted a general project site overview in the proposed infrastructure areas.
- Mr. John Huang, Ph.D., P.Eng. of Tetra Tech, visited the site on August 9, 2010 and conducted an overview of the proposed processing plant site.
- Mr. Michael O'Brien, P.Geo. of Red Pennant, visited the Property on October 30, 2020 and reviewed drill cores and the general layout of camp and topography.
- Mr. Daniel Friedman, P.Eng. of Knight Piésold, visited the site from July 28 to August 11, 2008 and conducted an overview of the proposed general project and TSF site.
- Mr. Brendon Masson, P.Eng. of McElhanney, visited the site on December 10, 2010 and conducted a general project site overview in the proposed access road areas.

Table 2-1: Summary of Report Sections and Consultants

Report Section	Company	QP
1.0 Summary	All	Sign-off by Section
2.0 Introduction	Tetra Tech	Hassan Ghaffari, M.A.Sc., P.Eng.
3.0 Reliance on Other Experts	Tetra Tech	Hassan Ghaffari, M.A.Sc., P.Eng.
4.0 Property Description and Location	Red Pennant	Mike O'Brien, P.Geo.
5.0 Accessibility, Climate, Local Resources, Infrastructure, and Physiography	Red Pennant	Mike O'Brien, P.Geo.
6.0 History	Red Pennant	Mike O'Brien, P.Geo.
7.0 Geological Setting and Mineralization	Red Pennant	Mike O'Brien, P.Geo.
8.0 Deposit Types	Red Pennant	Mike O'Brien, P.Geo.
9.0 Exploration	Red Pennant	Mike O'Brien, P.Geo.
10.0 Drilling	Red Pennant	Mike O'Brien, P.Geo.
11.0 Sample Preparation, Analyses, and Security	Red Pennant	Mike O'Brien, P.Geo.
12.0 Data Verification	Red Pennant	Mike O'Brien, P.Geo.
13.0 Mineral Processing and Metallurgical Testing	Tetra Tech	John Huang, Ph.D., P.Eng.
14.0 Mineral Resource Estimates	Red Pennant	Mike O'Brien, P.Geo.
15.0 Mineral Reserve Estimates	Tetra Tech	Sabry Abdel Hafez, Ph.D., P.Eng.
16.0 Mining Methods	Tetra Tech	Sabry Abdel Hafez, Ph.D., P.Eng.
17.0 Recovery Methods	Tetra Tech	John Huang, Ph.D., P.Eng.
18.0 Project Infrastructure	-	-
18.1 Overview	Tetra Tech	Hassan Ghaffari, M.A.Sc., P.Eng.
18.2 Major Layout Modifications Since FS	Tetra Tech	Hassan Ghaffari, M.A.Sc., P.Eng.
18.3 Site Layout	Tetra Tech	Hassan Ghaffari, M.A.Sc., P.Eng.
18.4 Access Roads	McElhanney	Brendon Masson, P. Eng.
18.5 Bob Quinn Lake Airport Upgrade	Tetra Tech	Hassan Ghaffari, M.A.Sc., P.Eng.
18.6 Water Supply and Distribution	Tetra Tech	Hassan Ghaffari, M.A.Sc., P.Eng.
18.7 Waste Disposal	Tetra Tech	Hassan Ghaffari, M.A.Sc., P.Eng.
18.8 Tailings Storage Facility and Tailings Management	Knight Piésold	Daniel Friedman, P.Eng.
18.9 Plant Ancillary Facilities	Tetra Tech	Hassan Ghaffari, M.A.Sc., P.Eng.
18.10.1 Power Supply	Knight Piésold	Daniel Friedman, P.Eng.
18.10.2 Power Distribution	Tetra Tech	Hassan Ghaffari, M.A.Sc., P.Eng.
18.11 Communications	Tetra Tech	Hassan Ghaffari, M.A.Sc., P.Eng.
19.0 Market Studies and Contracts	Tetra Tech	John Huang, Ph.D., P.Eng.

table continues…

Report Section	Company	QP
20.0 Environmental Studies, Permitting and Social or Community Impact	-	-
20.1 Environmental Setting and Studies	Greenwood	Hassan Ghaffari, M.A.Sc., P.Eng.
20.2 Socio-Economic and Cultural Setting	Greenwood	Hassan Ghaffari, M.A.Sc., P.Eng.
20.3 Human Health Setting	Greenwood	Hassan Ghaffari, M.A.Sc., P.Eng.
20.4 Environmental Management and Monitoring	-	-
20.4.1 Environmental Management Plans	Greenwood	Hassan Ghaffari, M.A.Sc., P.Eng.
20.4.2 Water Management	Knight Piésold	Daniel Friedman, P.Eng.
20.4.3 Waste Management	-	-
20.4.3.1 Waste Management (Tailings, Waste Rock, and Overburden)	Knight Piésold	Daniel Friedman, P.Eng.
20.4.3.2 Hazardous Waste Management	Greenwood	Hassan Ghaffari, M.A.Sc., P.Eng.
20.4.3.3 Non-hazardous Waste Management	Greenwood	Hassan Ghaffari, M.A.Sc., P.Eng.
20.5 Closure and Reclamation	Knight Piésold	Daniel Friedman, P.Eng.
20.6 Environmental Assessment and Permitting	Greenwood	Hassan Ghaffari, M.A.Sc., P.Eng.
20.7 Environmental and Socio-cultural Considerations	Greenwood	Hassan Ghaffari, M.A.Sc., P.Eng.
21.0 Capital and Operating Costs	-	-
21.1 Capital Cost Estimate	Tetra Tech / Knight Piésold	Hassan Ghaffari, M.A.Sc., P.Eng. / Daniel Friedman, P.Eng./ Brendon Masson, P. Eng.
21.2 Operating Cost Estimate	-	-
21.2.1 Summary	Tetra Tech	John Huang, Ph.D., P.Eng.
21.2.2 Mining Operating Cost	Tetra Tech	Hassan Ghaffari, M.A.Sc., P.Eng.
21.2.3 Mill Operating Cost	Tetra Tech	John Huang, Ph.D., P.Eng.
21.2.4 General & Administrative	Tetra Tech	John Huang, Ph.D., P.Eng.
21.2.5 Surface Services	Tetra Tech	John Huang, Ph.D., P.Eng.
21.2.6 Tailings and Site Water Management	Knight Piésold	Daniel Friedman, P.Eng.
22.0 Economic Analysis	Tetra Tech	John Huang, Ph.D., P.Eng.
23.0 Adjacent Properties	Red Pennant	Mike O'Brien, P.Geo.
24.0 Other Relevant Data and Information	Tetra Tech	Hassan Ghaffari, M.A.Sc, P.Eng.
25.0 Interpretation and Conclusions	All	Sign-off by Section
26.0 Recommendations	All	Sign-off by Section
27.0 References	All	Sign-off by Section

3.0 RELIANCE ON OTHER EXPERTS

Michael O'Brien, P.Geo. of Red Pennant relied on Copper Fox Metals Inc. for matters relating to mineral tenure and mining rights permits, surface rights, royalties, agreements, and encumbrances relevant to this report.

Hassan Ghaffari, P. Eng. of Tetra Tech, relied on Shane Uren, R.P.Bio. of Greenwood, for matters related to environmental, permitting and social or community impact detailed in Section 20.0 of this report.

Jianhui (John) Huang, P.Eng. of Tetra Tech relied on Copper Fox Metals Inc.'s financial consultant concerning tax matters relevant to this PEA and detailed in Section 22.0. The reliance is based on a letter to Copper Fox titled "*Assistance with review of the income and mining tax portions of the economic analysis prepared by Tetra Tech Inc. ("Tetra Tech") in connection with the 2021 Preliminary Economic Assessment Report (the "Report") on Copper Fox Metals Inc.'s ("Copper Fox") Schaft Creek Joint Venture project with Teck Resource Limited (the "Project")*", dated October 7, 2021, in connection with the Schaft Creek PEA, NI 43-101 Technical Report.

4.0 PROPERTY DESCRIPTION AND LOCATION

The Project is located on the eastern side of the Coast Mountains in northwestern British Columbia, within the Cassiar/Liard Mining Division. The Property is approximately 120 km southwest of Dease Lake, and 375 km northwest of Smithers (Figure 4-1). The closest population center is Telegraph Creek, located approximately 61 km to the north. Highway 37 is 45 km east of the Property. The deposit is centred at approximately 379850mE, 6360080mN (UTM NAD83, zone 9). An exploration camp is located 1 km to the southwest of the deposit, at 378715mE, 6358605mN. Drill core is stored on site at the core logging facility next to the camp.

The Schaft Creek tenure consists of 181 mineral claims. The tenure is comprised of a north block and a south block, with three isolated claims to the northeast; in total, the Property encompasses 55,779.56 ha (Figure 4-2). The Schaft Creek tenements are presented in Table 4-1.

Figure 4-1: Location of the Schaft Creek Project

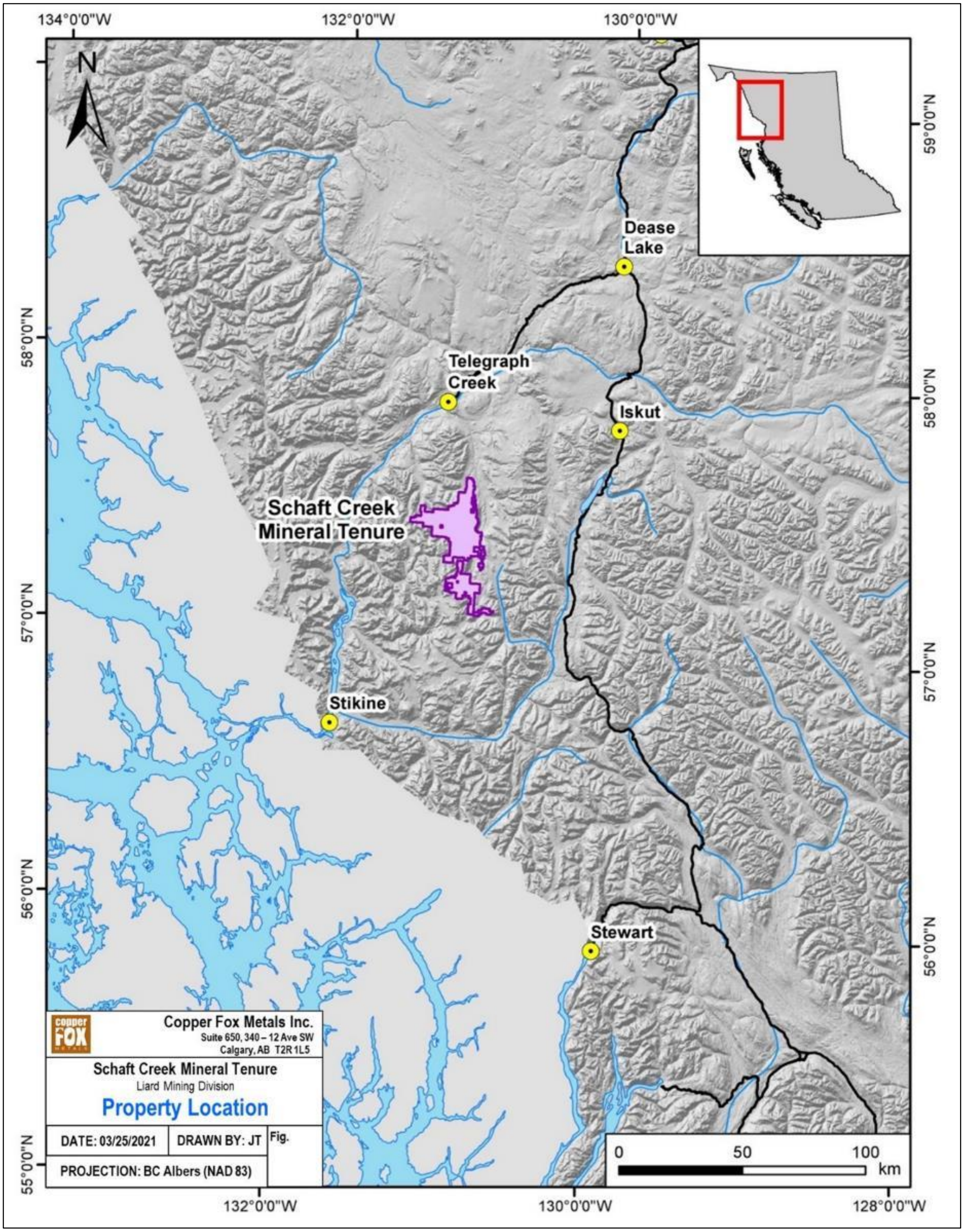
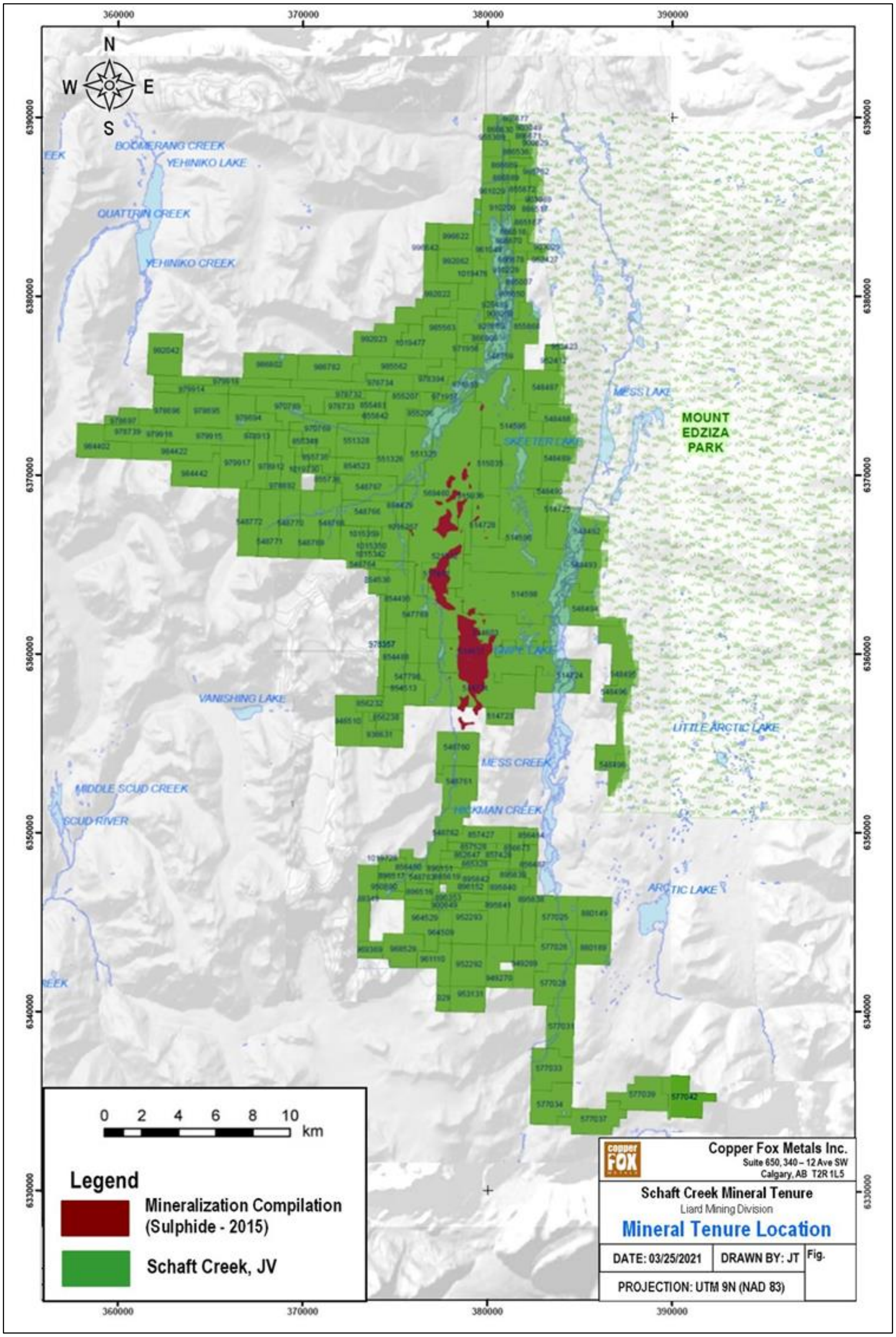


Figure 4-2: Schaft Creek Property Mineral Tenure



4.1 Project Ownership

4.1.1 Ownership History

The initial claims were staked in 1957 by the BIK Syndicate, a consortium of companies that incorporated as Liard Copper Mines Ltd. (Liard) in 1966. These initial claims, and subsequent additions, were staked over the Liard Zone (the Liard Property). In 1968, Liard entered into an option agreement with Hecla Mining Company (Hecla), under which Hecla earned a 70% interest in the Liard Property. Liard retained a 30% carried NPI in the Liard Property. Subsequently, Teck acquired a 78% interest in Liard, which represented a 23.4% interest in the Liard Property.

Paramount Mining Ltd. (Paramount) staked claims north of the Liard Property (the Paramount Claims). In 1969, Paramount entered into an option agreement with Hecla for the Paramount Claims. Hecla later terminated their option on the Paramount Claims, and these claims were allowed to lapse. Subsequently, Teck acquired tenure over the area that comprised the Paramount Claims.

In 1978, Hecla assigned its 70% ownership in the Liard Property to Teck, reserving a 5% NPI from this 70% ownership. This yielded for Hecla an effective 3.5% NPI on the property, payable only after Teck has recovered certain expenditures. In February 2005, Hecla assigned this 3.5% NPI to International Royalty Corporation. In 2010, International Royalty Corporation was acquired by Royal Gold Inc, at which time the 3.5% NPI was transferred to Royal Gold Inc.

In January 2002, Teck signed an option agreement with Guillermo Salazar (Salazar). The agreement allowed Salazar to earn Teck's 70% direct participating interest in the Liard Property by incurring certain exploration expenditures, as well as Teck's 23.4% indirect carried interest in the Liard Property (through its 78% shareholding of Liard) by completing a positive bankable feasibility study (FS). In addition to the Liard Property, the option agreement also included the Paramount Claims. Teck retained a back-in right, whereby it could acquire an interest of up to 75%.

In February 2003, Salazar assigned the option agreement to 955528 Alberta Ltd., which amalgamated with Copper Fox in 2004. From 2005 to 2012 Copper Fox conducted work on the Project, filing an FS in 2013.

In July 2013, Teck and Copper Fox entered into an agreement for the formation of the Schaft Creek JV whereby Teck holds a 75% interest and Copper Fox holds a 25% interest in the Schaft Creek property. Teck's 78% interest in Liard was included in the formation of the Schaft Creek JV (see Section 4.5).

In December 2015, the Schaft Creek JV entered into an agreement to acquire an additional 7.4% of the issued and outstanding shares of Liard. As a result, the Schaft Creek JV has a total ownership of approximately 85.5% of the issued and outstanding shares of Liard.

4.1.2 Current Ownership

The Project is managed through the Schaft Creek JV. Teck is the operator and holds a 75% interest. Copper Fox holds the remaining 25% interest. The mineral claims listed in Table 4-1 are held by Teck on behalf of the Schaft Creek JV.

4.1.3 Mineral Tenure

The Schaft Creek JV consists of 181 mineral claims (55,779.56 ha). The tenure consists of a north block and a south block, with three isolated claims to the northeast (Figure 4-3). A listing of the mineral claims and royalty agreement applicable to the claims is provided in Table 4-1.

Figure 4-3: Mineral Tenure Summary Plan

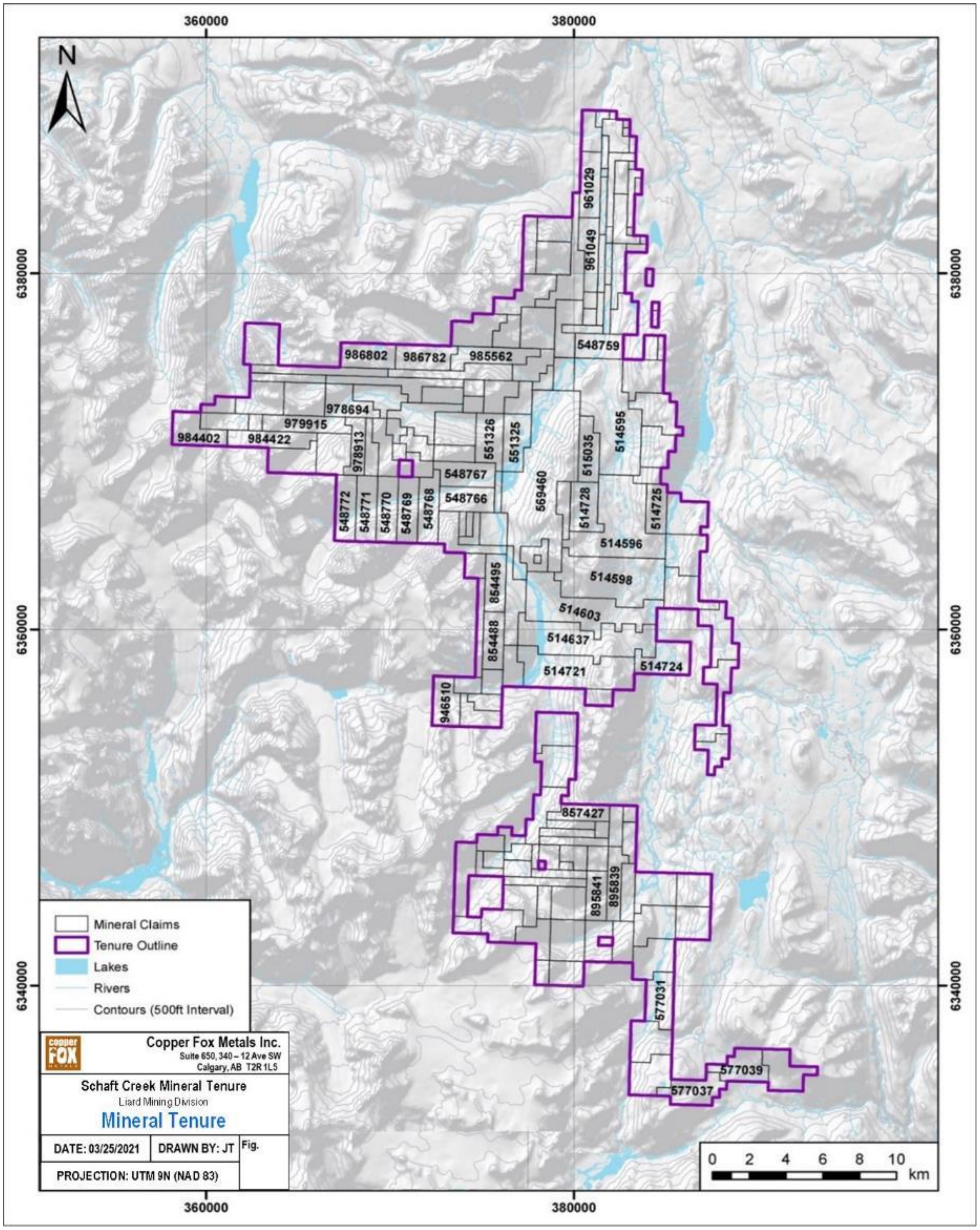


Table 4-1: Schaft Creek JV Mineral Claims Table

ID	Name	Parties	Type	Status	Grant Date	Expiry Date	Official Area (ha)	Royalty
514595	CL 514595	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	6/16/2005	2/13/2026	1,653.04	
514596	CL 514596	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	6/16/2005	2/13/2026	1,550.96	
514598	CL 514598	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	6/16/2005	2/13/2026	1,412.62	
514603	CL 514603	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	6/16/2005	2/13/2026	1,291.06	
514637	CL 514637	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	6/16/2005	2/13/2026	1,256.71	
514721	CL 514721	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	6/16/2005	2/13/2026	1,169.95	
514723	CL 514723	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	6/16/2005	2/13/2026	139.745	
514724	CL 514724	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	6/16/2005	2/13/2026	471.387	
514725	CL 514725	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	6/16/2005	2/13/2026	313.607	
514728	CL 514728	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	6/16/2005	2/13/2026	465.589	
515035	CL 515035	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	6/16/2005	2/13/2026	383.005	
515036	CL 515036	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	6/16/2005	2/13/2026	191.645	
517462	CL 517462	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	7/12/2005	2/13/2026	17.436	Kreft/Greig
521312	Schaft 1	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	10/18/2005	2/13/2026	191.784	Pembrook
547789	CL 547789	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	12/21/2006	2/13/2026	418.7	
547798	CL 547798	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	12/21/2006	2/13/2026	227	
548487	Block B1	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	1/2/2007	2/13/2026	434.782	

table continues…

ID	Name	Parties	Type	Status	Grant Date	Expiry Date	Official Area (ha)	Royalty
548488	Block B2	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	1/2/2007	2/13/2026	434.989	
548489	Block B3	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	1/2/2007	2/13/2026	365.568	
548490	Block B4	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	1/2/2007	2/13/2026	121.904	
548492	Block C1	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	1/2/2007	2/13/2026	435.989	
548493	Block C2	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	1/2/2007	2/13/2026	435.829	
548494	Block C3	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	1/2/2007	2/13/2026	436.064	
548495	Block C4	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	1/2/2007	2/13/2026	436.309	
548496	Block C5	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	1/2/2007	2/13/2026	436.695	
548498	Block C6	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	1/2/2007	2/13/2026	227.243	
548759	Area A	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	1/2/2007	2/13/2026	365.065	
548760	Area C1	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	1/2/2007	2/13/2026	436.903	
548761	Area C2	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	1/2/2007	2/13/2026	437.115	
548762	Area C3	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	1/2/2007	2/13/2026	367.411	
548763	Area C4	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	1/2/2007	2/13/2026	122.542	
548764	Area B1	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	1/5/2007	2/13/2026	366.043	
548766	Area B2	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	1/5/2007	2/13/2026	418.11	
548767	Area B3	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	1/5/2007	2/13/2026	435.38	
548768	Area B4	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	1/5/2007	2/13/2026	435.6	
548769	Area B5	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	1/5/2007	2/13/2026	418.19	

table continues…

ID	Name	Parties	Type	Status	Grant Date	Expiry Date	Official Area (ha)	Royalty
548770	Area B6	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	1/5/2007	2/13/2026	418.19	
548771	Area B7	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	1/5/2007	2/13/2026	418.19	
548772	Area B8	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	1/5/2007	2/13/2026	418.19	
551325	Area D1	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	2/6/2006	2/13/2026	435.18	
551326	Area D2	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	2/6/2006	2/13/2026	435.17	
551328	Area D3	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	2/6/2006	2/13/2026	417.71	
569460	Greater Kopper	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	11/5/2007	2/13/2026	2,769.10	Kreft/Greig
577025	SC South 1	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	2/23/2008	2/13/2026	437.8319	
577026	SC South 2	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	2/23/2008	2/13/2026	438.0366	
577028	SC South 3	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	2/23/2008	2/13/2026	438.2416	
577031	SC South 4	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	2/23/2008	2/13/2026	438.4862	
577033	SC South 5	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	2/23/2008	2/13/2026	438.7322	
577034	SC South 6	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	2/23/2008	2/13/2026	438.9363	
577037	SC South 7	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	2/23/2008	2/13/2026	439.0198	
577039	SC South 8	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	2/23/2008	2/13/2026	438.876	
577042	SC South 9	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	2/23/2008	2/13/2026	438.8966	
854488	Silver Fox 86	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/13/2011	2/13/2026	366.5575	Marko/Mott
854495	Silver Fox 87	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/13/2011	2/13/2026	366.2694	Marko/Mott
854513	Silver Fox 89	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/14/2011	2/13/2026	157.1843	Marko/Mott

table continues…

ID	Name	Parties	Type	Status	Grant Date	Expiry Date	Official Area (ha)	Royalty
854523	White Rabbit 90	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/14/2011	2/13/2026	208.9252	Marko/Mott
854536	Silver Fox 91	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/14/2011	2/13/2026	156.9374	Marko/Mott
855206	Ptarmigan 93	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/18/2011	2/13/2026	208.7684	Marko/Mott
855207	Ptarmigan 95	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/18/2011	2/13/2026	278.339	Marko/Mott
855348	White Rabbit 92	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/21/2011	2/13/2026	104.4313	Marko/Mott
855461	Ptarmigan 97	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/24/2011	2/13/2026	104.3678	Marko/Mott
855735	White Rabbit 101	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/26/2011	2/13/2026	191.496	Marko/Mott
855736	White Rabbit 102	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/26/2011	2/13/2026	139.3092	Marko/Mott
855842	Ptarmigan 103	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/27/2011	2/13/2026	104.3915	Marko/Mott
855868	Tern 120	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/30/2013	2/13/2026	295.4047	Marko/Mott
855872	Tern 103	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/30/2013	2/13/2026	138.7507	Marko/Mott
856232	Silver Fox 118	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	6/3/2011	2/13/2026	139.7259	Marko/Mott
856238	Silver Fox 119	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	6/3/2011	2/13/2026	157.23	Marko/Mott
856450	Elk 151	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	6/8/2011	2/13/2026	105.0158	Marko/Mott
856464	Elk 152	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	6/8/2011	2/13/2026	69.983	Marko/Mott
856487	Elk152	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	6/9/2011	2/13/2026	157.52	Marko/Mott
856673	Elk 153	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	6/10/2011	2/13/2026	174.9874	Marko/Mott
857427	Elk 154	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	6/21/2011	2/13/2026	279.9349	Marko/Mott

table continues…

ID	Name	Parties	Type	Status	Grant Date	Expiry Date	Official Area (ha)	Royalty
857428	Elk 155	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	6/21/2011	2/13/2026	69.9989	Marko/Mott
857528	Elk 156	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	6/22/2011	2/13/2026	122.4914	Marko/Mott
862647	Elk 158	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	7/4/2011	2/13/2026	140.0061	Marko/Mott
865007	Tern 125	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	7/7/2011	2/13/2026	243.131	Marko/Mott
865167	Tern 127	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	7/8/2011	2/13/2026	242.9604	Marko/Mott
865328	Elk 166	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	7/9/2011	2/13/2026	175.0273	Marko/Mott
865619	Elk 167	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	7/11/2011	2/13/2026	140.0507	Marko/Mott
866050	Tern 128	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	7/13/2011	2/13/2026	104.2511	Marko/Mott
866517	Tern 130	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	7/18/2011	2/13/2026	138.7842	Marko/Mott
866518	Tern 131	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	7/18/2011	2/13/2026	208.137	Marko/Mott
866536	Tern 132	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	7/18/2011	2/13/2026	208.0058	Marko/Mott
866630	Tern 131	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	7/19/2011	2/13/2026	51.9883	Marko/Mott
866669	Tern 133	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	7/20/2011	2/13/2026	69.3512	Marko/Mott
866670	Tern 134	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	7/20/2011	2/13/2026	34.715	Marko/Mott
866671	Tern 135	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	7/20/2011	2/13/2026	17.3328	Marko/Mott
866677	Tern 135	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	7/20/2011	2/13/2026	17.3287	Marko/Mott
866678	Tern 136	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	7/20/2011	2/13/2026	86.822	Marko/Mott
866889	Tern 137	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	7/20/2011	2/13/2026	17.3428	Marko/Mott
866909	Juskatla Resources 2	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	7/20/2011	2/13/2026	104.2799	

table continues…

ID	Name	Parties	Type	Status	Grant Date	Expiry Date	Official Area (ha)	Royalty
880149	Bonanza	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	8/3/2011	2/13/2026	350.2622	
880189	Bonanza1	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	8/3/2011	2/13/2026	350.4197	
884429	Gold Bear	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	8/7/2011	2/13/2026	87.0967	Marko/Mott
895838	Eagle 800	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	9/1/2011	2/13/2026	245.1966	
895839	Eagle 801	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	9/1/2011	2/13/2026	332.7258	
895840	Eagle 802	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	9/1/2011	2/13/2026	157.5583	
895841	Eagle 803	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	9/1/2011	2/13/2026	315.2703	
895842	Eagle 804	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	9/1/2011	2/13/2026	175.0619	
896151	Eagle 805	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	9/6/2011	2/13/2026	52.5198	
896152	Eagle 806	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	9/6/2011	2/13/2026	35.0163	
896353	Eagle 807	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	9/9/2011	2/13/2026	140.0808	
896516	Eagle 808	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	9/11/2011	2/13/2026	140.0725	
896517	Eagle 809	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	9/11/2011	2/13/2026	105.0451	
900609	Juskatla Resources 3	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	9/25/2011	2/13/2026	17.3566	
900629	Juskatla Resource 4	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	9/25/2011	2/13/2026	34.6717	
900649	Eagle 810	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	9/25/2011	2/13/2026	210.1447	
903029	Juskatla Resources 5	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	9/28/2011	2/13/2026	17.3584	

table continues…

ID	Name	Parties	Type	Status	Grant Date	Expiry Date	Official Area (ha)	Royalty
903049	Juskatla Resources 6	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	9/28/2011	2/13/2026	17.3308	
903069	Juskatla Resources 7	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	9/28/2011	2/13/2026	34.6917	
908069	Tern Around	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	10/8/2011	2/13/2026	69.5009	
910209	Tern Around	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	10/12/ 2011	2/13/2026	121.455	
910229	Tern Around	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	10/12/ 2011	2/13/2026	121.5508	
927669	Tern Left	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	11/1/2011	2/13/2026	69.5086	
928489	Tern West	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	11/8/2011	2/13/2026	69.493	
936631	Eagle 815	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	12/7/2011	2/13/2026	262.1002	
946510	Eagle 816	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	2/6/2012	2/13/2026	384.3474	
949269	Eagle 812	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	2/13/2012	2/13/2026	262.8877	
949270	Eagle 811	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	2/13/2012	2/13/2026	315.4651	
950890	Eagle 814	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	2/20/2012	2/13/2026	105.0552	
952292	Eagle 813	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	2/23/2012	2/13/2026	438.1459	
952293	Eagle 817	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	2/23/2012	2/13/2026	350.3396	
952412	Retern100	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	2/24/2012	2/13/2026	104.304	
952423	Retern101	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	2/24/2012	2/13/2026	52.1522	
952427	Retern 102	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	2/24/2012	2/13/2026	52.0814	

table continues…

ID	Name	Parties	Type	Status	Grant Date	Expiry Date	Official Area (ha)	Royalty
953131	Eagle 818	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	2/27/2012	2/13/2026	263.0055	
955309	Tern North	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	3/4/2012	2/13/2026	225.3319	
961029	North Tern 2	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	3/13/2012	2/13/2026	416.2977	
961049	North Tern 3	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	3/13/2012	2/13/2026	381.9542	
961110	Silver Eagle 900	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	3/13/2012	2/13/2026	280.4076	
964509	Silver Eagle 902	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	3/16/2012	2/13/2026	140.1358	
964529	Silver Eagle 903	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	3/16/2012	2/13/2026	332.823	
965029	Silver Eagle 901	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	3/17/2012	2/13/2026	105.2024	
968529	Silver Eagle 904	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	3/21/2012	2/13/2026	367.9731	
969349	Silver Eagle 905	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	3/21/2012	2/13/2026	385.2511	
969369	Silver Eagle 906	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	3/21/2012	2/13/2026	140.183	
970769	Silver Rabbit	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	3/24/2012	2/13/2026	435.0489	
970789	Silver Rabbit 2	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	3/24/2012	2/13/2026	347.9524	
971953	Tern South	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	3/26/2012	2/13/2026	208.6899	
971956	Tern South 2	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	3/26/2012	2/13/2026	382.3436	
971957	Tern South 3	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	3/26/2012	2/13/2026	104.355	
978394	South Tern 4	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	4/6/2012	2/13/2026	260.8388	
978694	Crown 500	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	4/9/2012	2/13/2026	400.1872	
978695	Crown 501	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	4/9/2012	2/13/2026	417.5738	

table continues…

ID	Name	Parties	Type	Status	Grant Date	Expiry Date	Official Area (ha)	Royalty
978696	Crown 502	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	4/9/2012	2/13/2026	417.5846	
978697	Crown 503	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	4/9/2012	2/13/2026	139.2108	
978732	Crown 504	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	4/10/2012	2/13/2026	434.8568	
978733	Crown 505	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	4/10/2012	2/13/2026	208.7595	
978734	Crown 506	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	4/10/2012	2/13/2026	434.779	
978739	Crown 507	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	4/10/2012	2/13/2026	243.6704	
978892	Crown 507	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	4/10/2012	2/13/2026	295.9702	
978912	Crown 508	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	4/10/2012	2/13/2026	191.5177	
978913	Crown 509	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	4/10/2012	2/13/2026	278.5544	
979914	Crown 510	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	4/12/2012	2/13/2026	191.3341	
979915	Crown 511	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	4/12/2012	2/13/2026	435.1219	
979916	Crown 512	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	4/12/2012	2/13/2026	69.6198	
979917	Crown 513	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	4/12/2012	2/13/2026	417.8764	
979918	Crown 514	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	4/12/2012	2/13/2026	347.8393	
984402	Crown 515	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/8/2012	2/13/2026	417.7842	
984422	Crown 516	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/8/2012	2/13/2026	435.2212	
984442	Crown 517	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/8/2012	2/13/2026	383.0989	
985562	Crown 519	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/10/2012	2/13/2026	434.6695	
985563	Crown 520	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/10/2012	2/13/2026	434.4922	

table continues…

ID	Name	Parties	Type	Status	Grant Date	Expiry Date	Official Area (ha)	Royalty
986762	Retern 105	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/16/2012	2/13/2026	17.3408	
986782	Crown 521	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/16/2012	2/13/2026	434.6827	
986802	Crown 522	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/16/2012	2/13/2026	417.2993	
992022	Panda 1	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/31/2012	2/13/2026	434.2899	
992023	Panda 2	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/31/2012	2/13/2026	417.1348	
992042	Panda 3	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/31/2012	2/13/2026	434.6455	
992062	Panda 4	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/31/2012	2/13/2026	347.2546	
996622	Panda 5	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	6/12/2012	2/13/2026	260.3358	
996642	Panda 6	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	6/12/2012	2/13/2026	243.0354	
1015257	CL 1015257	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	12/12/2012	2/13/2026	278.8546	
1015342	CL 1015342	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	12/17/2012	2/13/2026	104.5769	
1015350	CL 1015350	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	12/17/2012	2/13/2026	52.2821	
1015359	CL 1015359	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	12/17/2012	2/13/2026	52.2821	
1019476	Tern 2 U	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/13/2013	2/13/2026	503.5922	
1019477	Tern 2 U	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/13/2013	2/13/2026	225.9553	
1019728	CL-1019728	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/23/2013	2/13/2026	140.0324	
1019730	CL-1019730	Teck Resources Limited (100%)	Mineral Claim (BC)	Active	5/23/2013	2/13/2026	69.642	

The Schaft Creek copper–gold–molybdenum–silver deposit is located within the southern boundary of claim 514603 and the northern boundary of claim 514637.

Pertinent terms and conditions of the Schaft Creek JV are discussed in Section 4.6.

4.2 Surface Rights

The surface is Crown land. Overlapping surface interests currently known include:

- Trapline TR062IT006
- Licence of Occupation #6406985 for commercial recreation
- Outfitting area held by Heidi Gutfruch.

The Property is within the Tahltan territory

Mining will require the exclusive use of the surface; an Industrial Surface Lease would be required to overlap the claims and provide exclusivity outside any Mines Act Permit boundary. Powerlines, upgraded road access, and other ancillary land use will require appropriate Land Act dispositions (Licences of Occupation, Statutory Rights of Way, etc.). Standing timber outside the existing cut blocks will require an Occupant Licence to Cut if removal becomes necessary.

4.3 Water Rights

The water rights belong to the Crown; the use of water for mining will require the issuance of a Section 10 *Mines Act* Permit to exempt the operator from the *Water Sustainability Act* requirements.

Water use outside of exploration drilling will require a *Water Sustainability Act* approval until a Section 10 permit is issued.

4.4 Royalties and Encumbrances

4.4.1 Schaft Creek Joint Venture

4.4.1.1 Royal Gold and Conversion Royalties

Pursuant to the 2002 Option Agreement with Teck, Copper Fox acquired a 100% working interest in the Project subject to a 3.5% net profits interest held by Royal Gold Inc., a 30% carried interest held by Liard, and an earn-back option held by Teck.

Teck exercised the earn-back option on July 15, 2013, and has an unconditional 75% direct interest in the Project. Copper Fox retains a 25% interest. Teck is operator of the Schaft Creek JV, formed on 15 July 2013. The Schaft Creek JV agreement includes a provision that if any party's Project interest falls below 20%, that interest will become a "Conversion Royalty" of 15% net profits interest, and the party will retain no other interest in the Project.

4.4.1.2 Areas of Interest

There are two areas of interest in the Schaft Creek JV, as indicated in Figure 4-4.

The Teck / Copper Fox area of interest is a 2 km zone around the original 2002 tenure holding. Any ground acquired by either party within this zone is added to the JV, unless the ground was previously held and relinquished by either party.

Pursuant to the Liard Agreement, the Project is subject to a 5 mile Area of Interest clause as indicated in Figure 4-4.

4.4.1.3 Pembrook Royalty

Under the terms of an agreement signed on March 22, 2011, between Copper Fox and Pembrook Mining Corp. (Pembrook), a 2% NSR royalty is payable to Pembrook on any production from claim 521312. The Schaft Creek JV can purchase half of this royalty for $1.5 million at any time, leaving a 1% NSR on the claim.

4.4.1.4 Kreft/Greig Royalty

A purchase agreement signed on March 18, 2011, between Copper Fox and two private vendors, Kreft and Greig, includes a 2% NSR payable on claims 517462 and 569460. The Schaft Creek JV can purchase half of this royalty for $1.5 million at any time, leaving a 1% NSR on the claims.

4.4.1.5 Marko/Mott Royalty

Copper Fox purchased the claims listed in Table 4-1 that are subject to the Marko/Mott royalty from two private vendors, Marko and Mott, on September 14, 2011. Under the purchase agreement, the claims are subject to a 2% NSR. The Schaft Creek JV can purchase half of this royalty for $1 million at any time, leaving a 1% NSR on the claims.

The mineral claims subject to the Pembrook, Kreft/Grieg, and Marko/Mott royalties are located outside the resource area of the Project.

4.5 Property Agreements

The Schaft Creek JV has the following key terms:

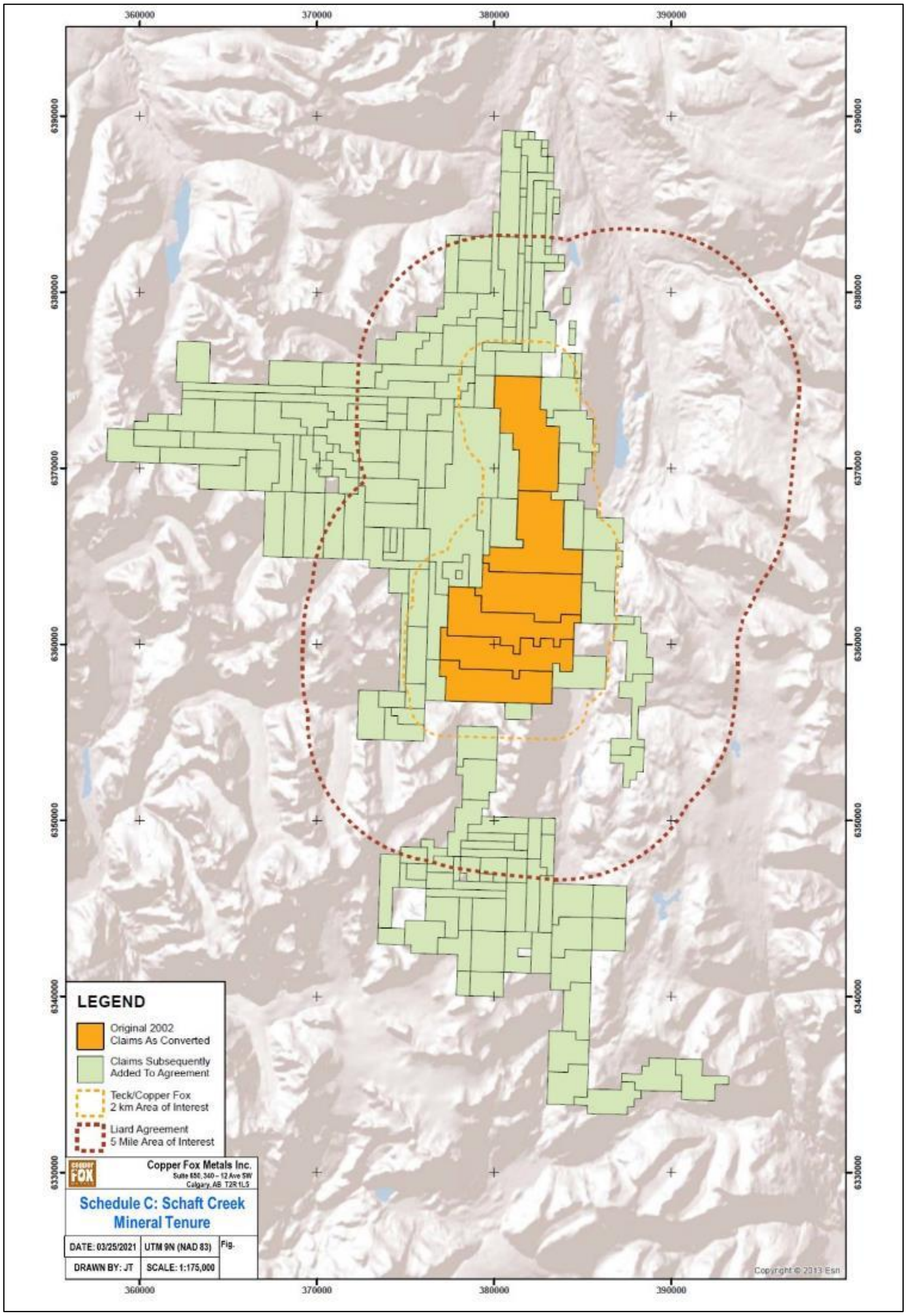
- Teck will pay a total of $60 million in three direct cash payments to Copper Fox: $20 million upon signing the JV agreement, $20 million upon a production decision, and $20 million upon the completion of the mine facility.
- Teck will fund 100% of costs incurred prior to a production decision, up to $60 million; Copper Fox's pro rata share of any pre-production costs in excess of $60 million will be funded by Teck, and the two remaining direct cash payments payable to Copper Fox will be reduced by an amount equal to Copper Fox's pro rata share of any pre-production costs in excess of the initial $60 million, to a maximum of total pre-production costs of $220 million.

- Teck will fund any additional costs (in excess of $220 million) incurred prior to a production decision, if required, by way of loan (at an interest rate of prime +2%) to Copper Fox to the extent of its pro rata share, without dilution to Copper Fox's 25% JV interest.

- Management of the Schaft Creek JV will be made up of two representatives from Teck and Copper Fox with voting proportional to equity interests.

- Teck agreed to use all reasonable commercial efforts to arrange project debt financing for not less than 60% of project capital costs of constructing a mining operation. If a production decision is made, Teck will fund Copper Fox's pro rata share of project capital costs by way of loans (at prime + 2%), if requested by Copper Fox, without dilution to Copper Fox's 25% JV interest.

Figure 4-4: Areas of Interest



4.6 Permitting Considerations

4.6.1 Environmental Assessment

Permission to develop and operate major mines in British Columbia is granted after the completion of an EA and permitting reviews by both the Provincial and Federal governments. The first review phase is conducted by the Provincial British Columbia Environmental Assessment Act (BCEAA) and the Federal Canadian Environmental Assessment Act (CEAA). These reviews involve approval of the mine concept and lead to issuance of an EA certificate under the BCEAA process, and approval from the Federal Minister of the Environment under the Federal CEAA process. Projects in British Columbia that are subject to both Provincial and Federal EA processes undergo a harmonized EA review. The subsequent provincial and federal permitting review phase, which can only be concluded after completing the EA reviews, involves a more detailed review of specific aspects of the mine, and results in issuance of a number of Provincial and Federal permits.

In 2013, Copper Fox had entered into the first stage of the EA process; however, in 2016, subsequent to significant changes in the permitting process, the Schaft Creek JV voluntarily withdrew from the EA process.

4.6.2 Current Permits

The Schaft Creek JV secured a multi-year area-based (MYAB) permit, MX-I-647 in 2018 from the Ministry of Energy, Mine and Petroleum Resources, which included an approval for a maximum of 50 core drill holes, 5 km of new drill road, and 20 km of line cutting. The Schaft Creek Project is operating under a five-year MYAB permit for exploration related activities. The MYAB was granted in 2018 and expires on March 31, 2023.

4.6.3 Future Permits

The majority of the major permits, licences, and authorizations required to support construction activities are under Provincial jurisdiction; however, some are under Federal authority. The key permits would include:

- Provincial: mining lease, discharge permit, water licence, authority to make a change in and about a stream, occupant licence to cut, special use permit, licence of occupation, surface lease, waste management permit, special waste generator permit (waste oil), sewage registration, camp operation permits, waterworks permit, fuel storage approval, food service permits, and highway access permit.

- Federal: CEAA approval; metal mining effluent regulations, fish habitat compensation agreement, Section 35(2) authorization for harmful alteration or disruption, or the destruction of fish habitat, navigable water stream crossings authorization, international river improvement permit, explosives factory licence, explosives magazine licence, ammonium nitrate storage facilities and radio licences.

A number of minor approvals from local governments (regional districts and municipalities) and Crown corporations would also be required, although these approvals are not required to commence construction.

4.7 Environmental Considerations

4.7.1 Copper Fox

Copper Fox initiated baseline environmental studies in the period 2006–2012. These included:

- Terrain
- Climate
- Air quality
- Noise
- Groundwater quality and quantity
- Surface hydrology
- Water quality
- Aquatic resources: streams, lakes and wetlands, sediment quality, periphyton, phytoplankton, zooplankton, benthic invertebrates
- Fisheries, fish and fish habitat
- Vegetation and plant communities
- Soils
- Wildlife and wildlife habitat
- Visual and aesthetic resources
- Economic
- Social, including traditional knowledge and traditional land use, non-traditional land use
- Heritage and archaeology
- Health, including drinking water and country foods

These studies remain relevant as baseline information for any future project development.

4.7.2 Schaft Creek JV

Since formation of the Schaft Creek JV, ongoing work has included environmental monitoring and collection of field data including humidity cell tests and other environmental baseline data.

4.8 Social and Community

4.8.1 Copper Fox

During 2006–2012, Copper Fox engaged with various members and institutions of the Tahltan Nation by way of informal meetings, working groups, and open house discussions. These consultations included members of the Tahltan Central Government and the Tahltan Heritage Resources Environmental Assessment Team (THREAT).

During the consultation process, Tahltan Nation leadership and members raised a range of discussion points under several different categories. These included effects on areas of high archaeological potential, cumulative effects of Schaft Creek worker and family wellbeing, ability to practice Tahltan Nation traditional activities, access to remote areas, interference with past and present Tahltan Nation uses of the Mess Creek Valley, Tahltan Nation food availability and safety, and wildlife habitat and migration corridors.

4.8.2 Schaft Creek JV

Since formation of the Schaft Creek JV, the Schaft Creek JV has continued ongoing consultation with the Tahltan Nation on social and cultural matters. A Communications and Engagement Agreement was signed between the Tahltan Central Government and Teck on behalf of the Schaft Creek JV on May 22, 2020.

4.9 Comments on Section 4.0

In the opinion of the QP:

- Information provided by Copper Fox supports that the Schaft Creek JV has valid title that is sufficient to support declaration of Mineral Resources.
- Surface rights are held by the Crown. Overlapping surface rights include a trapline, a licence of occupation for recreational purposes, and an outfitting area. The Schaft Creek JV would need to conduct appropriate negotiations for use of the surface, and obtain permits such as an Industrial Surface Lease, Licence of Occupation, easements, and rights-of-way to support future operations.
- No water rights are currently held. The Schaft Creek JV would need to obtain a Section 10 *Mines Act* Permit in support of operations.
- The royalty agreements related to various mineral tenures are in accordance with acceptable industry practice at the time these agreements were executed and have not been amended.
- As the Project is at the resource estimation stage, no major permits are in hand, or under application. In 2018, the Schaft Creek JV received a MYAB permit, which included approval for a maximum of 50 core drill holes, 5 km of new drill road, and 20 km of line cutting.
- Environmental liabilities include the reclamation bond posted by the Schaft Creek JV in the amount of $695,000 to cover the cost of reclaiming the camp, airstrip, storage, and core facilities.

To the extent known, there are no other significant factors and risks known to Copper Fox that may affect access, title, or the right or ability to perform work on the Project that are not discussed in this Report.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Property is accessible via rotary or fixed-wing aircraft. The two gravel airstrips adjacent to the Schaft Creek camp are approximately 700 m long and support fixed-wing and helicopter access. Dease Lake and the Ch'iyone camp on the Galore Creek road were used for various staging purposes during the 2015 and 2019 field seasons. In previous years, Telegraph Creek, Bob Quinn, and the Burrage airstrip have been used as staging areas. During the summer months in previous years, scheduled commercial flights have been available from Terrace or Smithers to the Dease Lake airstrip.

5.2 Local Resources and Infrastructure

The main road access route through the region is Highway 37, which runs north from the Terrace and Smithers area to the Yukon. Highway 37 passes by the Bob Quinn airstrip and Dease Lake en route to the Yukon. Telegraph Creek is accessed from Dease Lake via a 110 km drive on gravel roads to the southwest.

Terrace and Smithers have moderate-sized airports with commercial flights that connect to Vancouver. Amenities include a variety of stores, gas stations, rental car providers, hotel accommodations, RCMP detachments, hospitals, government offices, and offices for various exploration services contractors and drilling companies.

Dease Lake has a small airport with a paved runway. Amenities in Dease Lake include a grocery store, a gas station, hotel accommodations, a hardware store, an RCMP detachment, a medical centre, and the Tahltan Central Government office.

Basic services are available in Telegraph Creek, including a general store, an RCMP detachment, a nursing station, and the Tahltan Band Council office. Accommodation can be arranged, and fuel is available for purchase. A small gravel airstrip just outside of Telegraph Creek can be used as a staging area.

The BC Hydro Northwest Transmission Line (NTL) was completed in mid-2014. The NTL is a 287 kV transmission line that connects Bob Quinn to the Skeena Substation, located near the City of Terrace. An extension of the NTL has also been completed from Bob Quinn to Tatogga Lake and from Tatogga Lake to the Red Chris Mine.

5.3 Climate and Physiography

The Property is situated in mountainous terrain on the eastern side of the Coast Mountains of northwestern British Columbia. To the west of the Property is the rugged and glaciated terrain of the Coast Mountains. To the east of the Property is the Edziza Plateau. Elevations in the immediate area range from 850 m at valley bottom to mountain peaks over 2,000 m above mean sea level. The deposit is located west of a low saddle between the Schaft Creek and Mess Creek valleys at the south end of Mount LaCasse. The exploration camp is located at the base of this slope, in the broad valley of the north-flowing Schaft Creek, a braided stream with thick glacial-fluvial and fluvial deposits.

Treeline on the Property is located at approximately 1,400 m elevation. Below treeline, forests are composed primarily of balsam fir, sitka spruce, alder, willow, and cedar. Above this elevation, vegetation consists of scrub bushes, stunted ‘krummholz’ trees, and alpine grass, moss, and lichen. Above 1,800 m, vegetation is sparse. Much of the area immediately overlying the Schaft Creek deposit was burned during a fire in 1980. Another wildfire occurred in 2013 northeast of Mount LaCasse along the Mess Creek valley near Skeeter Lake.

The Property location on the eastern side of the Coast Mountains corresponds to a transitional zone between the warmer, moist maritime climate of the Coast Mountains to the west and the cooler, drier continental climate of the Interior Ranges to the east (Jones and Volp, 2013). In nearby Dease Lake, high temperature mean values during summer months range from 17°C to 20°C, with low temperature mean values between 3°C and 6°C. In winter months, high temperature mean values range from -5°C to -13°C, with low temperature mean values of -18°C to -22°C (www.climate-charts.com).

In the Project area, annual precipitation ranges between 70 cm and 100 cm at 1,000 m elevation. Snowfall is possible any month of the year, but commonly begins to accumulate at higher elevations in late September or early October. By late October, snow typically begins to accumulate in the valley bottom near the exploration camp (860 m elevation), reaching a typical depth of 1 m to 2 m before melting in May or early June. At 1,200 m elevation, the snowpack typically ranges from 2 m to 3 m, with considerably more snowfall expected in higher elevation alpine areas (Jones and Volp, 2013).

Field work can normally commence at lower elevations in early June and at upper elevations by July. Cold weather, high wind, and snow make field work difficult beyond mid-October.

5.4 Protected Areas

The Project is located within the boundaries of the Cassiar Iskut-Stikine (CIS) Land and Resource Management Plan (LRMP), which encompasses approximately 52,000 km^2 of northwestern British Columbia. The LRMP supports exploration and development in the area (excluding protected areas), including the development of access roads. Any activities are subject to any applicable environmental review processes. The LRMP identifies 15 resource management zones (RMZs) for area-specific management direction. The Middle Iskut RMZ boundary is approximately 25 km from the proposed Schaft Creek mine site and/or access road. The Schaft Creek project is located adjacent to the Mount Edziza Provincial Park and Tahltan Highlands to the east.

5.5 Seismicity

Information on the Project seismic setting is summarized from Farah et al. (2013).

The coastal northwest region of British Columbia and southwest Yukon is one of the most seismically active areas in Canada. The seismic hazard in the region is also influenced by the seismically active region of southeast Alaska.

A probabilistic seismic hazard analysis was carried out by Knight Piésold using the NRCan database to provide seismic ground motion parameters. The corresponding maximum acceleration is 0.06*g* for a return period of 475 years, indicating a low seismic hazard for the site.

5.6 Comments on Section 5.0

In the opinion of the QP:

- The existing local infrastructure, availability of staff, methods whereby goods could be transported to the Project area are well-established and well understood by Copper Fox, and can support the declaration of Mineral Resources.
- Within the tenure holdings, there is sufficient area to allow construction of infrastructure to support future mining operations.

6.0 HISTORY

6.1 Regional Government Geological Surveys and Academic Research

The area surrounding the Schaft Creek deposit has been mapped by several generations of geologists. The first geological study in the Stikine River area is thought to have been made by a group of Russian geologists who assessed the mineral potential of the district in 1863. Dawson and McConnell were the first Canadian geologists to explore the area in 1887, but the first geological mapping in the region was completed by Forrest Kerr from 1924 to 1929, published in 1948. Kerr's mapping along the Stikine and Iskut Rivers defined the Late Triassic Stuhini Group (Brown et al., 1996).

The Geological Survey of Canada's helicopter-supported 'Operation Stikine'[1] mapped the geology of the 104G Telegraph Creek map sheet in 1956 (Geological Survey of Canada, 1957). Jack Souther directed Operation Stikine and published a series of 1:250,000 scale geologic maps of several National Topographic System (NTS) map sheets, including 104G that contains the Schaft Creek Project (Souther, 1972). James Monger's regional synthesis subdivided the Late Paleozoic rocks and informally named them the Stikine Assemblage (Monger, 1977).

In 1988, Peter Holbek completed a master's thesis study on the BJ prospect, located 24 km south of Schaft Creek. This thesis included a detailed study of the BJ prospect and the host Stikine assemblage, as well as the intrusive and structural history of the region, including the Schaft Creek deposit area. Holbek's study included the Hickman, Yehiniko, and Nightout Plutons, which were grouped and named as the Hickman Batholith.

Logan et al.'s work on the Geology of the Forrest Kerr – Mess Creek Area resulted in the publication of Geoscience Map 1997-3 (Logan et al., 1997), which provides the most recent regional scale mapping available for the deposit area, and the publication of Bulletin 104 (Logan et al., 2000), which provides a detailed description of the regional geology. Brown et al.'s 1996 British Columbia Geological Survey Bulletin 95, detailing the 'Stikine Project' mapping of the Geology of Western Telegraph Creek Map Area in Geoscience Maps 1993-3 to 1993-6, provides the most recent regional scale mapping available for the area west and northwest of the deposit.

Geological descriptions of the Schaft Creek deposit were published following the conclusion of Teck's exploration campaigns during the 1970s and 1980s (Fox et al., 1976; Spilsbury et al., 1995). James Scott's 2007 master's thesis and CIM publication (Scott, 2007; Scott et al., 2008) comprises a more recent, detailed study of the geology and genesis of the deposit.

Recently, both the British Columbia Geological Survey and the Geological Survey of Canada have been active in northwestern British Columbia, and this recent work is relevant to the Project. Recent investigations of the KSM-Brucejack district (Nelson and Kyba, 2013), and the Khyber-Sericite-Pins mineralized trend (Kyba and Nelson, 2014) have improved the knowledge of regional stratigraphy during the Late Triassic and Early Jurassic and improved the understanding of mineralization in these

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[1] Systematic regional mapping of four 1:250,000 map areas of the Stikine Region of northwest BC by the Geological Survey of Canada in 1956.

areas. Recent investigations of Triassic mafic and ultramafic rocks within the Stikine Terrane have identified primitive, magnesium-rich, olivine-bearing rocks that are interpreted to represent a slab-metasomatized mantle environment from which magmatism was sourced during the Late Triassic (Milidragovic et al., 2016).

6.2 Exploration History

The Stikine River was used as an access route to the gold fields of the Klondike, and placer gold was discovered along the river in the early 1920s.

Copper mineralization was first discovered in the Schaft Creek area in 1957 by prospector Nick Bird, employed by the BIK Syndicate. He staked the first four claims in August of 1957 for BIK, a consortium of Silver Standard Mines, Ltd. (Silver Standard), McIntyre Porcupine Mines Ltd., Kerr Addison Mines Ltd., and Dalhousie Oil Ltd. (Linder, 1975). This was a year after the Geological Survey of Canada's Operation Stikine began mapping the regional geology of the area. Only two years earlier, in 1955, copper mineralization had been discovered at Galore Creek. Helicopter access to this remote region played a significant role in these 1950s discoveries. In 1959, while the Project was under option to Kennco Explorations (Western) Limited, an additional 42 claims were staked, and geological, geophysical, and geochemical surveys were carried out before the option was dropped (Department of Energy, Mines, and Resources, 1986).

Silver Standard, acting as the operator for the BIK Syndicate, completed geochemical and geophysical surveys, and staked more ground. The first drill holes were completed in 1965, when Silver Standard completed three BQWL-sized drill holes totaling 629 m (Figure 6-1 summarizes the drilling completed on the Project). Liard, a private company, was incorporated in 1966 by the participants of the BIK Syndicate to hold the Project, with Silver Standard holding a 65.6% interest (Department of Energy, Mines, and Resources, 1986).

Liard optioned the ground to the American Smelting and Refining Company (Asarco) in 1966. Before dropping the option in 1968 Asarco built a camp and an airstrip, and completed an exploration program, including geological mapping and prospecting, induced polarization (IP) surveys, and 3,334 m of drilling in 24 holes (Kulla, 2011).

In 1966, the adjacent tenure to the north was acquired by Paramount. Paramount completed ground geochemical and geophysical surveys, 450 m of trenching, and one AX-sized drill hole to 150 m depth (Department of Energy, Mines, and Resources, 1986).

In 1968, Liard optioned the Project to Hecla. Hecla optioned the Paramount ground in 1969, and completed extensive exploration on both the Liard tenure and Paramount tenure from 1968 to 1972. Hecla's work included percussion and diamond drilling totaling 29,616 m in 83 drill holes, in addition to geophysical surveys and geological mapping, as well as a pre-feasibility study (PFS) (Kulla, 2011). Hecla's interest in the Project waned in 1973, with the company citing provincial and federal government policy changes as the cause of their curtailed work. Hecla dropped the option on Paramount in 1973, stating that future commitments under the option agreement could not be logically fulfilled by due dates, but continued minor work on the Liard tenure, drilling one hole in 1974 and another in 1977 totaling 1,178 m (Department of Energy, Mines, and Resources, 1986).

In 1978, Teck acquired Hecla's option to earn a 70% interest in the Liard Project (Department of Energy, Mines and Resources 1986). The Paramount tenure lapsed, and Teck re-staked this ground (Spilsbury, 1995). In 1980 and 1981, Teck completed 25,642 m of drilling to confirm and expand on Hecla's work. Work included geophysical surveys, condemnation drilling, resource estimation, and engineering studies (Kulla, 2011). Exploration in the area slowed in the early 1980s, with low metal prices curtailing exploration funding.

Copper Fox acquired an option on the Project through an assignment of an agreement made between Teck and Guillermo Salazar dated January 1, 2002. In February 2003, Salazar assigned the option to 955528 Alberta Ltd, which amalgamated with a newly incorporated Copper Fox in 2004. Between 2005 and 2012, Copper Fox carried out extensive exploration work on the Project, including 41,345 m of drilling. In 2004, Copper Fox had an initial Mineral Resource estimate prepared; this resource estimate was updated in 2007 for a PEA. It was again updated, and mineral reserves were declared for a 2008 PFS. The Mineral Resource was updated in 2011 and again in 2012. The Mineral Resource completed in 2012 is historical in nature and no longer current but is set out below for information.

The 2012 project mineral resource was prepared by Tetra Tech, with an effective date of May 23, 2012 (Table 6-1, see news release dated May 31, 2012).

Table 6-1: Historic 2012 Mineral Resource

Mineral Resource Estimate – Schaft Creek Deposit										
Robert Morrison - Ph.D., MAusIMM (CP), P.Geo., Effective Date: May 23rd, 2012										
Resource Category	**Cut-off CuEq (%)**	**Tonnes**	**Copper (%)**	**Molybdenum (%)**	**Gold (gpt)**	**Silver (gpt)**	**Contained Metal Cu (Lbs)**	**Mo (Lbs)**	**Au (oz.)**	**Ag (oz.)**
Measured	0.15	146,615,300	0.31	0.017	0.24	1.78	1,001,824,600	55,624,000	1,149,100	8,402,700
Indicated	0.15	1,081,939,500	0.26	0.017	0.18	1.68	6,104,400,000	399,718,500	6,218,000	58,335,500
Measured & Indicated	0.15	1,228,554,800	0.26	0.017	0.19	1.69	7,106,224,600	455,342,500	7,368,000	66,738,200
Inferred	0.15	597,191,300	0.22	0.016	0.17	1.65	2,872,034,300	206,252,100	3,359,600	31,601,400

Notes: These Mineral Resources are not current and the following notes were made at the time 2012 Mineral Resource estimate and do not apply to the current Mineral Resource Estimate.

Mineral Resources are inclusive of Mineral Reserves;

While the terms "Measured (Mineral) Resource", "Indicated (Mineral) Resource" and "Inferred (Mineral) Resource" are recognized and required by National Instrument 43-101 – *Standards of Disclosure for Mineral Projects*, investors are cautioned that except for that portion of Mineral Resources classified as Mineral Reserves, Mineral Resources do not have demonstrated economic viability. Investors are cautioned not to assume that all or any part of measured or indicated Mineral Resources will ever be upgraded into Mineral Reserves. Additionally, investors are cautioned that inferred Mineral Resources have a high degree of uncertainty as to their existence, as to whether they can be economically or legally mined, or will ever be upgraded to a higher category;

A 0.15% CuEq cut-off was selected for the base case resource estimate. A 0.15% CuEq cut-off was the minimum grade of CuEq estimated by Tetra Tech required (using the estimated copper recovery rate, the milling and sales cost) to break-even on an operating cost per tonne basis;

CuEq grade cut-offs were used to report the Mineral Resource estimation as a function of copper, molybdenum, gold, and silver. The CuEq is based on Tetra Tech's long-range metal prices of US $2.97/lb for copper, US $16.80/lb molybdenum, US $1,256.00/oz gold and US $20.38/oz for silver and metal recoveries of 60.90% for molybdenum, 70.6% for gold, and 43.4% for silver. No copper recoveries were applied to the copper equivalent grade;

Rounding as required by reporting guidelines may result in apparent summations differences between tonnes, grade and contained metal content; and

Tonnage and grade measurements are in metric units. Contained copper and molybdenum are reported as lbs and contained gold and silver are reported as troy ounces.

On January 23, 2013, Copper Fox filed a NI 43-101 technical report entitled "Feasibility Study on the Schaft Creek Project, BC Canada" prepared by Tetra Tech with A. Farah, P.Eng. et al. as Qualified Persons. The Mineral Reserve estimate used in the 2013 FS is historical in nature and is not current, which are provided here for information purposes. The Proven and Probable Reserves used in the 2013 FS are summarized in Table 6-2.

Table 6-2: Historic 2013 Mineral Reserves

Mineral Reserve					
Reserve Category	**Run of Mine (Mt)**	**Copper %**	**Molybdenum %**	**Gold gpt**	**Silver gpt**
Proven	135.4	0.31	0.0175	0.25	1.81
Probable	805.4	0.26	0.0176	0.18	1.70
Proven & Probable	940.8	0.27	0.0176	0.19	1.72

Notes: These Mineral Reserves are not current and the following notes were made at the time of the 2013 Mineral Reserve estimate. There are no current Mineral Reserves on the Project.

Mineral Reserves are contained within Measured and Indicated pit designs.

Appropriate mining costs, processing costs metal recoveries, and inter ramp pit slope angles varying from 27 degrees in overburden to 45 degrees in bedrock were utilized to generate the pit phase design;

Mineral Reserves have been calculated using a NSR cut-off. The NSR was calculated as follows: NSR = Recoverable Revenue – TCRC (on per tonne basis), where NSR = Net Smelter Return; TCRC = Transportation and Refining Costs; Recoverable Revenue = Revenue in Canadian dollars for recoverable copper, molybdenum, gold and silver, respectively, using metal prices of US $3.52/lb, US $15.30/lb, US $1,366.00/oz and US $25.96/oz for copper, molybdenum, gold and silver, respectively; at an exchange rate of CAD$0.96 to US $1.00; metal recoveries used are based on recovery curves if critical average recoveries could be calculated;

The LOM average strip ratio (waste to mill feed) is 2:1, excluding rehandle;

Rounded significant digits on ROM to one decimal point;

Rounding as required by reporting guidelines may result in apparent summations differences between tonnes, grade and contained metal content, and

Tonnage and grade measurements are in metric units. Contained copper and molybdenum are reported as lbs and contained gold and silver are reported as troy ounces.

The proven and probable Mineral Reserves are included within the Measured and Indicated Mineral Resources as estimated by Tetra Tech on May 23, 2012.

In February of 2013, Copper Fox filed an NI 43-101 technical report to support the 2013 FS.

In July 2013, Teck and Copper Fox formed the Schaft Creek JV, with Teck resuming as the project operator. Subsequent to the formation of this JV, the Schaft Creek JV completed a program of nine drill holes totalling 3,454 m in 2013. During 2013 and 2014, the Schaft Creek JV completed geological mapping, relogging of historical core, geological modeling, and an airborne geophysical survey. Table 6-3 outlines the work performed on the Project.

Table 6-3: Exploration History Summary

Year	Company	Comments
1957	BIK Syndicate	Mineral claims first staked in the region by prospector Nick Bird for the BIK Syndicate (consortium of Silver Standard, McIntyre Porcupine Mines Limited, Kerr Addison Mines Ltd., and Dalhousie Oil Ltd.) 914.4 m hand trenching, rock chip sampling.
1965–1966	Silver Standard	Prospecting syndicate was re-organized into Liard Copper in order to recognize the respective interests of its members and to consolidate the holdings in the area. Silver Standard, with a 66% interest, was the operator of the Project. Geological mapping (eight traverses), hand-trenching (3,000 ft.), IP survey, three core drill holes (629 m).
1966	Paramount	450 soil samples, IP and magnetic geophysical surveys, claim 517462 staked.
1966	Liard Copper Mines	Consolidated mineral tenures in area, optioned ground to Asarco.
1966–1967	Asarco	Two airstrips constructed, camp built; 24 core drill holes (3,334 m), IP survey.
1968–1977	Hecla	Asarco options property to Hecla, airstrip extended, 29,616 m core drilling and 6,500 m percussion drilling, IP and resistivity surveys, geological mapping (covered area of 10 by 6 miles at a scale of 1:400), aerial photography, tonnage and grade estimates, metallurgical test work, engineering studies, local grid established.
1969–1972	Paramount	10 drill holes (2,924 m).
1971	Geological Survey of Canada	Regional geology mapped at a scale of 1:250,000.
1972	Phelps Dodge Corporation of Canada Ltd.	Soil and silt geochemical survey, cobra drill and bulldozer trenching, IP and magnetometer geophysical surveys.
1974	Hecla	Established grid of cut lines, low level air photography, IP surveys.
1978	Hecla/Teck	Hecla sold interest to Teck.
1978 – 2002	Teck	1980: 45 diamond drill holes (14,490 m); rock chip samples 1981: 81 diamond drill holes (11,154 m), IP survey, tonnage and grade estimate; metallurgical test work; internal evaluation studies.
2002	Teck / Copper Fox	Teck options property to Salazar / Copper Fox.
2002	Copper Fox	Copper Fox completes assessment of project and geologic model

table continues…

Year	Company	Comments
2005–2006	Pembrook	Limited reconnaissance mapping program, collected five rock samples in 2005; 2006 follow-up work to 2005 program, 24 rock samples on Pembrook claims, identified two gold and copper anomalous zones.
2005–2013	Copper Fox	2005: 15 diamond drill holes (3,160 m); metallurgical bulk sample collected, metallurgical test work. 2006: 42 diamond drill holes (9,007 m); additional metallurgical sample collected; metallurgical test work. Environmental studies including hydrology baseline report, moose baseline report, bird study, meteorology baseline report, fisheries baseline report, aquatics baseline report. 2007: 42 diamond drill holes (6,275 m), Mineral Resource estimate, IP survey. Environmental studies including aquatic resources baseline report, archeological baseline study technical summary, meteorology baseline report, bat inventory, preliminary groundwater baseline report, noise baseline report, soils baseline report, vegetation baseline report, western toad baseline, hydrology baseline, fisheries baseline, Tahltan (country) foods baseline assessment, access road assessment, wetlands baseline report, metal leaching/acid rock drainage (ML-ARD) phase 1 report. 2008: 47 diamond drill holes (6,821 m), PEA, PFS. Environmental studies including environmental and social work plans, alternatives assessment report, geohazard tailings options, tailings assessment engineering, access road assessment Tahltan highland, tailings water management assessment, hydrology baseline, fisheries baseline report, fisheries addendum, bird studies addendum, aquatics baseline report, vegetation and ecosystem mapping baseline, navigable waters assessment, ML-ARD phase 2 report, ML-ARD assessment of surficial samples from the proposed access road, access route geohazards, mountain ungulate baseline. 2009: Metallurgical test work. Environmental studies including meteorology and air quality baseline, country foods baseline update, baseline hydrogeology study. 2010: 14 diamond drill holes (4,010 m). Mineral Resource estimate, metallurgical test work. Titan-24 geophysical survey comprising direct current IP and magneto-telluric surveys. Environmental studies including wildlife habitat suitability baseline, moose literature review, engineering hydrometeorology report, ML-ARD geochemical shake-flask testing of overburden in the proposed pit area, ML-ARD report on 3D modelling of acid-base accounting (ABA) data, ML-ARD report on ABA and total solid-phase elements for rock, socio-economic baseline, land use baseline, soils baseline, archaeology baseline study, geomorphic channel assessment and channel migration hazard mapping of Upper Mess Creek.

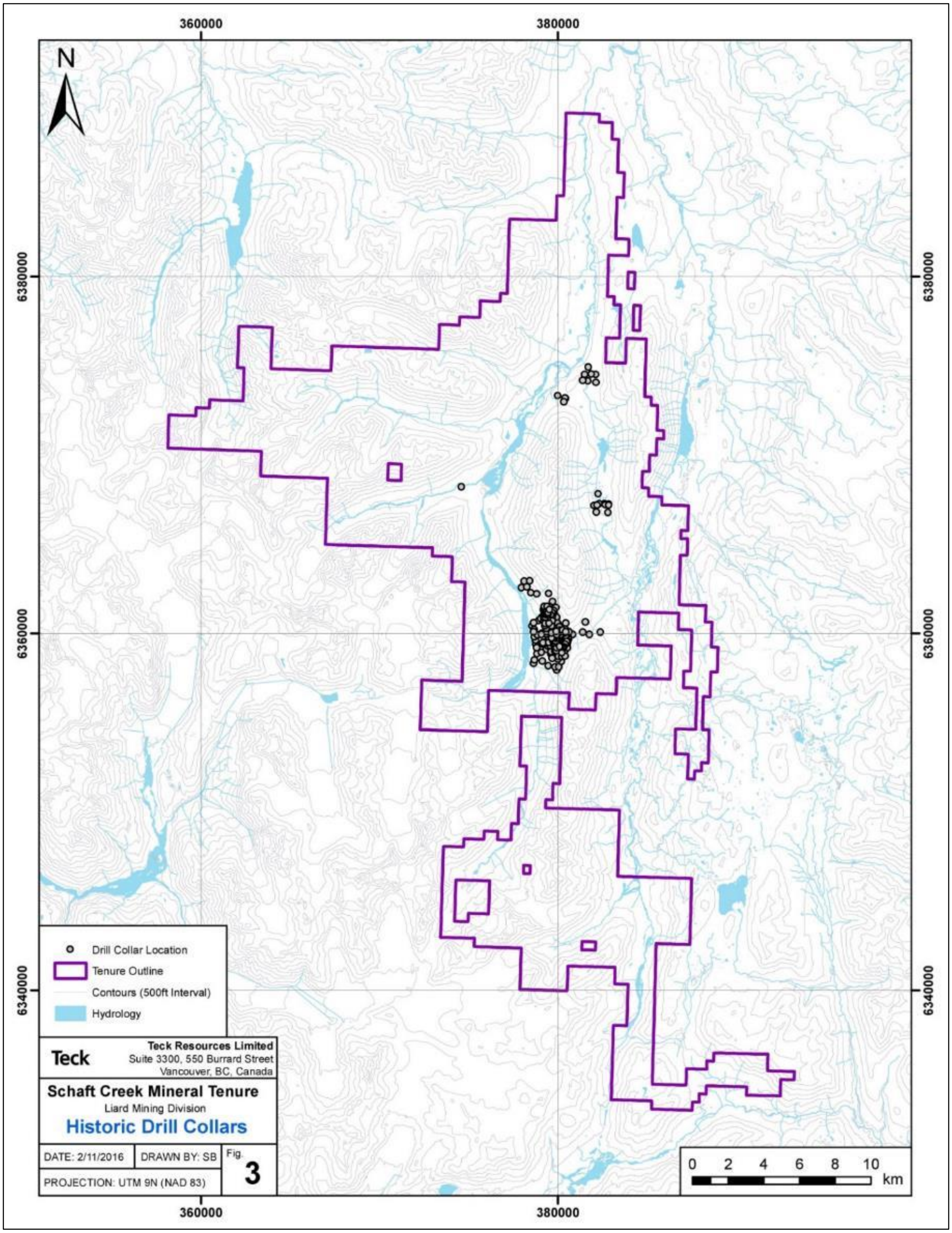
table continues…

Year	Company	Comments
		2011: 22 diamond drill holes (9,649 m), High-resolution aeromagnetic survey. Titan-24 geophysical survey. Completion of Schaft Creek Mine Project Application Information Requirements/Impact Statement Guidelines. Schaft Creek Project prediction of mine-site-drainage chemistry, ML-ARD through 2011 report. 2012: Mineral Resource estimate, metallurgical test work. Five diamond drill holes (2,266 m) in Discovery and Mike Zones. Additional high-resolution aeromagnetic survey. Winter moose population and distribution survey. 2013: Feasibility study. Z-Axis tipper electromagnetic geophysical survey. Schaft Creek 2011 baseline hydrogeology study.
2008	Claims held by Charles J. Greig, and John Bernard Kreft	Reconnaissance sampling spaced 25 m apart, 183 soil samples, 17 grab and chip samples.
2013–Date	Schaft Creek JV	2013: Five diamond drill exploration and four geotechnical holes (3,453 m). Metallurgical, pit slope design, mapping, core re-logging, geological modelling and environmental studies. Copper-gold-molybdenum-silver mineralization intersected east of the resource block model indicating that the mineralization in the Schaft Creek deposit is open to the east. 2014: Metallurgical, pit slope design, geological modelling and environmental studies; core re-logging, geometallurgical modelling, and collection of additional metallurgical samples for variability testing. Identified LaCasse Zone. 2015: Five diamond drill holes (2,634 m) in LaCasse Zone, optimization studies, core re-logging for geometallurgical, lithogeochemistry, and ARD investigations, structural modelling, evaluation of soils in areas that could potentially be locations for tailings storage, geophysical survey over the south extension of the Liard zone of the Schaft Creek deposit. A total of 100 representative samples of the lithologies and alteration across the Schaft Creek deposit were collected for geometallurgical test work. Results of the geotechnical, comminution and electrical portions of optimization studies indicated similar findings to those of the 2013 FS. 2016: Core re-logging, environmental monitoring, and collection of field data including humidity cell tests and other environmental baseline data and ongoing consultation with the Tahltan Community on social and cultural matters. Voluntary withdrawal from the EA process and the queue for the Northern Transmission Line for the Project. 2017: Geological and Resource modelling, desktop engineering and trade-off studies, continuation of collection of environmental baseline data and ongoing social activities. Application for a Multi-Year Area-Based permit.

table continues…

Year	Company	Comments
		2018–2019: Updated Resource modelling, ongoing environmental studies, desktop studies to further investigate the geotechnical characteristics of areas that could host tailings facilities, internal sizing and infrastructure alternatives study, internal conceptual study to confirm scenarios that could potentially lower costs, infrastructure and further define access options, and ongoing permitting and community relations activities. A berm reinforcement program was completed with the installation of an extensive gabion wire mesh retaining wall system to protect the camp, drill core, and fuel storage areas from the effects of a 100-year flood event. 2020: Despite the COVID-19 pandemic, a baseline monitoring program was safely completed which consisted of annual servicing of the Schaft Creek Camp and Mount LaCasse climate stations and inspection and downloading of the information from the Skeeter Creek hydrology station located at the northern end of the TSF. A Communications and Engagement Agreement with the Tahltan Central Government was successfully negotiated and signed.

Figure 6-1: Map of Historic Drilling on the Schaft Creek Project



Source: Schaft Creek JV (2016)

6.2.1 Schaft Creek JV 2015 Program

The 2015 field campaign completed by the Schaft Creek JV consisted of drilling, IP and magnetometer surveys, rock sampling, soil sampling, geological mapping, surficial geological mapping, geometallurgical sampling, relogging of historical drill core, and updating the 3D geological model.

The 2015 drill program tested the LaCasse Zone, located approximately 4 km north of the deposit area. Drilling at LaCasse consisted of a five-hole, 2,634 m program using a helicopter-portable diamond drill. All five drill holes intersected porphyry-style alteration and veins, as well as associated copper mineralization. The most significant assay results from this program were returned by drill hole SCK-15-444; this hole intersected 182.5 m of 0.20% Cu, including a subinterval of 30 m of 0.40% Cu. Interpretation of drill core and geological mapping suggests that mineralization at LaCasse may be contiguous with the Discovery Zone to the south. In addition, the mineralization style at LaCasse is interpreted to have similarities with the Paramount Zone. Mineralization at LaCasse remains open in several directions, and there is potential in this area to discover high-grade breccia mineralization of the style that occurs at the Paramount Zone.

The 2015 field program explored the Wolverine Creek area, located immediately south of the Liard Zone. The limits of the Liard Zone are not well constrained to the south, and this area has poor outcrop exposure and limited, shallow historical drilling. Exploration activities included geological mapping, relogging of historical drill core, rock sampling, B-horizon soil sampling, and an IP and ground magnetometer survey. As a result of this work, a drill target concept emerged in this area based on coincident chargeability and soil geochemistry anomalies, and structural interpretation. This work better defined the southern limits of the Liard Zone and highlighted potential for resource expansion south of the Basal Fault.

Regional geologic mapping was conducted along the margin of the Hickman Batholith and included two areas of focus: (1) the northern portion of Mount LaCasse; and (2) a mountain located south of the Schaft Creek deposit, colloquially named Mount Hicks. In total, 18 km^2 was mapped at 1:5,000 scale. The mapping completed north of Mount LaCasse included field checking of several historically known mineralized showings, including the Grizzly and Greater Kopper areas.

The 2015 geometallurgical sampling program focused on comminution testing. The test results were used for throughput simulation modelling; the results of this modelling are comparable to the throughput range calculated in the previous FS (Morrison and Karrei, 2012).

Surficial geological mapping was completed in the Skeeter Lake area, which is the location of the proposed TSF. Colluvial, bedrock, morainal, and fluvial units are the most widespread surficial units within this area. Geological hazards are widespread in the area and mainly include numerous debris flow paths, rock fall and snow avalanche hazards, debris slides, and rock slides. Previous interpretations of a large slope sag structure in the mountains to the east of Skeeter Lake (Holm, 2011) were not supported by the 2015 surficial mapping.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

7.1.1 Tectonic Setting

The Project is located in the northwestern portion of Stikine Terrane, within the Canadian Cordillera. The Stikine Terrane is a tectonostratigraphic domain comprised of Paleozoic to Mesozoic sedimentary rocks as well as volcanic and comagmatic plutonic rocks of island-arc affinity (Monger et al., 1982; Logan et al., 2000). Figure 7-1 shows the position of the Stikine Terrane in northwestern British Columbia. The geological, palaeontological, and paleomagnetic signatures of this island arc indicate that it is allochthonous (Gabrielse et al., 1991), and the terrane is interpreted to have accreted onto the margin of North America by the Middle Jurassic (Nelson and Colpron, 2007). Mineralization at the Schaft Creek deposit, along with the vast majority of other Cu-Au deposits and occurrences in northwestern British Columbia, occurred while the Stikine Terrane was located outboard from North America. During this time, the characteristics of the Stikine island arc were probably analogous to modern island arcs in the southeast Pacific Ocean, such as the Philippine Islands. Following mineralization, the Stikine Terrane accreted onto the margin of North America during the Middle Jurassic and was subsequently impinged upon by accretion of the Insular Terranes and other outboard terranes during the Late Jurassic (Nelson and Colpron, 2007; Nelson et al., 2013).

7.1.2 Regional Stratigraphy

The Stikine Terrane includes three major lithostratigraphic units: the Paleozoic Stikine Assemblage, the Late Triassic Stuhini Group, and the Early Jurassic Hazelton Group (Logan et al., 2000). These rocks are overlain by younger clastic sediments of the Middle Jurassic to Early Tertiary Bowser Lake and Sustut Groups, and Eocene to Recent volcanic rocks of the Edziza and Spectrum Ranges (Logan et al., 2000). The three major units of the Stikine Terrane are described below. Figure 7-2 presents the regional geology of the Schaft Creek area.

The Stikine Assemblage is the structurally and stratigraphically lowest package observed in the Project area (Figure 7-3). The assemblage consists of a Devonian, Carboniferous, Permian, and Early to Middle Triassic-aged submarine succession of volcanic and sedimentary rocks (Logan et al., 2000). The dominant rock types are tholeiitic to calc-alkaline, mafic and bimodal flows, and volcaniclastic rocks, with interbedded carbonate, shale, and chert. Thick sequences of Permian limestone and mafic volcanic rocks are among the most distinctive rock types of the Stikine Assemblage within the Project area. These rocks occur east of the deposit in the Skeeter Lake Valley and Mess Creek Valley and to the west, south, and southeast of the Hickman Batholith. The Stikine Assemblage is unconformably overlain by the Stuhini Group. This unconformity is locally masked by a steep normal fault or thrust fault.

Figure 7-1: Stikine Arch Map

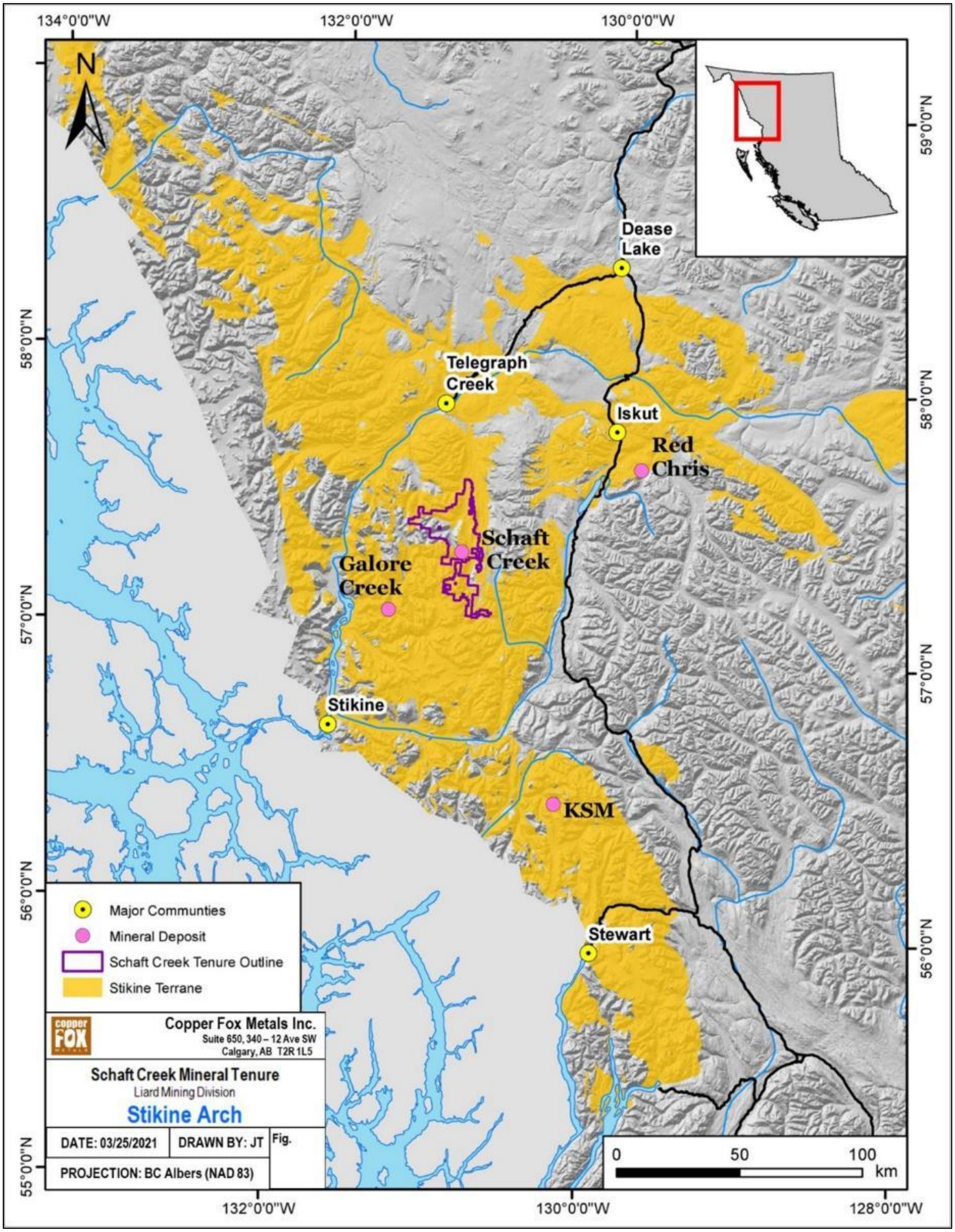


Figure 7-2: Regional Geology Map

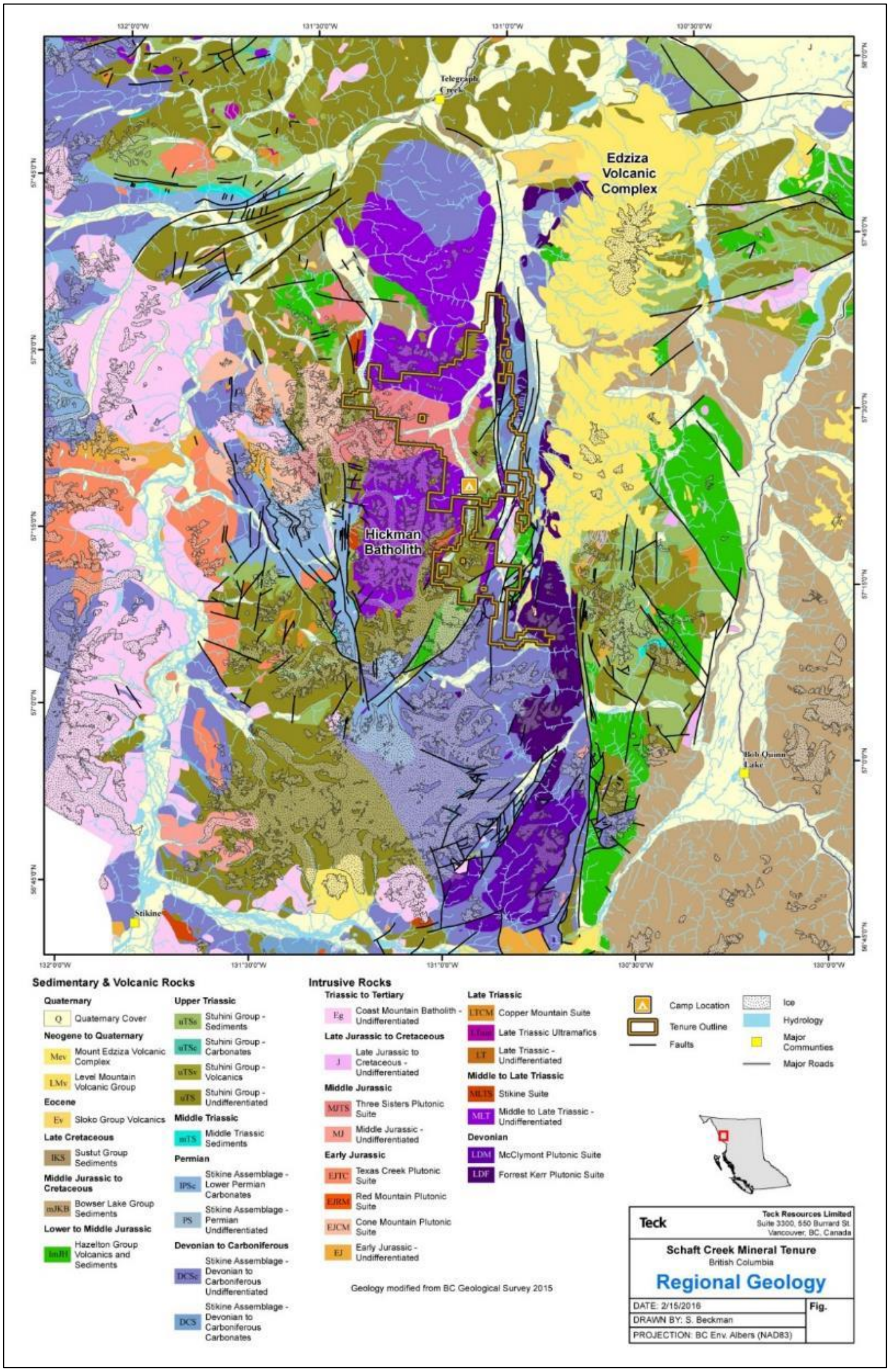
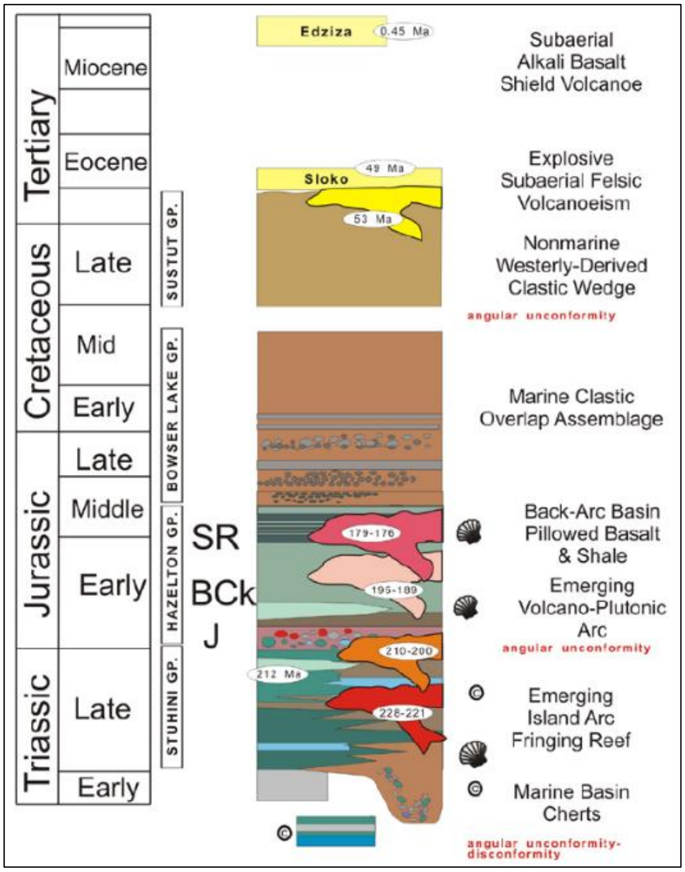


Figure 7-3: Regional Stratigraphic Column



Source: Logan et al., 2000

The Stuhini Group consists of Late Triassic pyroxene and/or plagioclase porphyritic andesitic to basaltic volcanic flows, coarse volcaniclastic breccias, lapilli tuffs, crystal tuffs, and rare epiclastic sandstone and siltstone. Subvolcanic pyroxene and/or plagioclase phyric rocks also occur within the volcanic pile and are considered to be part of the volcanic feeder system (Logan et al., 2000). In the Project area, the Stuhini Group has been divided by previous workers into five map units. From oldest to youngest, these are (1) a lowermost succession of dun-weathering, recessive, mafic to ultramafic lapilli tuff and minor flows (uTSmt); (2) dark grey, massive plagioclase-phyric basalt and subvolcanic intrusive rock (uTSvb); (3) grey to mauve, massive to weakly stratified, polylithic lapilli tuff and crystal tuff with augite and plagioclase crystal fragments (uTSvt); (4) green to maroon, interbedded, augite-phyric, plagioclase-phyric, augite and plagioclase-phyric, aphyric basaltic andesite flows, and subvolcanic intrusive rocks (uTSpp); and (5) green, well-bedded, fine grained volcaniclastic tuff, tuffaceous siltstone and sandstone, and wacke (uTSs) (Logan et al., 2000). The Schaft Creek deposit and other related mineralization occurs within the Stuhini Group, predominantly within the uTSvt and UTSpp map units. The uppermost Stuhini Group unit (uTSs) has been described to be locally conformable with the overlying basal conglomerate unit of the Hazelton Group (lJHcg) (Logan and Drobe, 1992); however, more commonly, the contact between the Stuhini Group and the overlying Hazelton Group is marked by an angular unconformity or a fault.

The Hazelton Group consists predominantly of volcanic rocks, including basalt, andesite, dacite, rhyolite, and coeval subvolcanic stocks and plugs, as well as lesser sedimentary and volcaniclastic rocks, such as tuff, volcaniclastic wacke, and shale (Logan et al., 2000). The majority of these rocks are subaerial, which is in contrast to the predominantly submarine depositional environment of the Stuhini Group. The lowermost unit of the Hazelton Group is a distinctive polylithic conglomerate that contains a variety of clast types, including volcanic, volcaniclastic, granodioritic, quartz, and limestone clasts (lJHcg). This basal conglomerate unit of the Hazelton Group crops out on top of a mountain colloquially named Mount Hicks, located approximately 3 km south of the Schaft Creek deposit.

7.1.3 Regional Plutonic Suites

The northwestern portion of the Stikine Terrane is intruded by plutonic rocks that represent at least seven magmatic episodes: Late Devonian, Early Mississippian, Late Triassic, Late Triassic to Early Jurassic, Early Jurassic, Middle Jurassic, and Eocene (Logan et al., 2000). Within the Project area, the majority of plutonic rocks occur in two main north-south trending intrusive belts: (1) the Late Devonian to Early Mississippian belt comprised of the Forest Kerr and More Creek Plutons; and (2) the Late Triassic to Middle Jurassic belt comprised of the Hickman, Yehiniko, and Nighout Plutons and the Loon Lake Stock. To the northeast, the Eocene to Recent rocks of the Edziza and Spectrum Ranges comprise a third major north-south trending volcanic and intrusive belt. Other small plutons occur scattered throughout the region, most notably the volumetrically minor alkalic intrusive rocks of the Late Triassic to Early Jurassic Copper Mountain Suite. The two major intrusive belts in the Project area are described in more detail below.

The Late Devonian to Early Mississippian intrusive belt occurs to the south and southeast of the Schaft Creek deposit, and is approximately 60 km in length and up to 10 km wide. The two main plutons that comprise the belt, the Forrest Kerr and the Mess Creek Plutons, are separated by the northeast-trending Newmont Lake Graben (Logan et al., 2000). The composition and mineralogy of these two plutons are similar, and each contains several intrusive phases, including hornblende monzodiorite, diorite, tonalite, leucocratic granodiorite, and biotite trondhjemite. In general, the mafic phases occur as inclusions within and are brecciated or crosscut by the felsic phases (Logan et al., 2000). U-Pb zircon dating of the felsic phases of the Forrest Kerr Pluton yielded ages of 369.4 ± 5.1 and 370.7 ± 6.7 Ma, whereas the felsic phase of the More Creek Pluton yielded an age of 356.9 +4.3/-3.8 Ma (Logan et al., 2000).

The Late Triassic to Middle Jurassic intrusive belt occurs immediately west of the Schaft Creek deposit and extends to the north and south along a strike length of approximately 70 km, with a width up to 20 km. The Hickman, Yehiniko, and Nighout Plutons are contiguous in this belt and occur within a distinct physiographical domain of high mountains and large valley glaciers; this geological and physiographical domain has been named the Hickman Batholith (Holbek, 1988). The Loon Lake Stock occurs as a separate intrusion located on the east side of the Mess Creek Valley. These intrusive units are compositionally distinct, and each is described separately below.

The Hickman Pluton is comprised of several distinct phases, including clinopyroxenite, hornblendite, hornblende biotite granodiorite, plagioclase porphyritic diorite, quartz monzonite, and quartz monzodiorite. Ultramafic and mafic phases are crosscut or brecciated by the more felsic phases, which are interpreted to be younger. Locally, mafic and felsic phases of the pluton display magmatic foliation, flow banding, and magma mixing textures. U-Pb zircon dating of samples collected by the Schaft Creek JV in 2013 and 2014 from the Hickman Pluton yielded ages including 222.32 ± 0.6 Ma for the plagioclase porphyritic diorite in the western part of the pluton; 221.52 ± 0.06 and 221.69 ± 0.07 Ma for granodiorite and monzonite phases in the central part of the pluton; 220.92 ± 0.06, 220.93 ± 0.13, 220.91 ± 0.21, and 220.32 ± 0.15 Ma for granodiorite and quartz monzodiorite phases in the eastern part of the pluton near the Schaft Creek deposit; and 219.27 ± 0.26 and 219.43 ± 0.18 Ma for porphyritic quartz monzodiorite dikes that are spatially associated with mineralization in the Liard Zone within the deposit (unpublished U-Pb dating by Richard Friedman, University of British Columbia; unpublished U-Pb dating by Jim Crowley, Boise State University). Previous U-Pb zircon dating of a porphyritic quartz monzonite dike from the Liard Zone yielded an age of 216.6 ± 2 Ma (Logan et al., 2000).

The Yehiniko Pluton consists of several phases, including coarse grained biotite granite, fine grained monzonite, and syenite. East of the pluton, near Mount LaCasse, a swarm of flow-banded syenite to rhyolite dikes and sills crosscut mineralized veins and breccias that are hosted within the Hickman Pluton. U-Pb zircon dating of samples collected by the Schaft Creek JV in 2013 and 2014 from the Yehiniko Pluton yielded ages including 176.23 ± 0.13 Ma for fine grained syenite in the eastern part of the pluton, 178.87 ± 0.09 and 177.20 ± 0.05 Ma for flow-banded rhyolite and syenite dikes near Mount LaCasse, and 178.20 ± 0.40 Ma for a plagioclase porphyritic felsic dike that crosscuts mineralization in the Paramount Zone within the deposit (unpublished U-Pb dating by Richard Friedman, University of British Columbia; unpublished U-Pb dating by Jim Crowley, Boise State University). The contacts of the Yehiniko Pluton with the Hickman Pluton to the south, and the Nighout Pluton to the north, are obscured by colluvium and alluvium.

The Nightout Pluton consists of hornblende biotite granodiorite, quartz diorite, and quartz monzodiorite and is compositionally similar to the outer phase of the Hickman Pluton (Holbek, 1988). Magmatic foliation and coarse grained poikilitic K-feldspar grains are common throughout much of the pluton (Brown et al., 1995). K-Ar and dating of samples from the pluton yielded ages including 236 ± 9 Ma for biotite and 228 ± 8 for hornblende, whereas Rb-Sr dating of biotite yielded an age of 232 ± 5 Ma (Holbek, 1988).

The Loon Lake Stock occurs to the southeast of the Hickman Batholith, on the east side of Mess Creek Valley, and is approximately 15 km long and 2 km to 3 km wide. The composition of the stock ranges from plagioclase-hornblende porphyritic monzonite to plagioclase porphyritic diorite and fine grained diorite. To the west, the Loon Lake Stock is in contact with a plagioclase diorite unit; this diorite may be a border phase of the Stock, although in part the contact appears faulted and the diorite could also be related to the Hickman Pluton (Logan and Drobe, 1992; Logan et al., 2000). No geochronological data is available for the Loon Lake Stock.

7.1.4 Regional Structural and Deformational History

Evidence of several major deformational events ranging in age from Paleozoic to Cenozoic is present in the rocks of the Stikine Terrane and preserved in foliations, folds, faults, and brittle fractures. Several of these deformational events are linked to large-scale tectonic changes and are associated with episodes of orogeny development, exhumation, and erosion that result in stratigraphic unconformities. The structural history of the Mess Creek area, which is approximately coincident with the Project area, is described by Logan et al. (2000), and is summarized below.

Evidence of early deformational events (D1 and D2) is preserved in the Paleozoic rocks of the Stikine Assemblage. The earliest event (D1) is characterized by a northeast-striking, moderately northwest-dipping penetrative foliation (S1) that parallel compositional layering within Early to Middle Devonian volcanic and sedimentary rocks. This S1 foliation fabric is axial-planar to mesoscopic, northeast-verging recumbent isoclinal folds (F1). A later event (D2) deforms and transposes the S1 foliation in Devonian rocks and also appears prominently as a foliation fabric (S2) in Carboniferous to Early Permian rocks. The D2 event is associated with southeast plunging, tight to isoclinal, inclined to recumbent folds (F2).

Numerous shear zones and thrust faults developed during D1 and/or D2, within Paleozoic strata that are parallel to compositional layering or that are subparallel to layering and west-dipping. The direction of this faulting and shearing was top plate to the east (Logan et al., 2000; Holbek, 1988). Low-angle faults within the Schaft Creek deposit area, such as the Saddle Fault and the Basal Fault, may represent reactivations of these Paleozoic faults that have projected upwards into the overlying Mesozoic rocks. Elsewhere in the region, similar shallowly west-dipping thrust faults are observed that do not crosscut the Cretaceous Sustut Group, and so this faulting is interpreted to predate the Cretaceous (Logan et al., 2000).

The D1 and D2 deformational events record the Paleozoic development of the Stikine Assemblage. Both events are associated with compressional deformation evidenced by tight to isoclinal folding and thrust faulting, and both events are also associated with greenschist-grade metamorphism. The timing of these two deformational events bounds the emplacement of the Late Devonian Forrest Kerr and More Creek Plutons. Cooling ages obtained from the More Creek Pluton of approximately 330 Ma are interpreted to represent the timing of uplift and unroofing of this pluton, prior to the nonconformable deposition of Mid-Carboniferous to Early Permian carbonates that overlie the pluton. Thus, the Devonian rocks of the Stikine Assemblage were exhumed and subsequently reburied during the Late Paleozoic. The uppermost member of the Stikine Assemblage is an Early Permian carbonate comprised of medium to thickly bedded micritic limestone with local patch reefs that preserve corals in growth positions, indicating a relatively shallow water depositional environment during the Early Permian (Logan et al., 2000).

Following this Paleozoic development of the Stikine Assemblage, the Late Permian to Middle Triassic is marked by a period of non-deposition, or non-preservation of strata, throughout much of Stikine Terrane. However, in some areas, such as in the vicinity of the Copper Canyon deposit east of Galore Creek, restricted bands of Middle Triassic thinly bedded chert and siliciclastic rocks are preserved. Elsewhere, such as the Tulsequah and Telegraph Creek areas, this Permian to Triassic period is associated with metamorphism, deformation, and uplift. These features have been linked together as part of an orogenic event that has been named the Tahltanian Orogeny. (Souther, 1972; Brown et al., 1996; Logan et al., 2000). Within the Project area, the Tahltanian Orogeny is marked by a disconformity or unconformity between the Permian rocks of the Stikine Assemblage and the overlying Late Triassic Rocks of the Stuhini Group. Based upon foliation fabrics and folds preserved in Paleozoic and Triassic rocks, the timing of the Tahltanian Orogeny is constrained to after the D2 deformation event and prior to the D3 deformation event (Logan et al., 2000).

Evidence of a third deformational event (D3) is preserved in the Paleozoic and Late Triassic rocks surrounding the Project area. Within Paleozoic rocks, this event is characterized by a crenulation cleavage (S3) and open to tight crenulation folds (F3). Within Late Triassic Stuhini Group, this event is associated with northwest-trending open folds (F3) that are upright or sometimes overturned. These folds in the Stuhini Group range in size from outcrop scale to several kilometres in amplitude, with bedding angles typically dipping steeply to the northeast and southwest (Logan et al., 2000). The timing of this deformational event is not well constrained, but the folding that occurs within the Stuhini Group is not present in the overlying Early Jurassic Hazelton Group. The D3 deformational event is therefore interpreted to have occurred during the end of the Late Triassic to Early Jurassic.

An angular unconformity, or in places a non-depositional disconformity, occurs between the Stuhini Group and the overlying Hazelton Group. This unconformity is apparent within the Project area near the top of Mount Hicks (described below) and is also observed in the surrounding region. Other localities displaying this unconformity include the Kerr-Sulpherets-Mitchell District to the south (Nelson and Kyba, 2013), near Yehinko Lake to the northeast (Brown and Greig, 1990; Brown et al., 1996), and near the GJ deposit to the northwest (Ash et al., 1997; mapping by Teck, 2011–2013). In all of these areas, as well as at other locations in northwestern British Columbia, this unconformity is spatially related to Late Triassic to Early Jurassic mineral deposits and occurrences. The presence of this unconformity has been suggested to be evidence of reactivation of faults, uplift, and erosion occurring broadly coeval with the timing of mineralization in these areas (Nelson and Kyba, 2014).

Evidence for a fourth deformational event (D4) is apparent in Late Triassic and Early Jurassic rocks surrounding the Project area. This deformational event is characteristically associated with northwest-striking, northeast-verging folds (F4) that are upright to overturned. These folds range in amplitude from outcrop-size to several kilometres. Some north-trending faults within the Project area may also be associated with this deformational event. Folds related to this deformational event occur in both the Stuhini group and the Hazelton Group, and as a result, the age of this event is constrained to be Middle Jurassic or younger. This deformation is interpreted to be associated with the accretion of the Stikine Terrane onto the margin of North America during the Middle to Late Jurassic.

Evidence for a fifth deformational event (D5) is widely apparent in the Project area within the Stuhini and Hazelton Groups, as well as to the east within the Bowser Lake Group. This event is characterized by north-trending, moderate to open upright folds with wavelengths ranging from outcrop scale to several kilometres. This phase of folding accommodates most of the shortening observed in Mesozoic rocks, particularly in the Bowser Basin. This deformational event is correlative with the Cretaceous-aged Skeena Fold Belt (Logan et al., 2000). The Skeena Fold Belt is a thin- and thick-skinned fold and thrust belt that is present throughout much of British Columbia. The Skeena Fold Belt accommodates northeastward shortening and is associated with dextral strike-slip faulting. Within the Project area, there are numerous southwest-striking, northwest-dipping dextral faults that are associated with prominent linear creeks and gullies. These dextral faults are interpreted to have formed during the D5 deformational event. Some of these southwest-trending structures were subsequently reactivated during the Tertiary to accommodate extensional normal faulting and additional dextral translation (Logan et al., 2000).

Large-scale, north-trending fault zones, such as the Mess Creek Fault and Forrest Kerr Fault, are another important structural feature within the region. These fault zones are commonly several hundred metres wide and contain numerous smaller faults within this width. These smaller faults commonly have contradictory offsets, indicating a history of repeated movement along these zones, with different faults active at different times. Other north-south trending valleys, such as the Skeeter Lake valley and the Schaft Creek valley, are interpreted to contain subsidiary structures that are related to the Mess Creek Fault. The Schaft Creek valley has a similar north-trending orientation and is interpreted to be a subsidiary fault structure. Within the Schaft Creek deposit, breccia bodies, porphyritic dikes, and sheeted quartz-sulphide veins all share a similar north-striking, steeply east-dipping orientation.

The age of these north-trending faults is poorly constrained. Some workers have interpreted these to be ancient faults that originated during the Paleozoic, and which subsequently shaped the tectonic evolution of the region (J. Nelson, personal communication). More recent movement on these faults is apparent at several locations within the surrounding region. To the west of the Hickman Batholith, the north-trending Scud Glacier Fault juxtaposes Permian rocks of the Stikine Assemblage against rocks of the Stuhini Group, implying more than 1,200 m of east-side down displacement that postdates the Late Triassic (Brown et al., 1996). To the south, the Forrest Kerr Fault juxtaposes 2,000 m of Middle Jurassic pillow basalts against rocks representing the Stikine Assemblage and the Stuhini Group, implying more than 2,000 m of east-side down displacement that postdates the Middle Jurassic (Logan et al., 2000). East of the Project area, the Mess Creek fault juxtaposes Late Tertiary to Quaternary rocks of the Mount Edziza volcanic complex against rocks representing the Stuhini Group, indicating east-side down displacement that occurred as recently as 1,340 years ago (Souther 1970; Logan et al., 2000).

7.1.5 Regional Metallogeny

The Schaft Creek deposit is located within a geological and metallogenic domain called the Stikine Arch. The Stikine Arch is located on the north to northwest margin of the Bowser Basin. Within this region, the Stikine Terrane is intruded by numerous Late Triassic to Jurassic plutons, which comprise several major magmatic suites. Several of these magmatic suites are associated with Cu-Au ± Mo ± Ag porphyry- and epithermal-style mineralization, and these styles of mineralization are widespread throughout the Stikine Arch.

The Stikine Arch contains several major undeveloped Cu-Au ± Mo ± Ag mineral deposits (Schaft Creek / Galore Creek) and numerous mineral occurrences. The region hosts an operating mine, Red Chris, a joint venture of Newcrest and Imperial Metals, and several historical mines, including Eskay Creek, Granduc, Premier, Golden Bear, and other smaller producers. Numerous smaller porphyry and epithermal occurrences have been explored by mining companies and junior explorers in recent years.

7.2 Local Geology

7.2.1 Lithology

The Project area is predominantly underlain by rocks representing two major lithological domains: the Hickman Batholith and the Stuhini Group volcanic rocks (Figure 7-4). The boundary between these two lithological domains is generally obscured by alluvial and colluvial cover in the Schaft Creek valley. West and northwest of Mount LaCasse, the boundary between the batholith and the adjacent volcanic rocks manifests as a steeply east-dipping intrusive contact that is faulted in some places. Within the deposit area, the mineralization straddles the boundary between the batholith and the volcanic rocks, but the orientation of this boundary at depth is not well constrained.

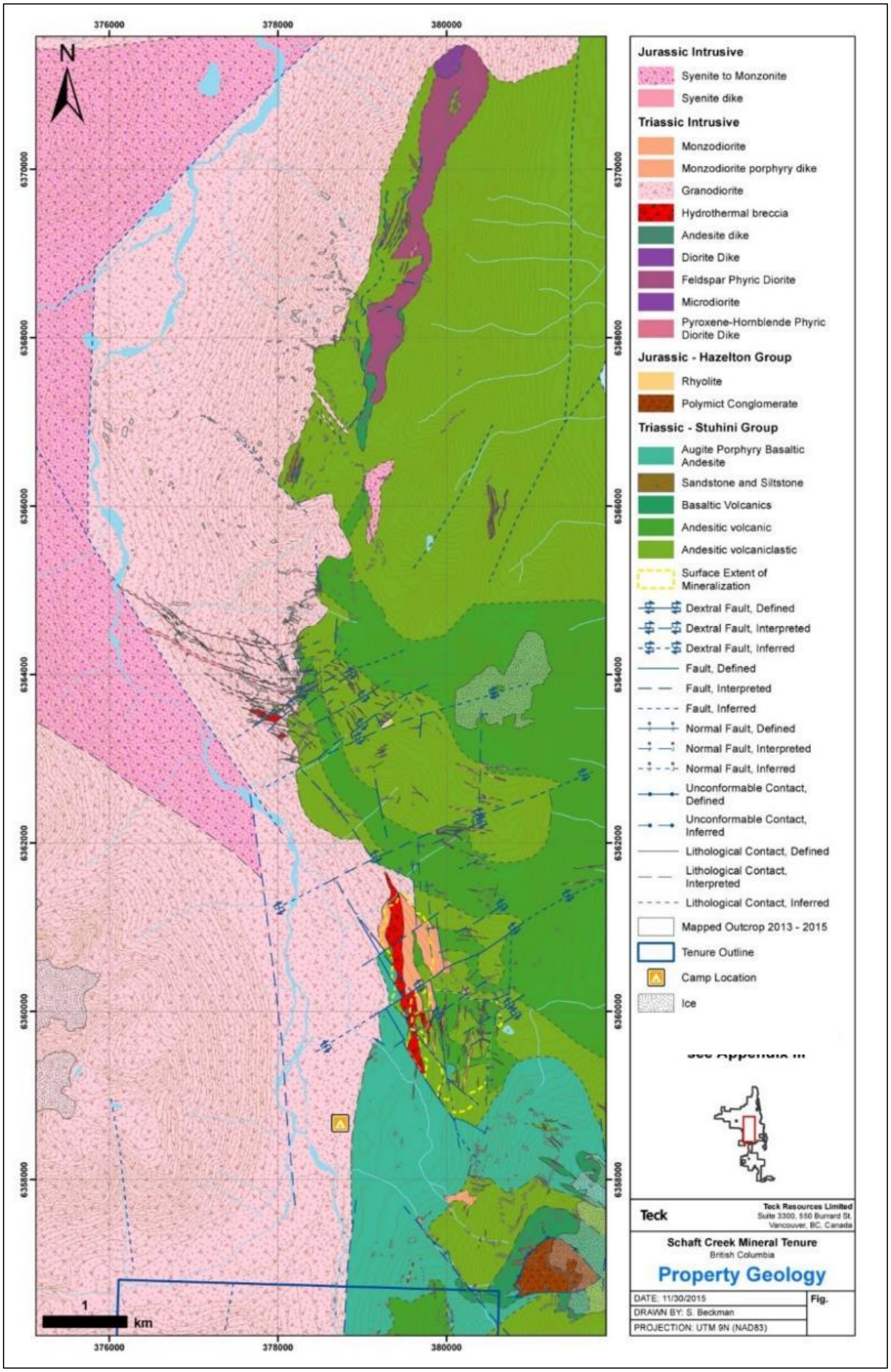
Rock types of the Hickman Batholith that occur within the Project area include, from oldest to youngest, ultramafic dunite and clinopyroxenite, hornblendite, equigranular to weakly porphyritic granodiorite to granite, porphyritic quartz monzonite to monzodiorite, and various feldspar-quartz porphyritic monzodiorite dikes. Crosscutting relationships between all of these intrusive phases have been observed either in drill core or in outcrops. North of the deposit, on the slopes of Mount LaCasse, syenite to rhyolite dikes and sills occur that are correlative with the Yehiniko pluton. The Yehiniko pluton is comprised of buff to salmon-colored monzonite to syenite.

Rock types of the Stuhini Group that occur within the Project area include basaltic to andesitic volcaniclastic tuffs and breccias, augite and plagioclase phyric coherent volcanic flows, and augite-phyric subvolcanic dikes and gabbroic sills. These Stuhini Group rocks are correlative with regional lithological units uTSvt, uTSv, and uTSvp (Logan and Drobe, 1993; Logan et al., 2000). The strike and dip of these units are not readily apparent within the deposit area due to intense alteration and a lack of marker horizons. Regionally, these rocks generally dip either to the northeast or to the southwest. This contrast in dipping orientations is due to either large-scale folding, fault block rotation, or a combination of both.

A rock type of unknown affinity crops out to the southeast of the summit of Mount LaCasse. In this area, coarse-grained hornblende ± plagioclase phyric volcanic flows (or subvolcanic sills) occur as subcrop and talus over an area of several hundred square metres. Near the periphery of the outcropping area, this unit appears to be conformable with underlying augite-phyric volcanic rocks of the Stuhini Group. The location of this rock type correlates with units “pl” and “h” mapped by Logan and Drobe (1993). The stratigraphic correlation of this unit is uncertain, but the unit may be either a stratigraphically higher part of the Stuhini Group or an isolated volcanic unit of the Hazelton Group.

Major mappable units within the deposit area are described below. The rock types are listed in order from interpreted oldest to youngest, although relative and absolute ages are not well constrained for some of these rock types.

Figure 7-4: Property Geology Map



7.2.1.1 Late Triassic Volcanic and Subvolcanic Rock Types

Coherent Andesite (Unit AN): Green to brown to grey, coherent andesite to basaltic andesite flows, and possibly subvolcanic sills. This unit includes variants that are augite-phyric, plagioclase-phryic, and augite-plagioclase-phyric. Augite and plagioclase phenocrysts are 1 mm to 5 mm in diameter, and phenocryst density ranges from sparse to crowded. This rock type is typically weakly magnetic, and locally contains calcite- or chlorite-filled amygdules. This unit is interpreted as the oldest rock exposed in the deposit area and is correlative with the unit uTSv of Logan and Drobe (1993). Locally, this unit is difficult to distinguish from the andesitic volcaniclastic (Unit vAN) described below.

Andesitic Volcaniclastic Rocks (Unit vAN): Green to grey-brown to purple crystal tuff, lapilli tuff, and breccia. These rock types are interpreted as reworked pyroclastic deposits, deposited in a submarine setting. Crystal tuff, lapilli tuff, and breccia facies are commonly interbedded, without obvious marker horizons or consistently mappable grain-size variations. This unit is commonly monolithic, but locally contains clasts of crystalline andesite that are interpreted to be part of Unit AN described above. Crystal tuff and lapilli tuff facies both contain broken crystals of augite and plagioclase 1 mm to 2 mm in diameter, along with glassy lapilli fragments up to 1 cm in diameter. Angular to sub-angular breccia fragments occur in some areas, with diameters up to 10 cm observed in drill core. These breccia fragments commonly contain intense epidote alteration that does not occur in the surrounding matrix. Locally, this unit is interbedded with lenses of brown to grey epiclastic volcanic siltstone, sandstone, and conglomerate, similar to Unit SST described below. This unit appears to be interlayered with Unit AN, although some of this interlayering is suspected to reflect structural imbrication and stratigraphic repetition. This unit is correlative with the unit uTSvt of Logan and Drobe (1993).

This volcaniclastic unit can be difficult to distinguish from coherent andesite flows. Typically, sedimentary textures are used to distinguish volcaniclastic andesite (Unit vAN) from coherent andesite (Unit AN). Specifically, bedding, lamination, normal and reverse graded beds, and lapilli or breccia, or a distinctive ‘grainy’ weathering surface are often used to distinguish Unit vAN from Unit AN. In some cases, it is extremely difficult to differentiate these units during outcrop mapping, and better classification can sometimes be made from drill core or hand samples that have been cut and polished.

Augite-Phyric Basaltic Andesite (bAN): Grey to green, plagioclase-augite ± olivine-phyric basaltic andesite. Augite and plagioclase phenocrysts are 1 mm to 3 mm in diameter and vary from moderately crowded to extremely crowded, with a cumulate texture in some areas. The groundmass for this unit is a glassy, dark green, fine-grained to aphanitic. The genesis of this unit is ambiguous: Near the Schaft Creek deposit, this unit is inferred from drill core intersections to be a subvolcanic sill, although contact relationships with Unit AN and vAN are rarely intact due to hydrothermal alteration and fault deformation. Near Mount Hicks, this unit is interpreted from mapping to be a volcanic unit with local evidence for flow-top brecciation, pillow textures, and conformable stratigraphic relationships with Units BAS and SST. Additionally, this unit has an extremely distinct geochemical signature with high Mg, Ni, and Cr relative to other nearby rock types. Overall, we have thus far been unable to differentiate a subvolcanic component from a volcanic component, and the main identifying characteristic of this unit is the crowded augite-phyric texture and the unusual geochemistry. Historically, this unit was mapped as ‘augite porphyry’ by Salazar (1978), and Logan and Drobe (1993) and included this rock type within unit uTSv.

Olivine-Phyric Basalt (Unit BAS): Grey to green, aphanitic to plagioclase-olivine-phyric basalt. This unit is locally scoriaceous and vesicular, and flow-top breccia textures are apparent in some areas. Vesicles, typically 1 mm to 5 mm in diameter, are filled with either calcite or zeolite. Pillow textures indicative of submarine volcanic deposition occur in several areas near Mount Hicks and Mount LaCasse. The pillows are 0.6 m to 1.5 m in length and with dark halos 1 cm to 2 cm thick. These pillows are commonly altered to epidote or hematite, with local disseminated pyrite. Mapping at several locations during 2015 indicates that the pillows are oriented right-way up. This unit is included within unit uTSv of Logan and Drobe (1993). Near Mount Hicks, the transition between this unit and Unit bAN appears to be gradational, and the rocks near the transition are fine-grained and difficult to categorize.

Plagioclase-Phyric Andesite to Diorite (pAN): Green to grey, plagioclase-phyric andesite to diorite. Plagioclase phenocrysts vary in both abundance and size (commonly 3 mm to 4 mm in length) and are hosted in an aphanitic green groundmass. The composition of this rock type is more dioritic adjacent to the contact with Unit vAN, but away from this contact, this rock type appears to be more andesitic. The dioritic variation is crowded with plagioclase and hornblende crystals. Previous mapping by Logan and Drobe (1993) incorporated this rock type into unit uTSv in some locations, whereas in other locations, it was mapped as plagioclase porphyry. We interpret this andesitic to dioritic unit to be a subvolcanic sill or dike complex, possibly coeval with Unit dDIO described below.

Volcanic Sandstone / Siltstone / Conglomerate (Unit SST): Brown to grey, epiclastic volcanic siltstone, very fine- to coarse-grained sandstone, and conglomerate. Siltstone and sandstone facies are laminated to bedded with beds ranging from 5 cm to over 10 cm thick. Conglomerate clasts are 5 cm to 20 cm in diameter, moderately sorted, and sub-angular. Other sedimentary structures observed locally, include ripple cross-lamination in sandstone and rip-ups of sandstone at the base of the conglomerate units. This rock type occurs interbedded with Unit vAN; this interbedding is particularly apparent on the northeastern and eastern ridges of Mount LaCasse. This unit is included in Logan and Brown's (1997) uTSvt unit as "minor bedded epiclastics".

Diorite Dike (Unit dDIO): Grey to green, plagioclase- and pyroxene-phyric diorite dike. Plagioclase phenocrysts are up to 5 mm long. The width of this unit is typically 1 m to 5 m. Trachytic textures are locally apparent. North of Mount LaCasse, this rock type is observed to occur as a dike that crosscuts interbedded sandstone and volcaniclastic rocks (SST and vAN). Nearby, this unit is observed to transition from a crosscutting dike into a bedding parallel sill within this interbedded sequence. This unit is interpreted to be coeval with Unit pAN described above.

Microdiorite (Unit mDIO): Grey, equigranular, fine-grained, plagioclase- and pyroxene-phyric diorite with fine-grained to aphanitic groundmass. This rock type occurs on the northwestern side of Mount LaCasse, near the limit of the recent mapping by the Schaft Creek JV. This unit may be correlative with unit uTpd mapped by Logan and Drobe (1993). The age of this intrusive unit is not known.

Hornblende- and Pyroxene-phyric Diorite Dike (Unit dhpDIO): Grey to light grey, hornblende-phyric ± pyroxene-phyric diorite. Hornblende phenocrysts are up to 2 cm long, and 4 cm to 5 cm wide. Pyroxene phenocrysts range from 5 mm to 25 mm in length. Locally, phenocryst size decreases with proximity to intrusive contacts. This unit intrudes Units AN and vAN.

7.2.1.2 Late Triassic Intrusive Rock Types

Granodiorite (Unit GDR): Pink and grey, medium-grained, equigranular, hornblende-biotite granodiorite to granite. Characteristically, this rock type has a crowded, cumulate-like, equigranular texture that distinguishes it from other porphyritic intrusive phases. Locally, this rock type contains xenoliths of andesitic volcanic rocks (Unit AN or vAN). This rock type is the oldest phase of the Hickman Batholith within the deposit area. Zircon grains from this unit have yielded U-Pb ages of ~221.7-220.3 Ma. This rock type is crosscut by Unit QMZ, described below, and is interpreted to be pre-mineral.

Monzonite (Unit MONZ): Grey, feldspar-phyric monzonite. This unit has crowded feldspar phenocrysts 1 mm to 3 mm in length, and smaller hornblende and biotite phenocrysts ~2% in a fine-grained groundmass. This unit is more porphyritic near the intrusive contact and more equigranular away from the margins. In addition, mafic phenocrysts increase (up to 10%) adjacent to the intrusive contact. This unit was not mapped by Logan and Drobe (1993), but may be correlative with either the Late Triassic Hickman Pluton or the Early Jurassic Yehiniko Pluton.

Quartz Monzonite (Unit QMZ): Pale pink and grey, medium-grained, equigranular to slightly porphyritic, feldspar-hornblende-quartz phyric monzodiorite to monzonite. Characteristically, this rock unit contains 1% to 5% rounded quartz phenocrysts. Locally, hornblende and feldspar phenocrysts display a trachytic texture. This unit is observed to crosscut Unit GDR and contains xenoliths of the granodiorite. This rock unit is interpreted to be pre-mineral or early-syn-mineral, as it is commonly proximal to mineralization and crosscut by quartz-sulphide veins, particularly in the Paramount Zone. Zircon grains from this rock unit have yielded U-Pb ages of ~220.9 Ma to 220.3 Ma.

Regionally, the quartz monzonite unit may be difficult to distinguish from Unit GDR, particularly in areas with weathered outcrop or limited exposure. More confident identification can sometimes be made from drill core or hand samples that have been cut and polished. Staining using hydrofluoric acid and sodium cobaltinitrate is also useful for determining the amount of K-feldspar and hematite in these rock types.

Quartz Monzodiorite Dikes (Unit sPOR): Pink to buff-grey, fine- to medium-grained, porphyritic, feldspar-hornblende-quartz phyric monzonite to monzodiorite. Characteristically, this rock unit has a crowded porphyritic texture with an aphanitic to fine-grained groundmass. This unit commonly occurs as narrow, 1 m to 30 m thick, porphyritic dikes that vary in thickness along strike and down dip. These dikes are observed to crosscut Units GDR and QMZ. In some locations, these dikes crosscut hydrothermal breccias, but this is not a consistent relationship. This rock type is interpreted to be syn-mineral because it has a close spatial association with sulphide mineralization, potassic alteration, and zones containing quartz-sulphide vein stockworks. Zircon grains from this rock unit have yielded U-Pb ages of ~219.4 Ma to 219.3 Ma, as well as older grains with ages of ~220.5 Ma to 220.1 Ma. These older grains are interpreted to be antecrysts inherited from the Hickman Batholith.

In previous years, efforts were made to differentiate this unit into separate dike variants based upon phenocryst composition. However, these compositional differences could not be consistently mapped between drill holes and between outcrops, and so all variants of syn-mineral monzodioritic dikes have been combined into Unit sPOR.

Intrusive Breccia (Units IBX1 and IBX2): Tan, grey, and pink, polylithic or locally monolithic breccia. Generally this unit is matrix-supported, with a coherent (igneous) matrix of quartz-feldspar ± hornblende ± biotite. Breccia cement typically consists of quartz ± sulphides, but also locally includes calcite and anhydrite. Clasts are typically subround to subangular and 1 cm to 10 cm in diameter. Clast types include Units QMZ and sPOR. In some locations, clasts have been observed to contain quartz-sulphide veins with potassic alteration that pre-dates brecciation. This unit is interpreted to be syn-mineral because it contains sulphide-bearing cement, and because it is closely associated with zones of quartz-sulphide vein stockwork. In some areas, this unit is associated with autobrecciation textures and a gradational contact into Unit sPOR, suggesting that these two units are genetically linked.

Hydrothermal Breccia (Units cHBX1, cHBX2, cHBX3, cHBX4, cHBX5, cHBX6, cHBX7): Tan, green, and pink, polylithic or locally monolithic breccia. Generally clast-supported and lacking a distinct matrix. Several varieties of hydrothermal breccia have been differentiated based upon their cement composition, including the following: quartz-epidote-chlorite-chalcopyrite ± molybdenum ± bornite cement (cHBX1); quartz-biotite-K-feldspar ± magnetite ± chalcopyrite (cHBX2); tourmaline-quartz-carbonate ± chalcopyrite ± bornite ± molybdenum (cHBX3); pyrite-carbonate-chlorite (cHBX4); feldspar-anhydrite ± quartz ± chlorite ± chalcopyrite ± bornite ± pyrite (cHBX5); sericite-quartz-pyrite-chlorite (cHBX6); and chlorite-actinolite-calcite ± tourmaline ± chalcopyrite-pyrite. Clasts in these breccias are generally subrounded to angular, locally with a jigsaw texture, and are 1 cm to 20 cm in diameter. The clasts include a variety of rock types, including Units AN, vAN, QMZ, and sPOR. As with the intrusive breccia unit, in some locations, this unit contains clasts containing truncated quartz-sulphide veins. Locally, there appears to be a gradational contact between hydrothermal breccia and intrusive breccia, and these two units are interpreted to be approximately coeval. Spatial and genetic relationships between the different breccia cement types are not well understood, and more work is required to resolve the distribution of these hydrothermal breccias.

7.2.1.3 Early Jurassic Sedimentary Rock Types

Polymictic Conglomerate and Volcanic Sandstone (Unit CGL): This unit consists of interlayered polymicitic conglomerate and volcanic sandstone. The sandstone is purple-pink with medium-grained quartz and feldspar broken crystals that are interpreted to be reworked tuffaceous material. The sandstone contains sub-angular fragments of basaltic andesite and volcanic sandstone that range from 1 cm to 10 cm in diameter. The polymicitic conglomerate is a conspicuous, purple-colored, pebble to boulder clast-supported to matrix-supported conglomerate with subrounded to subangular clasts. Clasts include rhyolite, basaltic scoria, and plagioclase-phyric andesite. The matrix is comprised of red-colored, fine-grained sand to mud-sized material. Near Mount Hicks, the conglomerate and the sandstone are interlayered, and both form beds that dip to the southeast, unconformably overlying the andesitic volcanic rocks of the Stuhini Group. Upsection, the layers of volcanic sandstone become thicker and more abundant than the layers of conglomerate. This unit was mapped as Unit lJcg by Logan and Drobe (1993).

7.2.1.4 Early Jurassic Intrusive Rock Types

Plagioclase Diorite Dikes (Unit pPOR): Green-grey, fine-grained, sparsely porphyritic, plagioclase-phyric diorite dikes, typically 1 m to 10 m thick. This unit crosscuts all the rock types described above, and also crosscuts quartz-sulphide vein stockworks and potassic alteration. This unit typically contains chlorite-epidote alteration and trace pyrite ± chalcopyrite mineralization. This rock unit is therefore interpreted to be post-mineral with respect to the main porphyry mineralization event at Schaft Creek. Zircon grains from this rock unit have yielded a U-Pb age of ~178.2 Ma, as well as an older U-Pb age of ~221.6 Ma. This older age is interpreted to be caused by antecrysts inherited from the Hickman Batholith.

Basaltic Andesite Dikes (Units dAN and dANp): Green to grey-green, fine-grained, plagioclase- and pyroxene-phyric basaltic andesite dikes, typically 1 m to 10 m thick. This unit has several characteristic features including chilled margins; calcite amygdules; and abundant, fine-grained magnetite. In the field, this unit forms blocky, resistive, rounded outcrops. This unit crosscuts all of the rock types described above and is not typically observed to host any significant sulphide mineralization. An absolute age has not yet been determined for this unit.

Syenite to Rhyolite Dikes (Unit dSY): Pink to buff, fine-grained, syenite to rhyolite dikes, typically 1 m to 10 m thick. Commonly quartz- or plagioclase-phyric and occasionally has an aphanitic texture. Flow banding and chill margins are a common feature of this unit, and in some instances, the banded appearance makes this rock type look similar to a bedded volcanic or sedimentary rock. This rock type crops out commonly on the southwest side of Mount LaCasse, and talus from this unit form rusty red talus fields that can appear similar to a hematite gossan from a distance. This unit also occurs commonly within the LaCasse Zone and crosscuts sulphide mineralization in this area. Zircon grains from a syentic dike yielded a U-Pb age of ~177.2 Ma, whereas grains from a rhyolitic dike yielded two U-Pb ages: a younger age of ~178.8 Ma and an older grain population with an age ranging from 221.3 Ma to 220.6 Ma. These older grains are interpreted to be antecrysts inherited from the Hickman batholith. The 178 Ma to 177 Ma ages are similar to the Unit pPOR described above and also similar to the Yehiniko Pluton, which is located to the west.

7.2.2 Structure

Major structures in the resource area that either truncate or offset mineralization, or are considered potentially significant geotechnical domains, are described below. The structures are described in order from interpreted oldest to youngest.

The **Basal Fault** is interpreted to be one of the oldest faults in the deposit area, based upon crosscutting relationships interpreted from drill core. The fault is an important boundary for mineralization in the Liard Zone and is interpreted to be late-mineral or post-mineral. The fault dips gently to the northwest and consists of a narrow sheared or cataclastic zone, sometimes with several separate anastomosing fault strands. These anastomosing strands commonly enclose slivers of mineralized rock within the fault zone. The anastomosing nature of the fault strands, the low angle of the fault, and other field relationships suggest that the Basal Fault is most likely a thrust fault, although other interpretations are possible. Currently, it is not apparent if this fault extends under the West Breccia Zone or under the Paramount Zone; drill holes in these areas are generally not deep enough to intersect the modelled position of this fault. The Basal Fault is interpreted to daylight south of the Liard Zone.

Work completed by the Schaft Creek JV during 2015 has highlighted the possibility of mineralization in the footwall of the Basal Fault, and to the south of the Liard Zone. Additionally, interpretation of geophysical data from the Wolverine Creek area suggests that there may be another flat-lying structure further south at depth; this could be an additional, lower, sub-parallel fault to the Basal Fault.

The **Saddle Fault** is interpreted to be a splay off of the Basal Fault and is an important boundary for mineralization on the north side of the Liard Zone. The fault dips moderately to the north and consists of a broken, fractured, or sheared zone. Late movement on this fault is interpreted to have normal offset based upon geological mapping observations, but it is not clear if the fault has an earlier movement history with reverse offset. The Saddle Fault has been modelled to truncate against the Paramount Fault, and it is not clear if the Saddle Fault extends further westward.

The **Breccia Footwall Fault** and the **Silica Fault** are important boundaries for mineralization in the West Breccia Zone. These faults strike to the northwest and dip steeply to the east. The offset of these faults is not known with certainty, but the Silica Fault has been modelled to have an apparent sinistral offset of approximately 300 m. The faults are inferred to be related because of their similar orientation, although their appearance can be different locally. The Breccia Footwall Fault has broken, fractured, gouged, and sheared textures, and is typically 5 m to 10 m wide. In contrast, the Silica Fault has broken, gouged, sheared, or healed textures, but is typically much wider, with a fault damage zone approximately 30 m to 50 m thick that contains several fault strands. The large damage zone of the Silica Fault is potentially an important geotechnical consideration.

The **Paramount Fault** and the **Snipe Fault** do not appear to be major boundaries for mineralization, but both faults have large damage zones that are potentially important geotechnical features. These faults strike approximately north, and dip vertical or steeply to the east. These faults are interpreted to be younger than the four faults described above, although this is an observation from geological mapping that is difficult to confirm in drill core. These faults contain broken, fractured, and gouged textures, with fault damage zones that are 10 m to 20 m thick. To the east of the Paramount Zone, the Paramount Fault has been modelled as a single structure. However, widely spaced drilling in this area has intersected a large amount of faults, and there may be several other faults that are sub-parallel to the Paramount Fault. This area is the location of the highwall for the proposed open pit, and the large damage zones of faults in this area are an important geotechnical consideration.

The **Northeast Fault** and the **Mike Fault** are the youngest faults in the deposit area that have been modelled. These faults are associated with northeast trending gullies that occur on the slopes of Mount LaCasse. There are numerous northeast trending faults throughout the deposit area, although many of these are too small to warrant modelling. These northeast trending faults contain broken and fractured damage zones, and are typically 5 m to 20 m thick and may be important for geotechnical consideration in some areas. These faults dip steeply to the north and have an apparent dextral sense of movement, although a component of normal movement is also inferred. These northeast faults do not appear to be major boundaries for mineralization.

7.2.3 Alteration

Hydrothermal alteration in the Project area includes a variety of mineral assemblages. These mineral assemblages are interpreted to be coprecipitated and to be representative of specific factors including pressure, temperature, pH, fluid composition, and in some instances, wall rock composition. Many of these hydrothermal alteration assemblages are spatially associated with sulphide mineralization, either as proximal assemblages or distal assemblages. The zonation pattern of these alteration assemblages can be a useful tool for locating areas of sulphide mineralization, although consideration must be given to post-mineral faults that disrupt the alteration zonation. Similar alteration assemblages throughout the property suggest a shared origin for sulphide-mineralized zones at Schaft Creek, Discovery, LaCasse, Grizzly, and other mineralized occurrences throughout the property area. Major alteration assemblages within the Project area are described below.

Early Magnetite Alteration (FEO): This alteration type is characterized by thin, wispy magnetite veinlets that sometimes have K-feldspar selvages as well as minor chalcopyrite and bornite. Mapping and identification of this early magnetite alteration is challenging because much of the magnetite has been subsequently replaced by hematite during a later alteration episode. Nevertheless, there is consistent, sparse evidence of early wispy magnetite/hematite veins throughout the deposit area. There is some evidence to suggest that this early magnetite alteration is best preserved around the fringes of the mineralized zone, but this spatial relationship is equivocal. A rare variant of this alteration assemblage consists of intense magnetite/hematite pseudomorphic alteration of augite phenocrysts within volcanic rocks.

K-Feldspar-Biotite Alteration (Potassic) (kPOT, bPOT, kbPOT): This alteration assemblage is characterized by K-feldspar ± biotite that occurs as a halo around quartz-sulphide veins or, less commonly, as pervasive alteration of a groundmass or matrix. Vein types that have been observed with this halo include quartz-only veins, quartz-molybdenite veins, quartz-chalcopyrite-bornite veins, and quartz-chalcopyrite-pyrite-veins. Rare biotite-only veins with K-feldspar selvages have also been observed locally. Some quartz veins with K-feldspar selvages have irregular to slightly wavy margins that are indicative of vein formation at higher temperatures. Some of these veins can be correlated with the “A vein” and “B vein” terminology that is classically applied at many porphyry deposits. However, other potassic veins at Schaft Creek exhibit sheared textures that do not fit this classic terminology. Within the core of the deposit, veins with K-feldspar alteration halos also commonly have selvages containing green mica. This green mica is inferred to be muscovite, and this mineral appears to be spatially associated with high-grade mineralization. Some other minerals also occur locally as part of the potassic assemblage, including calcite within the veins and epidote within the vein halo. At the LaCasse Zone, epidote is commonly associated with K-feldspar, in an assemblage that has been labeled as Calc-Potassic.

Potassic alteration appears to be the assemblage that is most closely associated with Cu-Au mineralization within the deposit area. Because of this important association, “Weak Potassic” and “Strong Potassic” domains have been differentiated during geological mapping and 3D modeling; the distinction between these domains is based upon intensity of alteration.

Albite-Hematite Alteration (Albitic) (ALB, SOD): This alteration assemblage is characterized by white feldspar (presumed albite) with hematite ± anhydrite ± chlorite. Vein types that have an albitic selvage include quartz ± bornite-chalcopyrite-molybdenite, quartz-albite ± bornite-chalcopyrite-molybdenite, and anhydrite-albite ± bornite-chalcopyrite-molybdenite. Narrow, wispy, anhydrite-only veins also occur in zones of albitic alteration, and these are presumed to be part of the same assemblage. However, there are also other, thicker anhydrite-only veins elsewhere in the deposit area that are interpreted to be paragenetically late. Locally, there is evidence of veins with albite selvages crosscutting veins with potassic selvages; however, this relationship is rarely observed. Overall, the abundance of albite-hematite alteration is difficult to quantify, because this alteration assemblage is difficult to distinguish from K-feldspar alteration or quartz alteration. However, albite alteration is closely associated with some areas of higher-grade Cu-Au mineralization, particularly in some breccia phases within the Paramount Zone.

Quartz Alteration (Silicic) (SI): This alteration assemblage is characterized by fine-grained silicification that is typically fracture-controlled, although selective phenocryst alteration and pervasive alteration also occur locally. Silicic alteration is characteristically associated with tourmaline alteration, which helps to differentiate this assemblage from other feldspar alteration. Vein types that have a silicic selvage include quartz-only, quartz-carbonate ± chlorite, tourmaline ± chlorite, and tourmaline-quartz-carbonate ± chlorite ± bornite ± chalcopyrite ± molybdenite. Silicic alteration and tourmaline-bearing veins are sometimes associated with high Cu-Au grades, particularly in the Paramount and West Breccia Zones, although high grade material in these zones is also associated with other alteration assemblages.

Sericite-Carbonate-Clay Alteration (SER, SCC, PHY): This alteration assemblage is characterized by fine-grained white mica ± trace clay. Vein types that have a phyllic alteration selvage include quartz-only, carbonate ± quartz ± pyrite, pyrite-only, and tourmaline-bearing veins. Some grey sericite-pyrite-quartz veins occur that are broadly analogous to the “D vein” terminology classically used for porphyry deposits; however, veins of this type are rare at Schaft Creek.

Sericite-Albite-Carbonate-Chlorite-Tourmaline (Phyllic) (PHY): This alteration assemblage occurs in patchy zones around the periphery of the Schaft Creek deposit. For example, a “phyllic breccia” with intense sericite-chlorite-pyrite-albite ± tourmaline alteration occurs in the gap between high-grade mineralization at the Paramount Zone and the lower-grade mineralization at the West Breccia Zone. Another large, phyllic alteration zone occurs northeast of the deposit at the Mike target area. This area contains a sericite-pyrite-chlorite assemblage that is coincident with a large IP chargeability high.

Chlorite-Epidote-Carbonate (Sodic Calcic, Propylitic) (CHL, SOC, CAL): This alteration assemblage is characterized by chlorite-epidote ± calcite ± actinolite ± hematite. This alteration is widespread, but the intensity of alteration is generally low. Chlorite alteration of mafic phenocrysts and volcanic groundmass is common throughout the deposit area. Vein types that have a chlorite-epidote halo include quartz-carbonate-chlorite ± chalcopyrite ± pyrite, chlorite-only, and calcite-only. Rare magnetite-quartz-carbonate-chalcopyrite-pyrite veins are observed in chlorite-altered areas within the deposit, and these are interpreted to be related to this assemblage. Chlorite-epidote alteration occurs in all rock types within the deposit area, and commonly overprints earlier alteration assemblages such as potassic and sericitic. An assemblage of chlorite-epidote-pyrite appears to be the most distal part of the deposit footprint; this alteration extends several hundred metres beyond the limits of potassic alteration.

Hematite Alteration (HEM): This alteration assemblage consists of hematite ± chlorite ± calcite. This alteration is characterized by red, brown, or purple coloration, and can occur pervasively or selectively along fractures. This alteration is not typically associated with veins, except where it replaces other vein-hosted minerals (e.g., magnetite). This hematite alteration is interpreted to have a relatively late timing and is inferred to be part of the late propylitic alteration that overprints much of the deposit area. Hematite staining of feldspars is also common throughout the deposit area, although this is interpreted to be related to the albite alteration described above.

Purple-colored volcanic rocks containing abundant hematite occur immediately north of the Liard Zone, above the Saddle Fault. Historically, the rocks in this zone have been differentiated by some workers into a separate lithological unit labeled as “purple volcanics”. This area contains no mineralization, and some workers have argued that these purple volcanic rocks are a post-mineral unit that unconformably overlies the deposit. Based upon mapping and extensive relogging completed in 2013 to 2014, the Schaft Creek JV argues that this historical interpretation is incorrect, and that these rocks are part of the Stuhini Group. This area contains no mineralization because the “purple volcanics” are juxtaposed against the deposit by a major fault (the Saddle Fault).

7.2.4 Mineralization

Mineralization at Schaft Creek (Caron et al. (2012) has been described in terms of three separate zones: the Main (or Liard) Zone, the Paramount Zone, and the West Breccia Zone. Although mineralization in the Main Zone and the West Breccia Zone is found in two generally spatially separate areas in the southern portion of the Schaft Creek deposit, the same cannot be said for mineralization found further to the north. There, most mineralization is hosted within structurally prepared and altered breccias, with a lesser amount of mineralization found in proximal volcanic units. It seems clear that mineralization at Schaft Creek can be readily described in terms of an earlier phase of mineralization in the Main Zone and a later phase of mineralization related to breccias and proximal volcanic rocks found along the west margin of the Main Zone and extending to the north.

More details on the mineralization styles specific to the Liard, Paramount, and West Breccia Zones can be found in Section 8.0 Deposit Types.

8.0 DEPOSIT TYPES

The Schaft Creek deposit has been described by many workers as a calc-alkaline Cu-Mo-Au porphyry deposit (Fox et al., 1995; Spilsbury, 1995; Scott et al., 2009; Morrison and Karrei, 2012). Other workers have considered it a shear-hosted, low-sulphidation Cu-Mo-Au-Ag vein deposit (Le Boutillier, 2013). Early mapping assigned the intrusive host rocks of the Schaft Creek deposit to the Early Jurassic (e.g., Logan and Drobe, 1993), but subsequent geochronology work constrained the age of the host rocks to the Late Triassic (Logan et al., 2000; Scott et al., 2009; unpublished U-Pb dating by Richard Friedman, University of British Columbia; unpublished U-Pb dating by Jim Crowley, Boise State University). Interpretation of the deposit is complicated by a lack of outcrop, complex hydrothermal alteration, post-mineral faults, and sparsity of drilling near the fringes of the hydrothermal system.

The deposit has historically been subdivided into two or three distinct mineralized zones, although the boundaries of this subdivision have changed during the history of the Project. These three mineralized zones are named the Liard, Paramount, and West Breccia Zones. Historically, the West Breccia and Liard Zones have been grouped by some workers into a larger domain called the Main Zone. Other workers have grouped the Paramount and West Breccia Zones into a single domain called the Breccia Zone.

The division between these various zones originated during the earliest years of the Project, when the mineral claims overlying the deposit were divided between Paramount (to the north) and Liard (to the south). The old property line between these two companies coincides with a change from predominantly breccia-style mineralization in the Paramount Zone to predominantly vein stockwork-style mineralization in the Liard Zone. Recent work has shown that there is considerable spatial overlap between breccia and vein-hosted mineralization styles in both zones and that brecciation in the Paramount Zone extends southwards into the West Breccia Zone.

For this Technical Report, we retain the basic nomenclature of the Liard, Paramount, and West Breccia Zones to describe the Schaft Creek deposit. We caution that the historical boundaries between these zones were commonly based upon geography, and that there is considerable overlap in mineralogy and texture between these geographical boundaries. The characteristics of each of these three zones, along with the faults and other boundaries that constrain them, are described below.

8.1 Liard Zone

The Liard Zone comprises narrow, porphyritic quartz monzonite to quartz monzodiorite dikes that have been emplaced into andesitic volcanic and volcaniclastic host rocks. The dikes are typically 5 m to 50 m thick, strike north-northwest to north-northeast, and dip steeply to the east. Numerous narrow dikes occur within the eastern part of the Liard Zone, and in this area it can be difficult to trace individual dikes with confidence between drill holes or outcrops. In contrast, a single, thicker “Central Porphyry” dike occurs within the central portion of the Liard Zone.

The porphyritic dikes in the Liard Zone are spatially associated with potassic alteration, increased density of quartz-sulphide veins and vein stockworks, and a zone of elevated Cu-Au grade. The most intense alteration and highest copper grades commonly occur in the host rock immediate adjacent to the porphyry dikes, rather than within the dikes themselves. In some areas, chalcopyrite, bornite, and pyrite all occur disseminated within the host rocks and porphyry dikes, suggesting multiple mineralization episodes that have juxtaposed bornite and pyrite into the same area.

Several types of vein-hosted mineralization are recognized in the Liard Zone. These include (1) Cu-Au-Mo mineralization resulting from quartz-biotite-bornite-chalcopyrite ± hematite veins with associated K-feldspar ± green mica selvages; (2) overprinting Cu-Au mineralization resulting from quartz-chlorite-pyrite-chalcopyrite ± calcite ± epidote ± hematite with associated sericite and chlorite-epidote selvages; and (3) late, Mo mineralization resulting from veins of massive to semi-massive molybdenite with no apparent selvage. No preferred structural trend has been identified for this vein-hosted mineralization in the Liard Zone.

The boundaries of the Liard Zone are defined by faults in most directions. To the north, the mineralization is abruptly truncated by the Saddle Fault, which strikes west and dips moderately to the north. To the south and at depth, the mineralization is abruptly truncated by the Basal Fault, which strikes west and dips shallowly to the north. To the east, mineralization diminishes in proximity to the Snipe Fault, but this fault is not a hard boundary. To the west, mineralization diminishes in the vicinity of the Silica Fault, although the West Breccia Zone occurs to the west of this fault.

8.2 Paramount Zone

The Paramount Zone comprises an elongate, multiphase igneous-hydrothermal breccia body that has been emplaced into quartz monzonite and andesitic volcanic host rocks. The breccia body strikes north-northwest, dips steeply east, is 100 m to 300 m thick, has a strike length of approximately 1,200 m, and extends at least 600 m below surface. High-grade mineralization occurs within the breccia body and also extends 100 m to 200 m into the quartz monzonite hanging wall and, to a lesser extent, into the footwall andesitic volcanic rocks. Mineralization outside of the breccia body is associated with stockwork zones containing quartz-sulphide veins.

Three styles of mineralization occur within the Paramount breccia body, each of which is associated with different breccia cement minerals. These mineralization styles include (1) Cu-Mo mineralization associated with K-feldspar-biotite-quartz-chalcopyrite-molybdenite ± bornite veins and breccias with associated potassic alteration (Unit cHBX2), (2) Cu-Au-Mo mineralization associated with anhydrite-bornite-chalcopyrite ± molybdenite veins with associated albitic alteration (Unit cHBX5), and (3) Cu-Au-Mo mineralization associated with tourmaline-quartz-carbonate-chalcopyrite ± bornite ± molybdenite veins and breccias with associated silicic alteration (Unit cHBX3). All three of these breccia styles include sulphide cement, and the assay grade of sample intervals typically correlate with the amount of sulphide cement present (typically 0.5% to 3%). Locally, there appears to be an association between high-grade mineralization and a dark-colored intrusive breccia phase. The mineralogy of this intrusive breccia appears similar to the syn-mineral sPOR dikes in the Liard Zone, but work is required to confirm the link between these two rock types.

A mineral zonation pattern is apparent around the main breccia body in the Paramount Zone. Potassic alteration intensity, vein density, and vein thickness all increase towards the breccia zone. A clear sulphide zonation (from chalcopyrite > pyrite, to chalcopyrite > bornite, to bornite > chalcopyrite) is apparent outside of the breccia body and extends inwards. No pyrite was observed in bornite-bearing areas, which is in contrast to late pyrite that overprints areas of the Liard Zone. Bornite correlates closely with Au grades, and assay ratios of Cu:Au are an effective vectoring tool towards bornite-rich zones. Molybdenite occurs throughout the Paramount Zone, and does not appear to be a useful mineralization indicator.

The boundaries of mineralization in the Paramount Zone are related to the extent of the Paramount breccia body, and this body is reasonably well-constrained by the current drill pattern, although the breccia remains open at depth. The limits of the breccia are well defined by drilling to the east and west and to a considerable depth. To the north, the breccia appears to pinch out, although this area has only sparse drilling. To the south, the breccia body continues towards the West Breccia Zone; however, there is an important change in breccia mineralogy in the vicinity of 6360250N. In this area, the dominant mineralogy of the breccia changes from quartz-feldspar-sulphide ± chlorite cement (Units cHBX1, cHBX2, cHBX3, cHBX5), to an assemblage comprised of sericite-tourmaline-chlorite-pyrite ± minor chalcopyrite (Unit cHBX6). This domain has been labeled as a “phyllic breccia”, and mineralization in this area is pyrite-rich and low grade. The changes in cement mineralogy throughout the breccia body are not well understood, and more work is required to understand this mineral zonation pattern.

8.3 West Breccia Zone

The West Breccia Zone comprises an elongated, hydrothermal breccia body that has been emplaced into andesitic volcanic and volcaniclastic rocks. The breccia body strikes north-northwest, dips steeply east, is 80 m to 160 m thick, has a strike length of approximately 500 m, and extends at least 200 m below surface. Mineralization is limited to the breccia body and seldom extends far into the adjacent footwall or hanging wall. There are a few narrow monzodiorite dikes in the vicinity of the breccia that appear similar to the monzodiorite dikes seen in the Liard Zone.

The West Breccia Zone is similar to the Paramount Zone breccia and comprises different styles of mineralization associated with certain breccia cement minerals. However, the West Breccia Zone is dominated by low- to medium-temperature breccia mineralogy and lacks the higher temperature assemblages that are observed within the Paramount Zone. The three prominent styles of breccia mineralogy at the West Breccia Zone include (1) Cu-Mo-Au mineralization associated with tourmaline-carbonate-chalcopyrite-pyrite ± molybdenite veins and breccias with associated silicic alteration (Unit cHBX3), (2) Cu-Mo mineralization associated with chlorite-calcite-pyrite veins and breccias with associated propylitic alteration (Unit cHBX4), and (3) high-grade Cu-Mo-Au mineralization associated with chlorite-actinolite-calcite-tourmaline breccia cement (cHBX7). Interestingly, although the West Breccia Zone lacks bornite, assay grades in this area are sometimes very high because of the abundance of sulphide cement (typically 2% to 10%).

The boundaries of the West Breccia Zone are generally poorly constrained. Historical drilling in this area is sparse and shallow. The breccia body remains open to the north and south, and the recent drilling has only identified the breccia to a depth of 160 m to 200 m. There is no evidence that the breccia body pinches out in any of these directions; however, the grade in historical drill holes is inconsistent and the structural controls on the breccia body are poorly understood. To the east, the Silica Fault is interpreted to offset the West Breccia Zone with a sinistral sense of movement, but more work is required to confirm this. To the west, the Breccia Footwall Fault is interpreted to truncate the West Breccia Zone, but more work is required for confirmation.

8.4 Other Mineralized Zones Outside of the Deposit Area

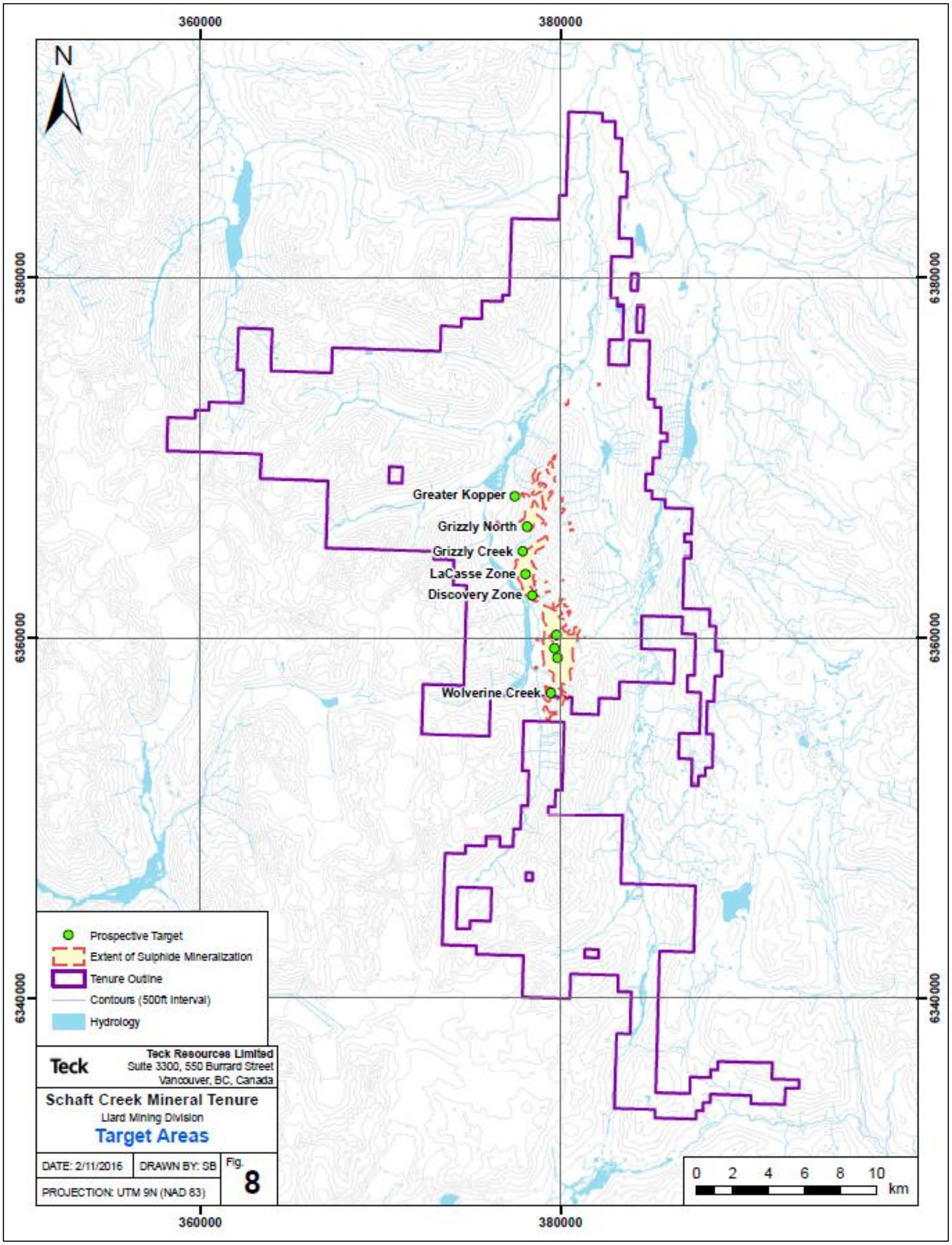
The Schaft Creek deposit is situated within a 12 km long trend of mineralization that occurs along the margin of the Hickman Batholith (Figure 8-1). This trend straddles the contact between intrusive rocks of the batholith and the adjacent volcanic host rocks. The trend was recognized by previous workers including Adera Mining (Lammle, 1966), Phelps Dodge (Phelps and Gutrath, 1972), and Teck (Betmanis, 1978; Raven, 1979; and Holbeck, 1981), and various workers at the Schaft Creek deposit. The history of work along this trend is comprehensively summarized by Greig (2009).

Mineralization along the trend consists of pyrite, chalcopyrite, and occasionally bornite that occur variably as fracture-controlled, shear zone-controlled, or sparsely disseminated. Sulphide mineralization occurs in an area that is typically 100 m wide along the intrusive contact, but locally expands into wider zones that are 200 m to 500 m wide. This sulphide mineralization appears to be continuous along the margin of the batholith, from Grizzly Canyon in the north to Wolverine Creek in the south.

Alteration along the trend consists predominantly of chlorite-epidote alteration of the volcanic host rocks and pink-colored or buff-grey-colored alteration of the intrusive rocks. Previous workers have recognized that the pink-colored alteration results predominantly from hematite rather than from K-feldspar. At the LaCasse Zone, mineralization and alteration are crosscut by syenite to rhyolite dikes that are associated with the Yehiniko Pluton; however, these dikes also contain disseminated chalcopyrite. More work is required to understand if the Yehiniko Pluton is associated with a second, younger mineralization episode during the Late to Middle Jurassic.

There is potential to discover and delineate additional zones of porphyry mineralization along this trend. Much of the trend has only been examined through piecemeal prospecting, soil and rock sampling, and limited geological mapping. Prior to 2015, the only substantial drill test of this trend was conducted at the Discovery Zone by Copper Fox in 2011 to 2012. This drilling returned encouraging results, including sizeable intervals of Cu-Au mineralization associated with intrusive breccias, quartz-sulphide vein stockwork zones, and high temperature potassic and albitic alteration assemblages. There is opportunity to conduct systematic exploration along this trend and to prioritize key targets for additional work. Some of the more interesting target areas that have been identified along this trend are summarized below.

Figure 8-1: Mineralized Corridor and Target Areas



Source: Schaft Creek JV (2016)

8.4.1 LaCasse-Discovery Zone

The LaCasse-Discovery Zone is located 3 km to 5 km north of the Schaft Creek deposit, along the margin of the Hickman Batholith. Early workers recognized a trend of mineralization that extended into this area (e.g., Lamelle, 1966). In the 1980s, an area of "consistent copper mineralization" was documented (Betmanis, 1978; Raven, 1979). Rock chip samples from within this zone included numerous samples with grades of 0.5% to 3.15% Cu. Key geological features mapped include disseminated chalcopyrite, bornite, pyrite, quartz veins, and mineralized breccia zones. In particular, the presence of outcropping breccias is considered to be a favorable indicator of prospectivity, because breccias are intimately associated with mineralization at the Paramount Zone and the West Breccia Zone.

Within the LaCasse area there is a timber platform of unknown age located at 378016E, 6363460N (UTM NAD 83 Zone 9N). At this site, narrow diameter drill casing is observed to be sticking out of the ground, but the depth of this drill hole is unknown, and there is no known record of drill core being recovered from this area.

During 2014, the Schaft Creek JV expanded their geological mapping program to encompass the LaCasse area and to collect additional rock chip samples. This mapping largely confirmed the previous mapping, but added an additional level of detail. The predominant rock type in the mineralized zone consists of quartz monzonite to monzodiorite to granodiorite, with crosscutting andesite and syenite dikes. The mineralized area at LaCasse was delineated as a zone of chalcopyrite > pyrite mineralization approximately 1 km by 1.5 km in size. This area contains patchy zones of K-feldspar-biotite-quartz (potassic assemblage) and hematite-albite-epidote-chlorite (sodic-calcic assemblage), as well as widespread chlorite, epidote, and magnetite. Sulphide mineralogy and zonation are similar to the Schaft Creek deposit, although only trace bornite has been observed at LaCasse.

During 2015, the Schaft Creek JV conducted a drill program at the LaCasse area. A key outcome of this program was the recognition that the LaCasse Zone and nearby Discovery Zone have similar mineralization styles. The results obtained from the widely spaced drilling at LaCasse and Discovery are suggestive of a large hydrothermal system with a sizeable mineralized footprint. The geometry of the mineralized system is inferred to be approximately tabular, paralleling the margin of the batholith, striking approximately north, dipping steeply east, and plunging to the south.

8.4.2 Grizzly Area

The Grizzly Area is located approximately 2 km north of the LaCasse Zone, or approximately 5 km north of the Schaft Creek deposit. In this area, pyrite-chalcopyrite mineralization occurs at the boundary between the Hickman Batholith and adjacent volcanic rocks of the Stuhini Group. Mineralization occurs as disseminated sulphides, and also as small sulphide veinlets. The most intense mineralization occurs in the fine-grained volcanic siltstone unit that out crops within the Grizzly area. This mineralization is associated with moderate to intense K-feldspar-epidote-chlorite alteration. Scree and talus slopes around the mineralized outcrops have abundant float with malachite and azurite. This area was previously mapped and sampled by several historical workers (Betmanis, 1978; Raven, 1979; Greig, 2009).

8.4.3 Greater Kopper Area

The Greater Kopper Area is located approximately 2 km north of the Grizzly Area, or approximately 7 km north of the Schaft Creek deposit. In this area, relatively intense chalcopyrite-pyrite mineralization occurs over a small area within intrusive rocks of the Hickman Batholith, near the contact with the volcanic rocks of the Stuhini Group. This copper sulphide mineralization occurs disseminated and within sulphide veins. Mineralization is associated with hematite-albite ± epidote alteration, and possibly with some K-feldspar alteration. This area was previously mapped and sampled by several historical workers (Betmanis, 1978; Raven, 1979; Greig, 2009). Mapping in the vicinity of the showing also identified numerous aplite dikes and barren quartz veins with K-feldspar halos. These features are interpreted as evidence of high temperature hydrothermal alteration in the Greater Kopper Area. Historically, the intrusive rocks hosting the Greater Kopper mineralization have been mapped as part of the Early Jurassic Yehinkio Pluton (Logan and Drobe, 1993). However, based on recent mapping from 2015, the Schaft Creek JV interpreted the host rocks to be Late Triassic granodiorite to quartz monzodiorite and thus similar to the Hickman Pluton further south.

8.4.4 Wolverine Creek Area

The Wolverine Creek Area is located immediately south of the Schaft Creek deposit, within a heavily forested area on the lower northwestern slopes of Mount Hicks. This area contains several separate mineralized showings, believed to be related to a possible southern extension of the Schaft Creek mineralized system. Four of these showings were examined in 2015, and each is described here.

Basal Fault Footwall Breccia: A previously unrecognized hydrothermal breccia was identified by relogging of historical drill core from near the southern limit of the Liard Zone during 2015. This breccia is a monomictic, clast-supported breccia with quartz-calcite-epidote-albite-chalcopyrite ± bornite cement. The clasts are typically subround to subangular andesitic volcaniclastic rocks and augite-phyric basaltic andesite. This breccia appears similar to a volcaniclastic breccia; however, the cement composition is interpreted as evidence of a hydrothermal origin. Currently, this breccia has only been identified over a relatively small area in approximately three historical drill holes; however, drilling in this area is very sparse and the breccia is open in several directions. Other drill holes nearby contain narrow intervals of pyrite-chalcopyrite-bearing monzodiorite dikes that appear similar to the mineralized dikes within the Liard Zone. Recognition of this breccia is potentially significant because it represents the first evidence of mineralization in the footwall of the Basal Fault.

Wolverine Creek Showing: Outcropping sulphide mineralization occurs along the creek bed of Wolverine Creek, approximately 1 km south of the breccia zone described above. In this area, monzodiorite dikes are observed to crosscut augite-phyric basaltic andesite. Pyrite and chalcopyrite mineralization and sericite-chlorite-epidote alteration occurs in the vicinity of these dikes. Historical mapping also identified chalcopyrite, pyrite, and minor bornite in outcrops along the creek bed of Wolverine Creek, as well as nearby along the creek bed of Hickman Creek (Salazar, 1978).

Monzodiorite Bluff Showing: Outcropping sulphide mineralization and intense chlorite-epidote-sericite alteration occurs near a small quartz monzodiorite stock that has been mapped northeast of Wolverine Creek, on the slopes below Mount Hicks. In this heavily forested area, several outcrops were observed that contained disseminated pyrite-chalcopyrite and pyrite-chalcopyrite-calcite veins with chlorite-epidote halos.

Mount Hicks Showing: A narrow zone of structurally controlled, high-grade Cu-Au mineralization occurs southwest of the summit of Mount Hicks. This structurally controlled zone is located at the faulted boundary between the Early Jurassic polymictic conglomerate and the underlying Late Triassic volcanic rocks. In this area, copper oxide mineralization occurs with quartz-chlorite-epidote-chalcopyrite veins that are up to 5 cm thick. These veins host highly anomalous copper and gold grades.

In addition to these four showings, the Wolverine Creek Area has been historically mapped and found to contain numerous other small zones of pyrite-chalcopyrite mineralization and chlorite-epidote ± sericite alteration (Salazar, 1978). The showings within the Wolverine Creek Area cover approximately 3 km^2. Much of this area is covered by colluvium and mature forest. The scattered occurrences of mineralization and alteration in this area are suggestive of a large contiguous zone that is poorly exposed. This zone could either represent the southernmost portion of the Schaft Creek deposit, or it could represent a separate hydrothermal center that is either located at depth or possibly to the south. The Wolverine Creek Area was a focus for soil sampling and an IP survey during 2015.

9.0 EXPLORATION

9.1 Introduction

After acquiring the Schaft Creek Project in 2002, Copper Fox performed exploration activities and completed several technical reports on the Project until formation of the Schaft Creek JV with Teck in July 2013. Since formation of the Schaft Creek JV, exploration activities have been completed by the Schaft Creek JV. The historical work programs by survey type completed on the property up to the effective date of this Report are summarized below.

9.2 Historical Mapping Programs

The initial geological investigation of the region was conducted by the Geological Survey of Canada as part of the Operation Stikine project, with mapping of the 104G 1:250,000 map sheet on which Schaft Creek is located. The geological investigation was carried out in 1956 and preceded the discovery of mineralization at Schaft Creek in 1957.

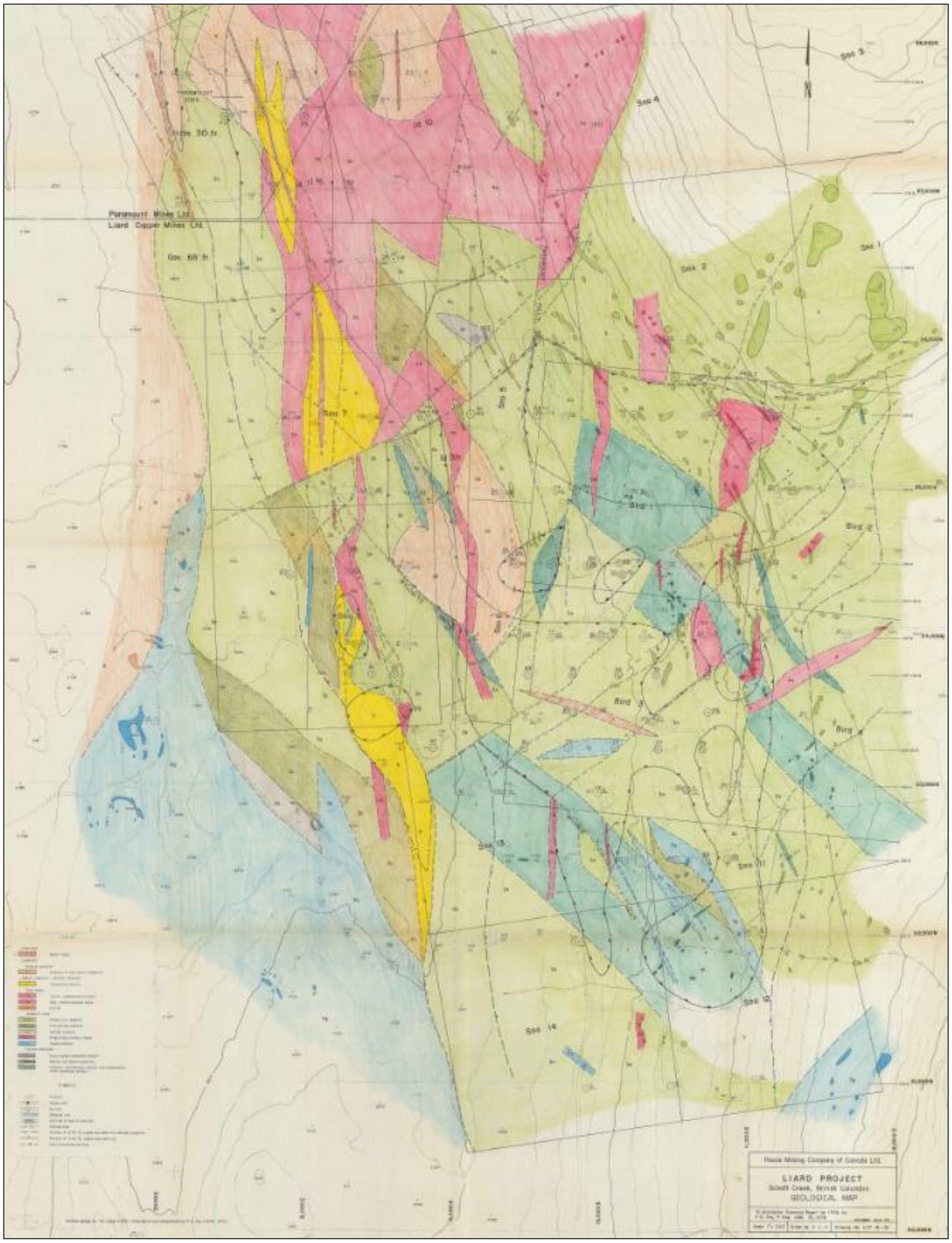
In 1964, a trenching and geological mapping program was carried out for Silver Standard and the BIK Syndicate. This program mapped the distribution of the known showings and the contact of the Hickman Batholith and Triassic andesites, noting chalcopyrite-bornite mineralization in closely spaced fractures in outcrop. Asarco's 1967 exploration program at Schaft Creek included geological mapping.

Exploration work carried out by Kennco Explorations (Western) Limited under option from the BIK Syndicate in 1959 included geological surveys. In 1964, a trenching and geological mapping program on eight profiles across the Schaft Creek deposit was carried out on what were then the Bud, Sno, and Bird claims for Silver Standard and the BIK Syndicate. This program assessed the prospectivity of claims, mapped the distribution of the known showings, and attempted to understand the relationships between the mineralized zones. The party mapped the contact of the Hickman Batholith and Triassic andesites, noting chalcopyrite-bornite mineralization in closely spaced fractures in felsites at the Bird showing. Asarco's 1967 exploration program at Schaft Creek included geological mapping.

9.2.1 Hecla / Paramount, 1968–1977

In 1969, Hecla optioned the property and continued work on the Project until 1973, conducting extensive exploration work. Hecla's geological map and cross sections of the deposit have stood the test of time, identifying many key features of the deposit, including the Paramount and West Breccias, syn-mineral porphyry dikes, and major structures. The level of detail of the Hecla work is demonstrated in the geological map shown in Figure 9-1, which formed the basis for the modern interpretation (Figure 9-4).

Figure 9-1: Hecla's Historic 1978 Geological Map of the Schaft Creek Deposit



9.3 Grids and Surveys

In 1969, Hecla contracted Underhill and Underhill to set up a local grid. Nine cairns were erected using a helicopter. A survey of the claims separating the Liard and Paramount properties was completed, and a legal boundary was established.

In 1980, Teck contracted McElhanney Associates (McElhanney) to survey all drill hole collars using a laser theodolite located at fixed, previously surveyed points with prisms at drill hole collars. McElhanney also surveyed some of the Hecla drill holes to tie in the survey to the previous Hecla surveys.

Eagle Mapping completed a photogrammetry survey during 2005. Data was provided at 1:2,000 scale and on 5 m contour intervals.

A light detection and ranging (LiDAR) survey was flown over the deposit area in 2011. A helicopter Z-Axis tipper electromagnetic (ZTEM) geophysical survey was conducted in 2013. Both surveys produced digital terrain models (DTMs) and digital elevation models (DEMs). The provincial government's Terrain Resource Information Management provides base topographic data for the Province of British Columbia, and a DEM can be purchased from the government.

The topographic surface used by the Schaft Creek JV in resource estimation was generated from a 1 m DTM. This file has been inherited from previous resource estimation efforts, and the ultimate source of the file has not been resolved, though it was likely originally sourced from the 2011 LiDAR survey. The collar locations were compared to the topographic surface, and due to some differences in elevation, all collar locations were relocated to the topographic surface.

9.4 Geological Mapping

In 1964, the BIK Syndicate completed basic mapping of eight traverses crossing the area of the Schaft Creek deposit.

In 1969, Hecla completed regional geological mapping of the area surrounding the Schaft Creek deposit covering an area of 10 miles by 6 miles at a scale of 1″:400′.

In 1971, the Geological Survey of Canada mapped the regional geology at a scale of 1:250,000.

In 2007, Copper Fox completed a mapping program that encompassed an area of 3.6 km by 2.6 km (936 ha) at a scale of 1:5,000 using global positioning system (GPS) control. Targeted outcrops were identified by airphoto interpretation and archival geological maps. Locations were subsequently plotted on a 1:5,000 topographical base map derived from digital orthophoto georeferenced maps produced by Eagle Mapping Ltd.

9.4.1 Schaft Creek JV, 2014

Geological mapping of the Schaft Creek deposit area was conducted during 2014 at two scales. The northeastern portion of the Liard Zone was mapped at a 1:1,000 scale utilizing the excellent exposure provided by a large number of historical drill road cuts, trenches, and drill pads. The remainder of the deposit area was mapped at a 1:5,000 scale. This work primarily focused on the outcrop above treeline to the northeast of the Main and Paramount Zones, and further north near the

Discovery and LaCasse Zones. Geological information outside of this field mapping area was compiled from historical maps by Hecla, Teck, Copper Fox, the British Columbia Geological Survey, and other workers.

The geological mapping was conducted using a modified version of the paper-based Anaconda-style. This method captures outcrop-based information on lithology, structure, veins, alteration, and mineralization, with a focus on alteration minerals and crosscutting relationships. Mapping was conducted on mylar sheets using aerial photographs, topographic maps, and a handheld GPS. Structural measurements were made using a Brunton compass. The geological mapping was supplemented by top-of-hole relogging of drill holes in areas with limited outcrop exposure.

9.4.2 Schaft Creek JV, 2015

The 2015 mapping program focused on two areas: Mount LaCasse to the north and Mount Hicks to the south. Mapping was carried out at a scale of 1:5,000 using a modified Anaconda-style, providing 18 km^2 of coverage. The Mount LaCasse mapping continued on from where the 2014 mapping left off at the LaCasse target, providing coverage northward along the prospective contact of the Hickman batholith and the Stuhini volcanics, including assessment of mineral occurrences along this trend. Mapping to the south in the Mount Hicks area aimed to put the south end of the Liard Zone into context with an improved understanding of the stratigraphy and structure to the south of the deposit, and to assist in the target development work carried out over a covered area immediately south of the resource area.

Geological maps were scanned, georeferenced, digitized into ArcGIS, and merged with 2014 and 2015 mapping to create a seamless geology map (Figure 9-4).

9.5 Geophysics

In 1974, Hecla established a grid for mapping and geophysical surveys, and conducted low level air photography. IP surveys completed by McPhar Geophysics Ltd. revealed the distribution of sulphides, in particular, the pyritic halo.

In 2007, Copper Fox retained Associated Geosciences Ltd. to performed IP, electrical imaging, and magnetic total field surveys to map bedrock topography to support project infrastructure. The IP survey located a 250 to 300 m wide IP-chargeability anomaly immediately east of the Liard Zone.

In 2010 and 2011, Copper Fox completed 70.2 km of direct current induced polarization (DCIP) and 64.0 km of magneto-telluric (MT) surveys over the Schaft Creek deposit. The surveys were completed on 24 east-west oriented lines and 3 north-south oriented tie lines that were surveyed using differential GPS instrumentation. The surveys were completed at 400 m line spacing, with stations at 50 m intervals along lines. The IP survey failed to reconfirm the chargeability anomaly located immediately east of the Liard Zone identified in 2007.

The main results of the 2010 to 2011 surveys are:

a. The chargeability anomaly associated with the Schaft Creek deposit extends over a strike length of 3,200 m.

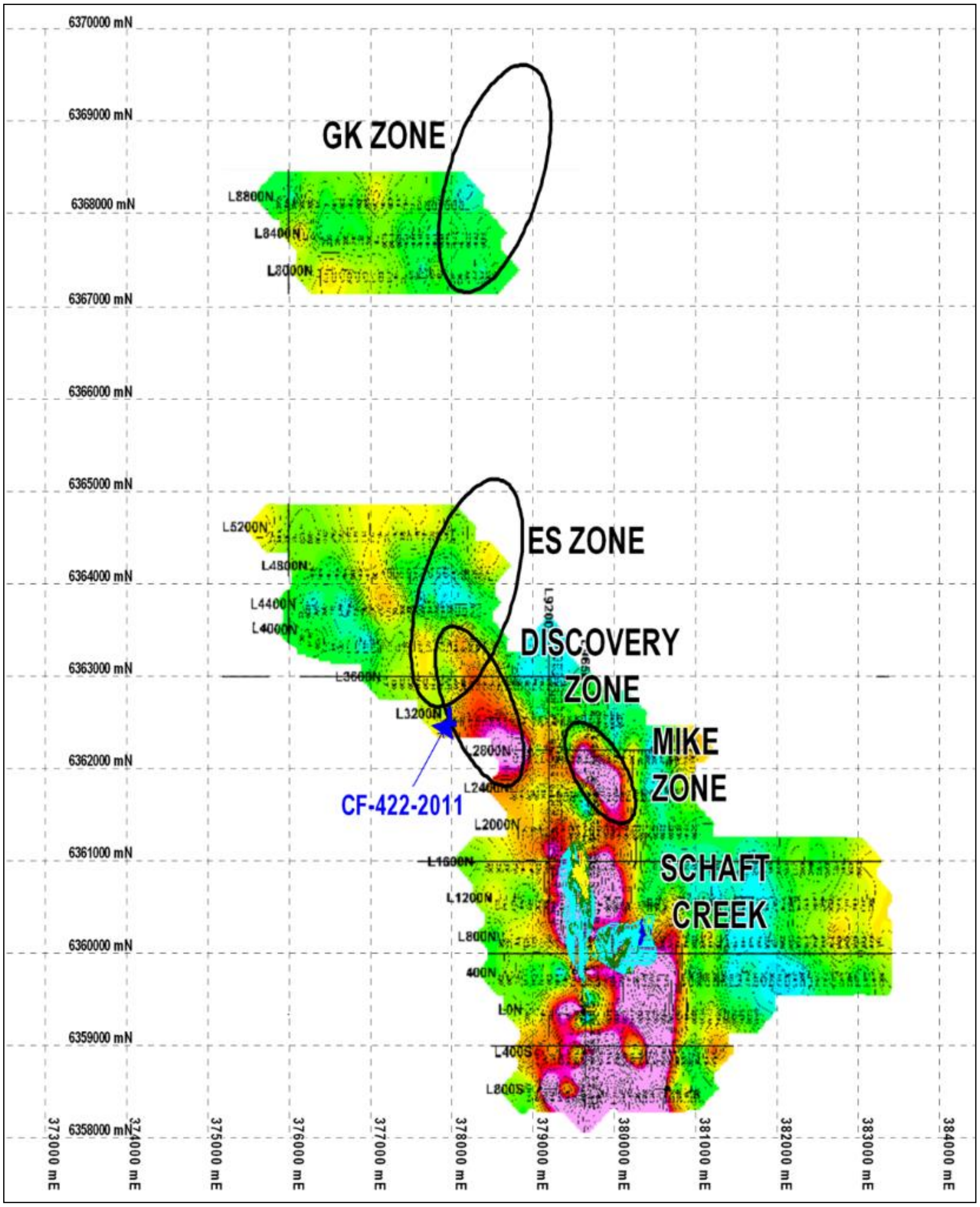
b. The chargeability anomaly suggested that the majority of the historical drilling was completed on the west flank of the deposit (Liard Zone) and was too shallow to test the eastern deeper part of the chargeability anomaly.

c. New chargeability anomalies were identified over the Mike Zone and the ES and GK Zones (all within a 6 km strike length) located north of the Schaft Creek deposit.

d. A total of 31 potential targets with different priority levels were identified by the 2010 to 2011 survey.

Results of the surveys are shown graphically in Figure 9-2.

Figure 9-2: Results of the Quantec Titan 24 IP and MT Survey, with Chargeability Anomalies Outlined



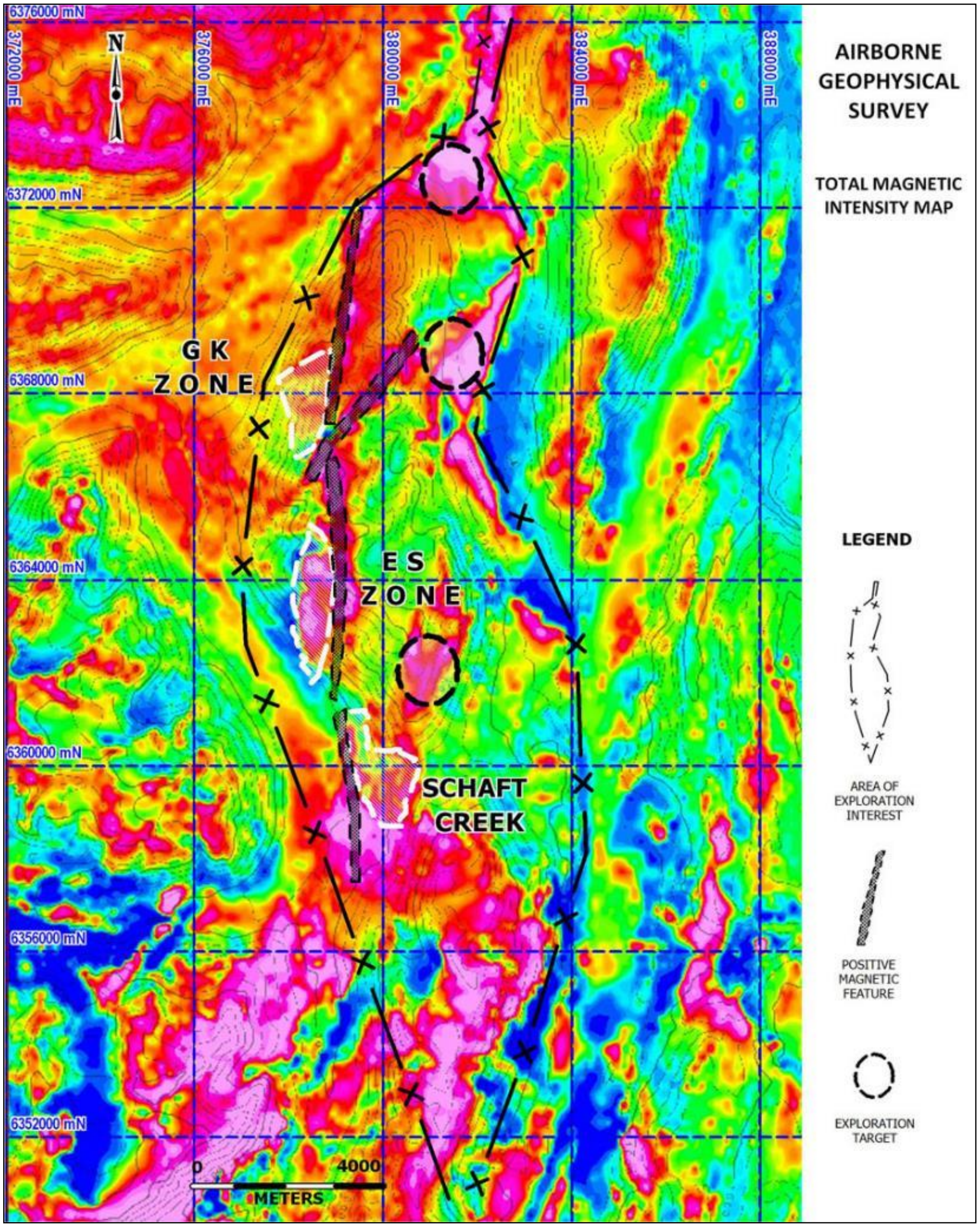
Source: Copper Fox (2021)

In 2011 and 2012, Copper Fox contracted Precision GeoSurveys Inc. from Vancouver, British Columbia, to complete high-resolution aeromagnetic surveys that effectively delineated the extent of the major intrusive rock units located near the deposit, and identified some areas of magnetite alteration that were verified through logging and surface mapping.

The aeromagnetic surveys were flown at 200 m spaced flight lines at an average altitude above ground level of 39 m. Tie lines were flown at 2 km intervals. The magnetic data were collected using a Scintrex cesium vapour CS-3 magnetometer, which is a high sensitivity / low noise magnetometer. The magnetometer and the base station used in the survey have absolute accuracy range of 0.1 nT (gamma) and a sensitivity of 0.1 nT (gamma) at a two-second sampling rate.

The survey covered a 25 km long by 17 km wide area. A total of 2,514 line kilometres (including tie-lines) of survey were completed. The total magnetic intensity map including the locations of the Schaft Creek deposit and the ES and GK zones of copper mineralization are shown in Figure 9-3.

Figure 9-3: Total Magnetic Intensity Map



Source: Copper Fox (2021)

During 2013, the Schaft Creek JV completed a ZTEM survey over the deposit area. The survey was flown at a high and variable sensor height, so the magnetic data acquired is of inferior quality to that obtained from the 2011 survey. The ZTEM data, however, provided an indication of the resistivity throughout the deposit area, and can show major faults and low resistivity layers at surface related to landslides or hydrothermal alteration. The resolution of this data is less than that of the DCIP resistivity inversion models produced in 2010 to 2011, but the ZTEM system has superior depth penetration;

therefore, it can show the depth extent of faults and any resistive features underlying the conductive surface material.

In 2015, the Schaft Creek JV completed an IP/ground magnetometer survey (16 line km) consisting of eight east–west trending lines spaced 300 m apart, with line lengths ranging from 1.5 km to 2.5 km. This survey identified an anomaly in the southern portion of the Liard Zone, consistent with pyrite, chalcopyrite, and molybdenum mineralization. The resistivity data was used to outline structures such as the Breccia Footwall Fault and an interpreted fault along Wolverine Creek.

9.6 Pits and Trenches

Little trenching has been conducted at Schaft Creek due to the presence of thick till and alluvial layers, which prevent surficial excavations from exposing bedrock. Prospector Nick Bird, employed by BIK Syndicate in 1957, discovered copper mineralization that became the Schaft Creek Property, and explored the occurrences using hand tools to excavate narrow trenches on outcrop in the Saddle area.

In 1972, Phelps Dodge Corporation of Canada Ltd. completed cobra drill trenching and bulldozer trenching. All of the trenches yielded disappointing results. The copper mineralization in the trenches appeared to be best developed in the vicinity of sericitized shears and fractures.

9.7 Petrology, Mineralogy, and Research Studies

Petrography, mineralogical, and paragenetic studies in support of mineralogical and geological interpretations have been completed on the Project.

In 2011, Copper Fox completed a geochemical study to refine the volcanic stratigraphy and investigate the chemical variability within the deposit. The study utilized 185 samples collected specifically for whole rock geochemical analysis and approximately 12,000 assay samples. Samples had been analyzed by lead (Pb)-fire assay, Inductively Coupled Plasma Mass Spectrometry (ICP-MS), and Inductively Coupled Plasma Emission Spectroscopy (ICP-ES) methods with an open vessel four-acid digestion. The whole rock geochemical data was used to construct a Pearce diagram.

The Pearce diagram generated 10 geochemical units and indicated the majority of the volcanic units were of sub-alkaline basaltic composition and that the majority of the felsic intrusives were likely derived from an intermediate felsic magma. The inconsistencies between geochemical grouping classifications and the lithologies from field logging were found to be a reflection of the intense alteration (Caron et al., 2012).

In January 2013, Copper Fox completed a petrographic, mineralogical (QEMSCAN), and geochemical study of 147 drill core samples from 41 core holes, plus two samples from the Hickman Batholith (LeBoutillier, 2013). The core holes span a period from 1966 to 2011 and represent drilling campaigns by Hecla, Teck, and Copper Fox. The samples cover all the main zones of the Schaft Creek deposit as well as its surrounding margins. The samples were analysed for major oxides and trace elements.

9.8 Geotechnical and Hydrological Studies

Geotechnical studies were performed by Knight Piésold in support of pit slope assumptions for the 2013 FS. Six pit design sectors were identified, and pit inter-ramp slope angles were recommended that ranged from 42° to 50°. The initial pit slope in overburden was recommended at 27°.

McElhanney conducted a preliminary hydrology assessment of the streams in the Project area in support of mining studies. BGC Engineering Inc. (BGC) conducted an independent hydrological assessment of the streams along the proposed mine access road.

9.9 Metallurgical Studies

Metallurgical test work is discussed in Section 13.0.

9.10 3D Geological Model, 2011

Copper Fox retained Cambria Geosciences to assist in updating the 3D geological model for the Schaft Creek deposit. Wireframes were constructed in Surpac™ software, and were generated for various geological domains including base of overburden, breccia zones, fragmental zones, alteration zones, intrusive units, and fault zones. The model assisted Copper Fox in identifying future targets for definition drilling.

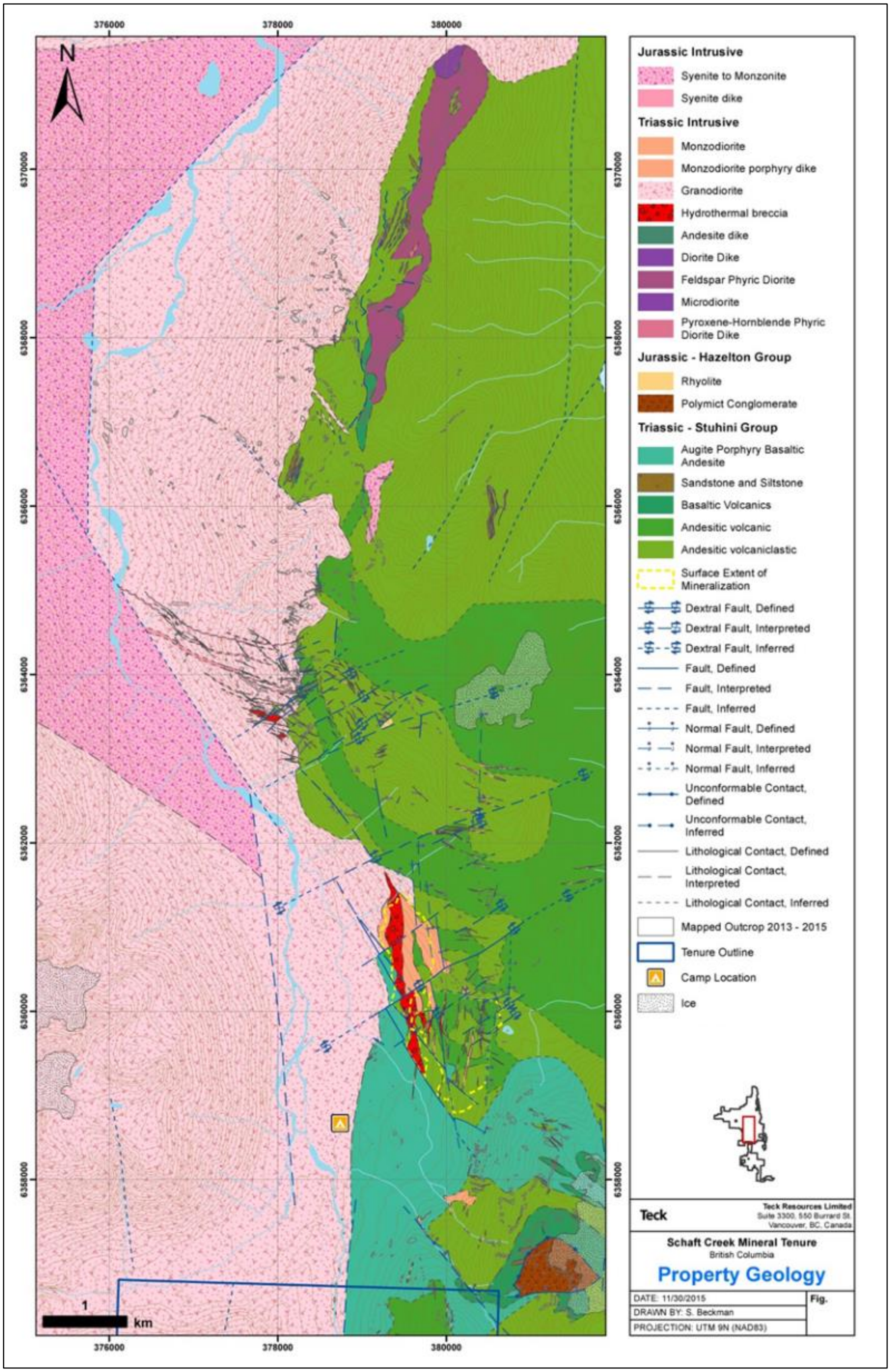
9.11 NI 43-101 Technical Studies

In 2007, Copper Fox released the results of a PEA of the Schaft Creek deposit. The technical report entitled "Preliminary Economic Assessment on the Development of the Schaft Creek Project located in Northwest British Columbia Canada" was prepared by Samuels Engineering Inc., with an effective date of December 7, 2007, Bender, M.R. et al. as QPs.

In 2008, Copper Fox announced the results of a preliminary FS on the Schaft Creek Project. The technical report entitled "Preliminary Feasibility Study on the Development of the Schaft Creek Project located in Northwest British Columbia Canada" was prepared by Samuels Engineering Inc., with an effective date of September 15, 2008, Bender, M.R. et al. as QPs.

In 2013, Copper Fox announced the results of an FS on the Schaft Creek Project. The technical report entitled "Feasibility Study on the Schaft Creek Project British Columbia Canada" was prepared by Tetra Tech with an effective date of January 23, 2013, Farah, A. et al. as QPs.

Figure 9-4: Consolidated Geology Map of the Schaft Creek Property



9.12 Drill Core Relogging

9.12.1 Copper Fox, 2011

In 2011, Copper Fox completed re-sampling and re-logging of historical drill core from the Schaft Creek deposit. Select holes and intervals of historic drill core were re-logged to formulate a consistent geological and structural interpretation as input for the construction of a 3D model of the deposit, as well as to confirm or revise the descriptions for lithologies, alteration, and structures (Caron et al., 2012). During this program, the rejects of 5,359 historic samples were re-assayed at Acme Analytical Laboratories Ltd. (Acme Labs) in Vancouver, British Columbia.

This work identified a significant amount of sodic feldspathization overprinting in the Schaft Creek deposit, and the gently dipping porphyritic units previously considered to be intrusives were found to be volcanic or sub-volcanic porphyritic feldspar phyric flows with sodic feldspar overprints.

The 2011 work also demonstrated early, strong, potassic alteration in the breccias of the Paramount Zone. The structural mapping program identified three faults as the main structural controls on the mineralization as well as the structurally controlled hydrothermal breccia.

9.12.2 Schaft Creek JV, 2013–2015

Relogging of historical drill core was a key component of the three field programs that the Schaft Creek JV completed from 2013 to 2015. This relogging was critical to understand historical work on the Project, as well as advancing the knowledge of the deposit. Relogging of historical drill core was conducted in camp at the core-logging facility. The majority of the historical drill core is stored in wooden storage racks that allow for easy access to individual holes or sections within a hole.

Geological relogging was conducted using a modified version of the paper-based Anaconda-style. This method captures interval-based information on lithology, structure, veins, alteration, and mineralization, with a focus on mineral paragenesis and crosscutting relationships. Select information from these paper logs was subsequently digitized into an Excel format, imported into an acQuire database, and incorporated into a 3D geological model. Geological relogging was conducted by a large team of geologists; efforts were made throughout the programs to maintain consistency between loggers. This was done by having group discussions regarding individual drill holes and by developing a library of reference samples, which is housed in the core-logging facility.

During 2013, approximately 10,850 m of drill core from 30 holes was relogged. This relogging was used as the basis for developing a new lithological classification scheme intended to be more consistent and simpler than the previous lithological naming convention scheme (as described in the geology section). This relogging work also improved understanding of major structures in the deposit and provided a basis for a new alteration assemblage classification scheme (as described in the geology section). The latter clarified the understanding of the distribution of hydrothermal alteration in the deposit.

In 2014, relogging focused predominantly on three cross sections within the Main Zone. The three main goals for this relogging were to understand (1) the controls on mineralization in the Liard Zone; (2) variations in mineralization within the Liard Zone; and (3) the transitions between the Liard, West Breccia, and Paramount Zones. In addition to these cross sections, additional holes within the deposit were logged to verify faults and examine geological relationships. Top-of-hole data was obtained for a large number of holes to add information to the geological map in areas without surficial outcrop exposure.

A preliminary 3D geological model was completed at the end of 2014. As part of this modeling process, a conversion was created to combine historical logging codes and recent relogging codes used by the Schaft Creek JV (Bailey et al., 2014). The preliminary 3D model completed in 2014 highlighted several regions of uncertainty that require additional work to resolve.

During the 2015 field program, 11,439 m of historical drill core was relogged from the deposit area, bringing the total metres relogged to 42,999 m, approximately 40% of the total drilled metres on the Project. The relogging in 2015 intended to accomplish three goals: (1) improve the 2014 3D geologic model; (2) increase the knowledge of areas within the deposit that have been recognized as opportunities to expand the resource and/or delineate additional higher-grade zones; and (3) collect data on intervals that had been selected for geometallurgical sampling and that had not previously been relogged by the Schaft Creek JV.

Following the completion of the 2015 field program, relogging data from 2013 to 2015 was combined with historical logging information and geological mapping data to create an updated 3D geological model. Further detail on geological modelling is provided in Section 14.0.

9.13 Surveying

Various surveying methods have been used over the life of the Project. The 2012 Technical Report details surveying methods used by Copper Fox between 2005 and 2011. Surveying methods used for the 2013 and 2015 drill programs are described below in Section 9.15.

9.14 Topographic Surface

The topographic surface used is called “topo_trim.00t” and was generated from a 1 m resolution DTM. The data was sourced from a 2016 DEM named “DEM_sck_topo1m_validated_higherRes.dxf”. This file has been inherited from previous resource estimation efforts, and the ultimate source of the file has not been resolved. The collar locations were compared to the topographic surface, and due to some differences in elevation, all collar locations were relocated to the topographic surface. The QP compared the 2016 DEM against downloaded NASA Shuttle Radar Topography Mission (SRTM) data (see Section 12.2). The high-resolution 2016 DEM compares well with the low-resolution SRTM with a mean difference of less than 0.04 m.

9.15 Exploration Potential

9.15.1 Overview

Schaft Creek is part of an extensive porphyry complex with copper mineralization occurring over a 12 km strike length along the Stuhini/Hickman contact. Geological mapping and geophysical surveys and limited exploratory diamond drilling located a number of mineralized areas that indicate good potential for additional porphyry copper–gold mineralization within the complex. The 2011 to 2012 geophysical surveys outlined an exploration area of interest that is approximately 4 km wide by 20 km long (Figure 9-3). Exploration activities in 2012–2016 outlined the prospects and zones shown in Figure 9-2. A number of these zones had been identified during legacy exploration activities.

9.15.2 Wolverine Creek / Liard Zone Extension

The limits of the Liard Zone are not well constrained to the south. This extension covers a large area that hosts a positive chargeability anomaly, a limited number of drill holes with chalcopyrite-bornite-molybdenite, and poor outcrop exposure. A previously unrecognized hydrothermal breccia was identified during the re-logging programs during the 2015. Recognition of this breccia is potentially significant because it represents the first evidence of mineralization in the footwall of the Basal Fault.

The Wolverine Creek Area is located immediately south of the Schaft Creek deposit, within a heavily forested area on the lower northwestern slopes of Mount Hicks. This area contains several separate mineralized showings, all of which are believed to be related to a possible southern extension of the Schaft Creek mineralized system.

10.0 DRILLING

A total of 449 drill holes (about 108,041 m) have been completed in the Project area. Of this total, 238 holes (60,432 m) were drilled by Silver Standard, Asarco, Hecla, Paramount, and Teck in the period from 1956 to 1981, 197 holes (41,524 m) were completed by Copper Fox from 2005 to 2012, and 14 holes (6,087 m) were drilled by the Schaft Creek JV from 2013 to 2015. No drilling has been completed on the Project since 2015.

A total of 21 drill holes have no assays and 40 drill holes within the Project lie outside the resource area.

Drilling conducted by Copper Fox and earlier operators to year end 2011 is described in the 2012 technical report (Sections 6.0 and 10.0). Drilling conducted by Copper Fox in 2012 and the Schaft Creek JV in 2013 and 2015 is described below with drill hole collars shown in Figure 10-1.

10.1 Copper Fox, 2012

10.1.1 2012 Diamond Drill Holes

In 2012, Copper Fox drilled six holes targeting chargeability anomalies 700 m to 1,600 m north of the Paramount Zone. These anomalies were identified in the 2011 geophysical surveys and named the Discovery and Mike Zones (Figure 10-1). In 2011, DDH 2011CF422 tested one of the chargeability anomalies and intersected significant porphyry style copper mineralization. Table 10-1 provides details for diamond drill holes completed in 2012.

Table 10-1: Summary 2012 Diamond Drill Holes

Drill Hole	Northing	Easting	Elevation (m)	Depth (m)	Collar Azi	Dip
2012CF426	378264.16	6362615.28	957.45	789.43	91.6	60.2
2012CF427	378113.24	6362930.99	1019.27	769.92	90.1	59.1
2012CF428	378812.97	6362213.35	1116.79	223.11	270.0	74.7
2012CF429	379498.94	6362219.71	1457.05	132.89	90.0	65.0
2012CF429B	379498.94	6362219.71	1457.05	178.61	90.6	73.1
2012CF430	378442.50	6362948.36	1176.59	171.30	265.0	76.6

Of the four holes drilled in the Discovery Zone, DDH 2012CF428 and DDH 2012CF430 were terminated prematurely due to drilling difficulties. In the Mike Zone, DDH 2012CF429 and DDH 2012CF429B were drilled from the same collar location approximately 700 m east of DDH 2012CF428 on the same section line and drilled due east. DDH 2012CF429B was drilled at a shallower angle than hole 2012CF429. Both holes were terminated prematurely in a major fault zone. Table 10-2 provides a summary of all significant mineralized intervals in the Discovery Zone.

Table 10-2: Significant Mineralized Intervals Discovery Zone

DDH ID	Azi.	Dip	Northing	Easting	From (m)	To (m)	Interval (m)	Cu (%)	Au (g/t)	Mo (%)	Ag (g/t)
2011CF422	90	-50	6362561	377976	83.00	184.35	101.35	0.204	0.09	0.009	1.60
					184.35	318.00	133.65	0.101	0.04	0.011	0.56
CF426-2012	90	-60	6362615	378264	76.55	767.66	691.11	0.16	0.04	0.003	0.81
				including	76.55	123.65	47.10	0.24	0.05	0.001	2.05
				including	476.50	658.40	181.90	0.21	0.05	0.003	0.68
				including	702.66	767.66	65.00	0.24	0.04	0.004	0.94
CF-427-2012	90	-60	6362930	378113	428.12	764.84	336.72	0.24	0.14	0.006	0.57
				including	509.00	556.00	47.00	0.62	0.59	0.006	2.02
				including	511.00	523.00	12.00	1.23	2.12	0.019	6.36
CF428-2012	270	-75	6362213	378812	No Significant Mineralization						
CF430-2012	90	-75	6362948	378442	122.75	132.14	9.39	0.14	0.01	tr.	5.55
				including	130.75	132.15	1.40	0.16	0.03	tr.	414.00
SCK13-436	90	-60	6362264	378492	50.50	68.50	18.00	0.09	0.117	0.00	0.56

Note: SCK-13-436 was drilled by the Schaft Creek JV in the 2013 drill campaign.

Tahltan Drilling Services Corporation provided diamond drilling services using a skid-mounted Zinex A5 drill rig and a helicopter-portable Zinex A5 drill. All drill holes were started with either HQ diameter coring tools (63.5 mm diameter core) or with HQ3 diameter coring tools (61.1 mm diameter core) with HW or HWT surface casing.

10.1.2 Core Logging Procedure

The 2012 core logging procedures included geotechnical core logging and core orientation measurements (where possible) at the drill site, and core interval and magnetic susceptibility measurements at the core logging facility. Observations of the general rock type, rock weathering, veining type, presence/absence of mineralization, and an estimate of the overall rock strength were also recorded. Preliminary analysis of rock and mineral geochemistry was performed using the Niton portable XRF analyzer to help identify unknown minerals. Data gathered from the logging was entered directly into a master database using an acQuire database system.

Thirty samples of full diameter drill core representative of lithology, alteration, or mineralization were selected at 100 m intervals from drill holes 2012CF426 to 2012CF430 for specific gravity (SG) determination. The SG of these samples were measured according to Acme Labs code G813-WAX.

10.2 Diamond Drill Hole Results

Significant results of the drill holes completed in the Discovery Zone in the period 2011 to 2013 are summarized below.

DDH 2011CF422 is located approximately 1,200 m north of the Paramount Zone and intersected copper-molybdenum mineralization at a core interval depth of 83 m and remained in mineralization to the bottom of the hole at 318 m. The mineralization shows a strong correlation to the outer edge of a large (1,800 m by 800 m) chargeability anomaly identified in 2011. The strongest portion of the chargeability anomaly is located east of the drill hole collar location.

DDH CF426-2012 is located approximately 300 m east of DDH 2011CF422 and tested the eastern extension of the chargeability anomaly in the Discovery Zone. The hole intersected variable concentrations of chalcopyrite mineralization occurring as disseminations, and in veins and veinlets in variably altered andesite. Visible molybdenite mineralization occurs sporadically throughout the core in quartz veinlets and in some instances also with chalcopyrite mineralization.

DDH CF427-2012 is located approximately 400 m northwest of DDH CF426-2012 and tested the strike extension of the chargeability anomaly located in 2011. The hole intersected variable concentrations of chalcopyrite and sporadic molybdenite mineralization from a core depth of 250 m to the end of the drill hole at 764.8 m. The chalcopyrite ± molybdenite mineralization occurs as disseminations, and in quartz veins and veinlets in a granodiorite intruded by a number of thin mafic dikes. Broad intervals of silicification were noted in this drill hole.

DDH CF430-2012 is located approximately 300 m east of DDH CF427-2012 and tested the eastern extension of the 2011 chargeability anomaly (the Discovery Zone). This hole was terminated due to drilling difficulties before reaching the chargeability anomaly intersected in DDH CF727-2012. The hole intersected a narrow interval of mineralization. The average silver grade in the mineralized interval is significantly affected by one sample that assayed 414 g/t (13.31 oz/t) silver. In determining the weighted average grade of this interval, the 414 g/t silver assay was arbitrarily cut to 31.1 g/t silver.

DDH SCK-13-436 was drilled by the Schaft Creek JV in 2013 to test the projected extension of the chargeability anomaly that exhibited high magnetism in the Discovery Zone. The hole was abandoned prematurely before reaching the projected top of the chargeability anomaly. The hole intersected volcaniclastic lapilli tuff, andesitic volcanics, and minor volcaniclastic breccia. Intense fracturing occurs to a depth of approximately 90 m accompanied by copper oxide staining of the fracture surfaces. The drill hole intersected predominantly propylitic and sodic-calcic alteration assemblages with minor zones of silicification. Iron oxide alteration consisted of disseminated magnetite and vein-controlled hematite. Sulphide mineralization in the hole is minimal, consisting of vein-hosted and disseminated chalcopyrite and pyrite to a depth of 124 m, below which pyrite was the only sulphide mineral encountered. The overall sulphide content diminishes downhole.

10.2.1 Mike Zone

DDH CF429-2012 and DDH CF429B-2012 were completed to test the 1,000 m long by 500 m wide strong chargeability anomaly identified in 2011 the (Mike Zone). Both drill holes were terminated at core intervals of approximately 140 m due to extremely difficult ground conditions. These holes did not reach the chargeability anomaly.

DDH SCK-13-431 was drilled by the Schaft Creek JV in 2013 and tested a chargeability anomaly (Mike Zone) located northeast of the Paramount Zone of the Schaft Creek deposit. This drill hole intersected a broad interval of disseminated pyrite and trace chalcopyrite hosted in propylitic altered (chlorite-epidote ± hematite ± calcite) andesitic volcanics and volcaniclastic tuff. Very minor K-feldspar alteration occurs in a porphyritic dike intersected over the core interval 593.4 m to 604.2 m. This hole is located further north and at higher elevation compared to all other historical drilling in the Schaft Creek deposit area and contains lithologies that could be representative of the overlying Hazelton Group. Pyrite concentrations of up to 15% occur between 113 m to 269 m and 10% between 560 m to 575 m to ~10%.

10.3 Schaft Creek JV, 2013

The Schaft Creek JV drilled five exploration drill holes and four geotechnical holes in in the Schaft Creek deposit in 2013, for a total of 3,453 m (Figure 10-1, Table 10-3, and Table 10-4). The geotechnical drill holes were designed to collect information on the Paramount Zone and on the slope to the northeast of the Paramount Zone. The exploration drill holes were designed to test for extensions to the Paramount Zone along strike and at depth, and to test targets in the Discovery and Mike Zones. Of the holes drilled in 2013, three exploration drill holes (SCK-13-434, 436, and 438), and one geotechnical drill hole (SCK-13-437) did not reach their target depths due to difficult ground conditions.

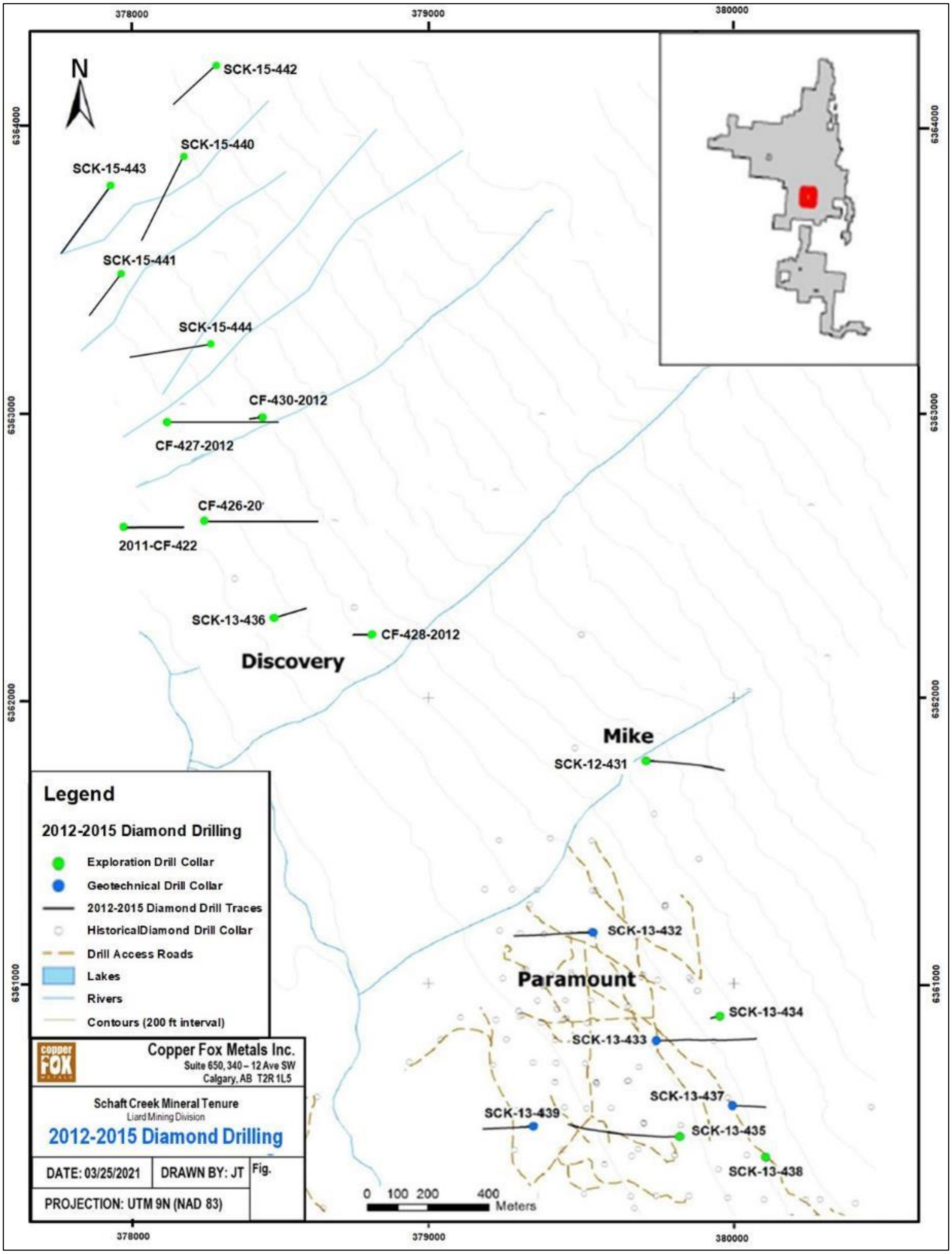
Table 10-3: 2013 Drill Hole Collar Information – Exploration Program

Collar ID	Easting NAD83	Northing NAD83	Elevation (m)	Collar Azimuth	Collar Dip	Depth (m)	Zone
SCK-13-431	379712	6361778	1332	090	65	628	Mike
SCK-13-434	379953	6360884	1171	270	80	180	Paramount
SCK-13-435	379822	6360465	996	270	70	797	Paramount
SCK-13-436	378493	6362265	972	090	60	224.5	Discovery
SCK-13-438	380103	6360393	1125	090	60	15	Paramount

Table 10-4: 2013 Drill Hole Collar Information – Geotechnical Program

Collar ID	Easting NAD83	Northing NAD83	Elevation (m)	Collar Azimuth	Collar Dip	Depth (m)	Zone
SCK-13-432	379538	6361178	1059	270	60	538.5	Paramount
SCK-13-433	379745	6360800	1040	090	55	566.5	Paramount
SCK-13-437	379995	6360572	1094	090	55	202.5	Paramount
SCK-13-439	379345	6360500	900	270	60	302.5	Paramount

Figure 10-1: 2012–2015 Exploration and Geotechnical Drill Holes Location Map



10.3.1 Diamond Drilling Procedures

Two separate drilling contractors conducted the diamond drilling in 2013. Geotechnical drilling was performed by Geotech Drilling Services Ltd. using two skid-mounted drill rigs. Exploration drilling was performed by Tahltan Drilling Services Corp. using two helicopter-portable drill rigs. Drills were mobilized to site from the airstrip in Telegraph Creek beginning on August 22nd. Drilling typically commenced using PQ or HQ diameter core, and the core size was typically reduced downhole to accommodate for poor ground conditions. Drill operations were completed in mid-October.

Prior to drilling each hole, collar locations and orientations were surveyed and staked by project geologists using a handheld GPS and Brunton compass. Timber drill pads were constructed by Sawtooth Range Enterprises Ltd. for the helicopter-supported drill sites. Skid rigs were dragged into position using pre-existing drill roads. All drilling was supported by an AS350B2 helicopter on contract from Pacific Western Helicopters. Drill core was transported to the exploration camp by helicopter for logging. Down-hole survey measurements of drill hole orientation were collected using a Reflex EZ-Shot tool, with measurements typically taken every 50 m. Following completion of drilling, timber drill pads were removed and drill sites were cleared and rehabilitated. Drill hole collar locations were then surveyed using a Trimble differential GPS.

10.3.2 Core Logging Procedures

Core logging was conducted at the core-logging facility located in the exploration camp. Upon arrival at the core-logging facility, core boxes were re-labeled, core recoveries calculated, and rock quality designation (RQD) geotechnical measurements taken. Geological logging was conducted using a modified version of the Anaconda-style to capture interval-based information on lithology, structure, veins, alteration, and mineralization. Select information from these paper logs was subsequently digitized into an Excel format and then imported into an acQuire database.

Photographs of drill core were taken following completion of logging and sample assignment. The core was then subsequently cut and sampled. After sampling, all core boxes were stacked outside on racks within the core storage area. All 2013 drill core is stored in the core storage area beside the core logging facility in the Schaft Creek exploration camp.

More details on the core logging procedures and an example log can be seen in Section 10.4.2 Core Logging Procedures from the 2015 program. The core logging procedures from both the 2013 and 2015 programs were the same.

10.3.3 Diamond Drilling Results

A summary of significant mineralized intercepts from 2013 drilling is presented in Table 10-5.

Four holes (SCK-13-434, 436, 437, and 438) were abandoned prior to reaching their target depths. Of the remaining five holes, two contain significant mineralized intercepts, including Geotechnical hole SCK-13-432 and Exploration hole SCK-13-435. Significant mineralized intercepts from the 2013 program are summarized here briefly.

SCK-13-432 (18 m to 286.1 m, grading 0.237% Cu, 0.238 g/t Au, 0.015% Mo): This hole was drilled at the northern limit of the Paramount Zone. This hole is significant because it extends the strike length of the Paramount Zone to the north, and also because mineralization occurs near-surface. In particular, the interval at 67.3 m to 166.5 m is significant because it demonstrates unusually high grade at fairly shallow depths (99.15 m grading 0.361% Cu, 0.401 g/t Au, 0.031% Mo).

SCK-13-435 (239 m to 665.5 m, grading 0.324% Cu, 0.112 g/t Au, 0.021% Mo): This hole was drilled to test the down-dip extent of the central Paramount Zone at depth, in an area with a high chargeability IP anomaly. No deep drilling had been conducted previously within this particular part of the Paramount Zone. This hole is significant because it demonstrates the continuity of the Paramount Zone at depth, and indicates that there are more opportunities remaining to expand the Paramount Zone resource at depth. This hole also intersected two small intervals of high grade mineralization (i.e., 307 m to 335 m grading 0.817% Cu, 0.324 g/t Au, 0.051% Mo). These high-grade intervals consist of hydrothermal breccias with abundant chalcopyrite-bornite-molybdenite cement. Further drilling and relogging is required to determine if these high-grade breccia intervals represent traceable domains that can be delineated by future drilling.

SCK-13-431 (Mike target area): This hole was designed as an initial drill test of the Mike target area. The Mike target area is defined by a large high-chargeability IP anomaly. This drill hole did not intersect any significant mineralization; however, a large amount of disseminated pyrite was intersected over a wide interval in this hole which explains the large high-chargeability IP anomaly in the Mike target area.

Table 10-5: Summary of 2013 Drilling Results

Zone	Drill Hole ID	From (m)	To (m)	Interval (m)	Cu (%)	Au (g/t)	Mo (%)	Ag (g/t)
Mike	SCK-13-431	*no significant results—exploration hole*						
North Paramount	SCK-13-432	18.0	286.1	268.1	0.237	0.238	0.0155	2.22
	including	67.4	166.5	99.2	0.361	0.401	0.031	4.04
	including	108.0	139.2	31.2	0.561	0.541	0.028	6.52
Paramount	SCK-13-433	400.8	456.0	55.2	0.275	0.040	0.022	1.85
Paramount	SCK-13-434	*no significant results—hole abandoned before target depth*						
Paramount	SCK-13-435	239.0	665.5	426.5	0.324	0.112	0.021	1.13
	including	307.0	335.0	28.0	0.817	0.324	0.051	1.84
	including	550.0	571.0	21.0	0.740	0.094	0.065	2.81
Discovery	SCK-13-436	50.5	68.5	18.0	0.088	0.117	0.001	0.56
Paramount	SCK-13-437	*no significant results—geotechnical hole*						
Paramount	SCK-13-438	*no significant results—hole abandoned before target depth*						
Paramount	SCK-13-439	59.0	71.3	12.3	0.202	0.035	0.001	1.49

10.4 Schaft Creek JV, 2015

The Schaft Creek JV completed five diamond drill holes in the LaCasse target area during the 2015 field season, for a total of 2,634 m drilled (Figure 10-1, Table 10-6).

Table 10-6: Collar Details for Holes Drilled during the 2015 Drill Program

Collar ID	Easting	Northing	Elevation (m)	Collar Azimuth	Collar Dip	Depth (m)
SCK-15-440	378232	6363815	1480	205	60	624
SCK-15-441	378037	6363422	1200	215	65	405
SCK-15-442	378331	6364121	1639	225	70	555
SCK-15-443	378003	6363717	1312	215	60	543
SCK-15-444	378304	6363191	1204	260	60	507

Note: Collar locations (Easting, Northing, and Elevation) were measured using a differential GPS. Azimuth is recorded relative to true north. The coordinate system used is UTM NAD 83 Zone 9N.

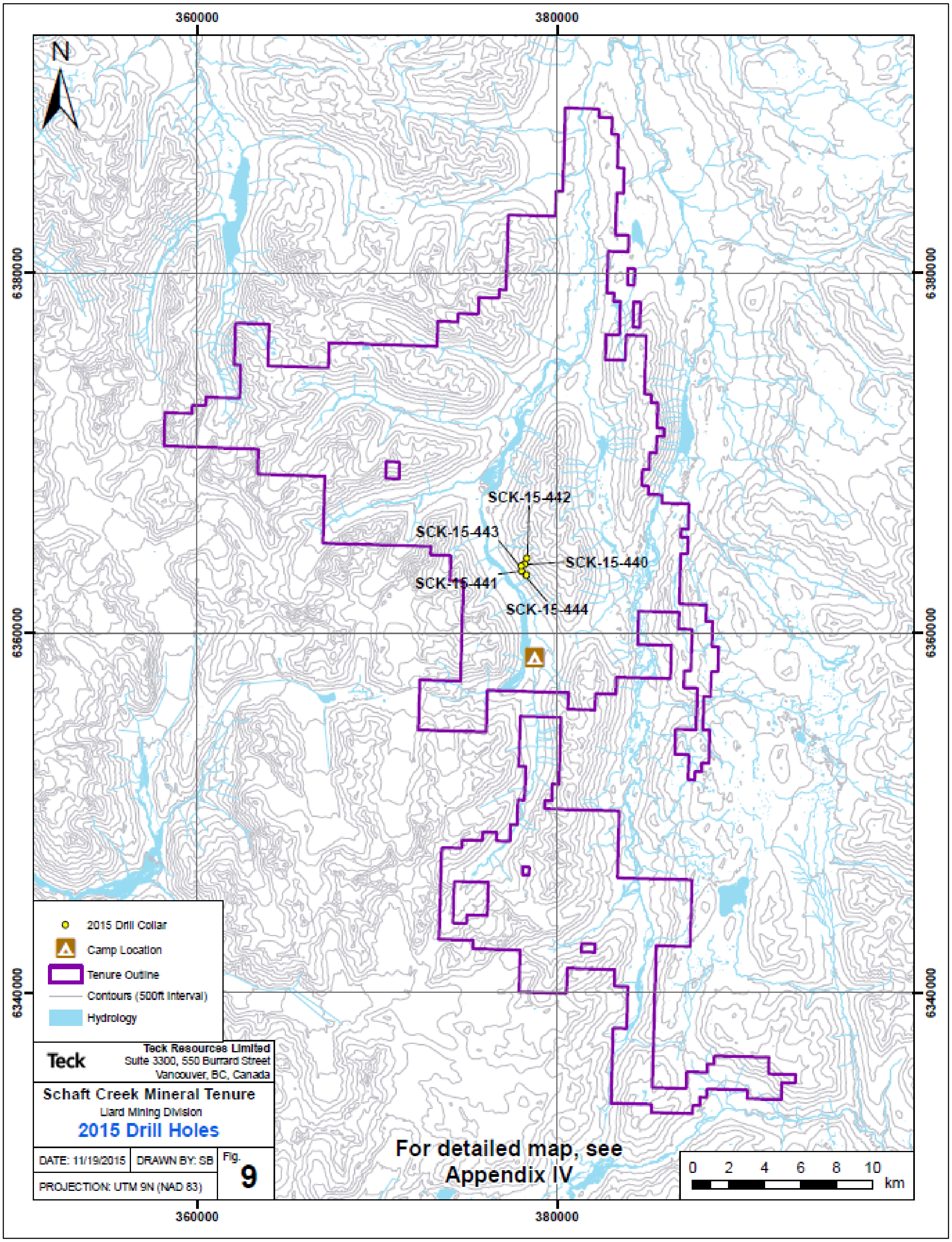
Prior to drilling, collar locations were located and staked by project geologists using a handheld GPS and Brunton compass. Archaeological assessment of the proposed collar locations was conducted by Rescan Environmental Services Ltd. (Rescan); no archaeological concerns were identified at any of the proposed collar locations. Timber drill pads and secondary timber platforms were constructed at each collar location by Rugged Edge Holdings Ltd. of Smithers, British Columbia.

10.4.1 Diamond Drilling Procedures

Hy-Tech Drilling Ltd. of Smithers, British Columbia, was responsible for performing the drilling by using a helicopter-portable FlyTech 5000 drill rig. The drill rig was equipped with a rod manipulator to reduce manual handling and a centrifuge system to manage drill cuttings and reduce water consumption. The cuttings management system was only functional when there was water return; this occurred only sporadically throughout the program. All drilling was completed using HQ tooling, with the exception of SCK-15-440, which was reduced to NQ at 393.4 m. Downhole survey measurements of drill hole orientation were collected using a Reflex EZ-Shot tool, with measurements typically collected every 50 m. Downhole depth was measured from the ground, rather than from the top of the drill string on the drill pad, because the pad was typically several metres above the steeply dipping slope. Drilling activities were supported by an AS350D2 helicopter on contract from Lakelse Air Ltd. of Terrace, British Columbia.

Following completion of drilling, the drill casing was broken near the ground and the holes were capped. Each collar location was surveyed using a differential GPS, with the instrument measuring the position where the casing met the ground. The timber drill pads were disassembled and each site was cleared and reclaimed. Drill cuttings captured by the centrifuge system were flown in bags to a historical un-reclaimed drill pad in the Liard Zone, where they were emptied into a sump. Following the completion of the drill program, this sump was backfilled.

Figure 10-2: 2015 Drill Holes Location Map



10.4.2 Core Logging Procedures

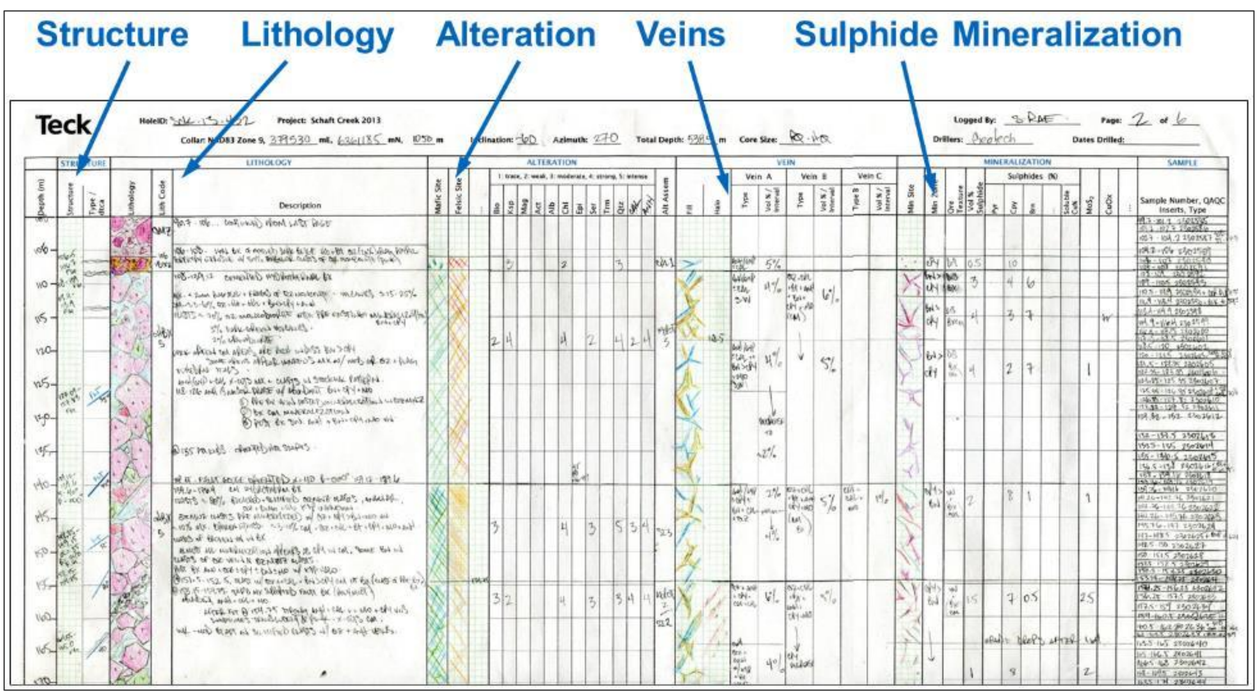
Drill core was logged in camp at the core-logging facility. Upon arrival, the core boxes were reviewed by the logging geologist to ensure there were no mistakes with run-blocks or box labels. Core boxes were then labeled with hole-id and "from-to" intervals on metal tags on the front of the boxes. The core was cleaned with a brush, and then the boxes were photographed while wet. Additional core photographs at a smaller scale were also collected at the discretion of the logging geologist to capture specific geological details.

Geotechnical measurements of core recovery and RQD were collected using a tape measure. Magnetic susceptibility measurements were collected using KT-9 or KT-10 magnetometers. Geotechnical and magnetic susceptibility measurements were initially recorded using a Trimble Juno T41 handheld computer; however, this was abandoned partway through the field program in favour of paper-based data recording. This data was subsequently entered into Excel and imported into the acQuire database.

Geological logging was conducted using paper logging sheets, following a modified version of the Anaconda-style logging sheet (Figure 10-3). This method captures interval-based information, while incorporating graphical logging of lithology, structure, veins, alteration, and mineralization data, with a focus on mineral paragenesis and textural relationships. Information from these paper logs was digitized into an Excel format, and then checked and validated before being imported into the acQuire database.

Throughout the program, efforts were made to ensure consistency between loggers. This was done through group discussions regarding drill core geology and by developing a library of polished reference samples.

Figure 10-3: Example of Modified Anaconda-Style Drill Log Used at Schaft Creek in 2013



10.4.3 Diamond Drilling Results

The drilling program completed during 2015 focused on testing the LaCasse Zone, located 3 km to the north-northwest of the Schaft Creek deposit area. The LaCasse area contains outcropping chalcopyrite-pyrite mineralization associated with hydrothermal-intrusive breccias and sheeted quartz veins, as well as disseminated sulphide mineralization. Mineralization in this area has been recognized by various historical workers (e.g., Lamelle, 1966; Betmanis, 1978; Raven, 1979; Luckman, 2005; Bradford, 2006; Greig and Kreft, 2009); however, there is no known record of previous drilling in this area.

The 2015 drilling program was designed to test a large area of outcropping sulphide mineralization along a strike length of approximately 1 km. Five holes were completed (2,634 m), with a spacing of 200 m to 300 m of horizontal distance between drill collars. Major rock types intersected by the drilling at LaCasse include fine- to medium-grained, weakly porphyritic granodiorite to quartz monzonite, pyroxene-phyric andesitic volcanic rocks, andesitic dikes, and post-mineral syenite to rhyolite dikes with distinctive flow banding near the intrusive margins. Two of the drill holes intersected hydrothermal-intrusive breccia containing rounded to subangular clasts of granodiorite and pyroxene-phyric andesite, with a pink colored matrix of quartz and K-feldspar. Locally, this breccia matrix appears to contain plagioclase phenocrysts and is interpreted to have an intrusive origin, whereas in other areas, the matrix is aphanitic and is interpreted to be hydrothermal in origin. The cement of the breccia is generally quartz and calcite, although rarely the breccia contains small intervals of chalcopyrite-cement with sulphide content up to 3% to 5% locally. The breccia also contains some clasts comprised of quartz vein fragments, and some clasts containing abundant disseminated chalcopyrite. These clast types suggest that some brecciation postdates the mineralization, and that there may have been multiple brecciation and mineralization episodes.

Major alteration assemblages include a potassic assemblage of K-feldspar-biotite ± hematite that is overprinted or crosscut by a sodic-calcic assemblage of chlorite-epidote-calcite-hematite ± albite. Both assemblages are associated with chalcopyrite-pyrite mineralization, although the potassic alteration is more closely correlated with the sulphide mineralization. Magnetite occurs locally as part of the potassic alteration assemblage within the granodiorite and within the chlorite-altered andesitic volcanic rocks adjacent to the granodiorite. The syenite to rhyolite dikes, which crosscut all other rock types and alteration assemblages, are associated with an alteration assemblage of clay-chlorite-sericite.

Sulphide mineralization includes quartz-chalcopyrite-pyrite-molybdenite veins with potassic halos, chalcopyrite-pyrite mineralization replacing mafic phenocrysts, disseminated and fracture-controlled chalcopyrite-pyrite with minor bornite, and rare well-mineralized breccia clasts containing disseminated chalcopyrite-pyrite with minor bornite. Generally, sulphide mineralization is sparse and low grade (0.3% to 0.5% total sulphide), although locally disseminated sulphide mineralization is more abundant (up to 2% to 3% locally). Quartz-sulphide veins occur throughout all of the drill holes, but are generally sparse (1 to 5 veins per meter) and thin (0.2 cm to 1 cm thick). Locally, quartz-sulphide veins are sheeted and more abundant (5 to 20 veins per meter); however, the overall sulphide content of these sheeted vein zones is still relatively low (0.4% to 0.7% total sulphide). Very fine-grained bornite occurs locally, particularly in SCK-15-444, and is associated with a notable increase in gold grade relative to copper grade, corresponding to an elevated Au:Cu ratio. Sparse, irregular clots of partially oxidized chalcopyrite and minor bornite also occur within the syenite to rhyolite dikes. However, based upon these textures, these sulphides are interpreted to have been “digested” or remobilized into these dikes from other rock types.

The drilling at LaCasse intersected rock types, alteration assemblages, and sulphide mineralization styles that are generally comparable to what had been mapped at surface. In many areas, features such as intrusive contacts, zones of sheeted veins, faults, and post-mineral dikes could be correlated with confidence between drill holes and geological mapping at surface. In some areas, the abundance and thickness of post-mineral dikes were greater than expected. These dikes introduced a significant amount of dilution to some of the mineralized zones (e.g., SCK-15-440).

In some key areas, the drilling did not correlate with previous mapping: in particular, SCK-15-441 failed to intersect any hydrothermal breccia at depth, although the collar location for this hole was positioned near several outcrops that contain breccia textures. The breccia body is therefore interpreted to have an irregular shape and/or to be offset by northeast-trending faults that parallel the prominent gullies in the slope. This fault set was intersected by at least two drill holes, and both encountered large zones of gouge and intensely fractured rock.

A summary of significant mineralized intercepts from the 2015 drilling is presented in Table 10-7. DDH SCK-15-444 intersected an interval or 182.5 m grading 0.20% Cu and is the southern-most hole completed in 2015, and is the closest to the drill holes completed in 2011 and 2012 in the Discovery Zone.

Table 10-7: Summary of Results from the 2015 Drill Program at the LaCasse Target

Drill Hole ID	From (m)	To (m)	Interval (m)	Cu (%)	Mo (%)	Au (g/t)
SCK-15-440	*Mineralization hosted in granodiorite cut by andesite and syenite dikes adjacent to contact with volcanics*					
	120.5	167.6	47.1	0.13	0.004	0.2
including	121.5	134.3	12.8	0.21	0.007	0.69*
including	158.3	165.5	7.3	0.24	0.009	0.06
SCK-15-441	*Planned test of breccia – not intersected* *Chalcopyrite in disseminations and veins in granodiorite*					
	120.5	150.0	29.5	0.1	0.001	0.01
	178.5	205.5	27.0	0.09	0.016	0.06
SCK-15-442	*Test of quartz-chalcopyrite vein target*					
	439.7	454.1	14.4	0.13	0.004	0.01
SCK-15-443	*Mineralization hosted in hydrothermal and intrusive breccia*					
	354.6	369.0	14.4	0.16	0.002	0.02
	404.0	424.0	20.0	0.17	0.011	0.02
	454.0	463.0	9.0	0.08	0.001	0.21

table continues…

Drill Hole ID	From (m)	To (m)	Interval (m)	Cu (%)	Mo (%)	Au (g/t)
SCK-15-444	*Test between Lacasse and Discovery Disseminated and vein chalcopyrite in granodiorite*					
	283.5	466.0	182.5	0.2	0.002	0.02
including	283.5	363.0	79.5	0.29	0.002	0.02
including	283.5	313.5	30.0	0.4	0.005	0.05
including	337.8	361.9	24.1	0.34	0.001	0.01

*Au grade is mostly due to two assays: 1.1 g/t and 6.7g/t.

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Schaft Creek JV, 2013

11.1.1 Sample Transportation and Security

Upon completion of logging and core cutting, cut core samples were collected in plastic sample bags secured with zip ties. For shipping, these plastic sample bags were put into numbered rice sacks and secured with zip ties. Rice sacks containing samples were picked up by a Cessna 208 Caravan operated by Northern Thunderbird Air at the exploration camp airstrip and flown directly to Smithers where Acme Labs personnel received the shipment.

11.1.2 Drill Core Preparation and Analysis

For the 2013 program, all sample preparation was conducted by Acme Labs in Smithers using the preparation method R200-1000. Sample preparation included drying, crushing, riffle splitting, and pulverizing. Drying was accomplished using an oven to dry the samples at 60°C for 24 hours to 48 hours. Prior to crushing, the crusher was cleaned using a charge of quartz sand. An additional blank sample was inserted at the start of each sample batch to limit contamination between sample batches. Samples were crushed to 80% passing 10 mesh using a Terminator crusher. This coarse crush material was riffle split three times for homogenization, and then split down to 1,000 g with the remaining material set aside as a coarse reject. The 1,000 g split was placed in a barcoded envelope and sent to pulverizing. For crusher duplicate samples, an additional split was collected at this stage and analyzed as a separate sample. Crushed samples were then pulverized using a bowl and puck pulverizer to a tolerance of 85% passing 200 mesh.

Following sample preparation, sample assays were completed on pulps using a variety of routine analysis methods. These methods are detailed below:

1. Mineralized material-grade assays for copper, molybdenum, and silver were determined by 4-acid digest with ICP-ES finish (Acme Group 7TD). This assay package also includes results for major elements (e.g., Fe, K, Al), other commodity elements (e.g., Zn, Pb), and trace elements (e.g., Ni, Bi). The analysis method uses a 0.5 g sample split, which is digested in a hot 4-acid solution and then taken to dryness. The sample is then leached with HCl and analyzed using an ICP-ES finish.

2. Mineralized material-grade assays for gold were determined by fire assay with atomic absorption (AA) finish (Acme Code G601). This analysis method includes lead-collection fire assay fusion on a 30 g sample split. After fusion, the doré bead is digested in HCl and HNO_3, and then analyzed for gold with an AA finish.

3. Concentrations of trace elements were determined by aqua regia digest with ICP-MS finish (Acme Group 1DX). This analytical method uses a 0.5 g sample split, which is digested in a hot (≈95°C) modified aqua regia solution (HCl, HNO_3, and H_2O). Following digestion the sample is analyzed by ICP-MS.

4. Concentrations of carbon and sulphur were determined by LECO analysis (Acme Group 2A12). This analytical method uses a 0.2 g sample split and LECO analysis.

5. Whole-rock lithogeochemistry was determined using a multi-part analytical package (Acme Code 4AB1). This analytical method includes results for major oxides determined by lithium borate fusion and ICP-ES finish (Code 4A), rare and refractory elements determined by lithium borate decomposition and ICP-MS finish (Code 4B), base and precious metals determined by aqua regia digestion and ICP-MS finish (Group 1DX), low to ultra-low trace element concentrations determined by aqua regia digestion and ICP-MS finish (Code 1F04), and loss on ignition (LOI) determined by weighing a sample split before and after ignition.

The QP is satisfied that sample preparation and analysis were carried out in an appropriate manner to provide suitable analyses and to maintain data integrity.

11.2 Schaft Creek JV, 2015

11.2.1 Core Sampling Procedures

All drill holes completed during the 2015 field program were sampled from the top of bedrock to the bottom of hole. Sample intervals were determined by the logging geologist. Sample intervals were selected to conform to natural variation in the geology, with breaks at all geologic contacts and major structures, and intervals selected to represent variation in alteration and mineralization as accurately as possible. Sample intervals generally ranged from 1 m to 2 m, with nearly all samples within the range of 0.5 m to 3 m; the shortest sample was 0.4 m and the longest 3.5 m. Sample intervals were marked in the box with metal tags and barcoded laboratory tags corresponding to tags included in submitted sample bags.

QA/QC samples were inserted into the core sample sequence by the core logging geologist. Three different QA/QC sample types were inserted, including standards, field blanks, and field duplicates. One of each of these QA/QC sample types was inserted into each batch of 20 samples, for a total of three QA/QC samples per 20 samples. Standards were matrix-matched and also matched to the estimated Cu-Mo-Au grades of the surrounding samples. Barren granite was used as blank material. Field duplicates consisted of quarter core duplicates, with half the core left remaining in the core box.

Core cutting was conducted on site in the core-cutting facility adjacent to the core logging area. Core cutting was performed by Northern Labour Services, under supervision of the geologists on site. Cutting was done using electrical core saws. Following the completion of cutting and sampling, all core boxes from the 2015 program were placed into newly built racks in the core storage area.

During the logging process, sample intervals were selected for additional lithogeochemical study. Selected sample intervals were flagged on the laboratory requisition form requesting separate 250 g crusher splits be created for these samples for later lithogeochemical analysis. These splits were held by the lab until the end of the program and were run under a separate work order. Intervals selected for lithogeochemistry were selected for best geological consistency throughout the sample interval, with a minimum of veins or other dilution.

Samples from the 2015 drilling were also selected for petrographic thin sections and K-feldspar staining; these were collected after the core cutting was completed. For petrographic thin sections, small billets were cut and labeled to indicate the area to be preserved in the thin section. For K-feldspar staining, wafers or slabs were cut and labeled to indicate the area to be stained. Samples for petrographic thin sections and K-feldspar staining were submitted to Vancouver GeoTech Labs of Vancouver, British Columbia for thin section preparation.

The QP believes that sampling was comprehensive and that the methodology applied is suitable for the deposit and meets industry standards.

11.2.2 Sample Transportation and Security

Samples including drill core, geometallurgical samples, rock samples, and soil samples were all transported in a similar fashion. Drill core, geometallurgical samples, and rock samples were collected in plastic sample bags secured with zip ties. Soil samples were collected in Hubco bags and allowed to air-dry inside a building on site, before being placed inside plastic sample bags. Barcoded sample label tags were placed inside all sample bags, and the corresponding sample number was written on the outside of the bag. For shipping, the samples were then put into numbered rice sacks and secured with zip ties. Rice sacks were batched into groups of 10 to 30 bags, and each batch was transported as a group. Rice sacks were transported by helicopter to a secure staging area at Ch'iyone Camp, located on the Galore Creek access road. From Ch'iyone Camp, sample batches were picked up and transported to the Bureau Veritas (BV) Laboratories preparation lab in Smithers by Bandstra Transportation Systems Ltd. Sample batching, shipping, and shipment pick-up were supervised by geologists on site.

11.2.3 Sample Preparation and Analysis

For the 2015 program, all sample preparation was conducted by BV Mineral Laboratories in Smithers. The preparation methods used are detailed below.

Samples were dried, crushed, split, and pulverized (BV laboratory code PRP80-1000). Drying was accomplished using an oven to dry the samples at 60°C for 24 hours to 48 hours. Prior to crushing, the crusher was cleaned using a charge of quartz sand. An additional blank sample of barren granite was inserted at the start of each sample batch to limit contamination between sample batches (reported as ROCK-SMI in the data certificates). Samples were crushed to 80% passing 10 mesh using a Terminator crusher. This coarse crush material was riffle split three times for homogenization, and then split down to 1,000 g with the remaining material set aside as a coarse reject. The 1,000 g split was pulverized using a bowl and puck pulverizer to a tolerance of 85% passing 200 mesh.

Routine analysis of drill core:

1. Mineralized material-grade assays for copper, molybdenum, and silver were determined by 4-acid digest with ICP-ES finish using a 0.5 g sample split (BV laboratory code MA370). This assay package also includes results for major elements (e.g., Fe, K, Al), other commodity elements (e.g., Zn, Pb), and trace elements (e.g., Ni, Bi).

2. Mineralized material-grade assays for gold were determined by fire assay with AA finish using a 30 g sample split (BV laboratory code FA430).

3. Concentrations of trace elements were determined by hot aqua regia digest with ICP-MS finish using a 0.5 g sample split (BV laboratory code AQ200).

4. Concentrations of carbon and sulphur were determined by LECO analysis using a 0.2 g sample split (BV laboratory code TC003).

Lithogeochemical analysis of surface rock samples and select drill core samples:

1. Concentrations of major element oxides were determined by lithium borate fusion with ICP-ES/ICP-MS finish using a 5 g sample split (BV laboratory code LF302).
2. Concentrations of lithophile elements including rare earth elements to low and ultra-low levels were determined by lithium borate fusion with ICP-MS finish using a 5 g sample split (BV laboratory code LF100).
3. Concentrations of trace elements to low and ultra-low levels were determined by hot aqua regia digest with ICP-MS finish using a 30 g sample split (BV laboratory code AQ252-EXT).
4. Concentrations of carbon and sulphur were determined by LECO analysis using a 0.2 g sample split (BV laboratory code TC003).
5. LOI was reported as part of the major element analysis suite.
6. Overlimit analyses were applied. If gold by AQ252 reported > 0.1 pmm then gold by FA430 (30 g fire assay) was applied. Likewise, if copper by AQ250 reported > 1,000 pmm then copper by MA370 (4-acid digest with ICP-ES finish) was applied.

The QP is satisfied that the sample preparation and analytical procedures are appropriate for the deposit and industry standard.

11.3 QA/QC

A life-of-project QA/QC review was requested in advance of and in support of the 2017 Schaft Creek resource estimation, completed by the Schaft Creek JV. Given the history of the Project and the large number of methods used through the life of the Project, the suite of elements evaluated has been limited to seven: silver, arsenic, gold, copper, molybdenum, rhenium, and sulphur. Even with this suite, 114 individual methods need to be assessed for precision and accuracy.

No spatial limit is applied to the sample locations, and samples from outside the current resource model were included in this QA/QC evaluation.

The suitability of all data is assessed against current and historical best practice recommendations given that some data in previous resource estimates is derived from drilling in the 1960s.

11.3.1 QA/QC Review of Copper Fox and More Recent Data

There are several plots commonly reproduced in the sections below. The control charts (e.g., Figure 11-3) have all the data ordered by drill hole age. The database does not have complete information on the return dates for all the generations of assay data, so it is not possible to order the data by report date, which would be more common practice given that this dataset contains several reassay campaigns. There are several sets of control lines on these charts. The blue lines are the certificate values, with the solid line as the average. The dashed lines are the average ± 2SD (standard deviation) and the dotted lines are the average ± 3SD. Note that these are missing for some or all of the elements, depending on the extent and availability of certification data. The red lines are these same limits derived from the data itself. Using the recommendations of AMEC summarized in Simon (2014), these limits are used to assess whether the precision of the process is under control. For each certified reference material (CRM) and for each element, the outliers were removed by successive Grubbs tests. Once the null hypothesis that the furthest point from the centre of the population was an outlier cannot be rejected, mean and standard deviations of the coherent dataset were calculated. If a data point is outside of six standard deviations from the mean, then the data point is not plotted and there is a break in the line between points.

The bias plots show the certified (or in some cases recommended) means against the reported data (e.g., Figure 11-4). These plots show a 1:1 line in red and the slope of regression in blue, with the error in regression as a shaded zone around this. Note that this is the regression error, not the prediction error. Below this are statistics for the regression, which assess the significance of the bias. If 1.00 is within error of the slope, then there is no evidence of bias. If there is evidence of bias, Simon (2014) recommends that it should be within 5%, so a slope of between 0.95 and 1.05, in order to be acceptable.

There are two blank plots used (e.g., Figure 11-5). The top plot shows the blank values ordered by drill-hole age. The lower plot shows the blank values plotted against the average of the three previous samples and one subsequent sample. This is not ideal, but there are generations of data where the blanks were inserted only at the beginning of the drill hole and generations where it is more random. Blanks at the beginning of a job offer no measure of the degree of contamination from sample into the blank. The red line is the mean +3SD of a coherent population of blank data, again assessed after the removal of outliers using successive Grubbs tests. This method allows the assessment of blanks independent of the detection limit of the method, which is an inherently contradictory way of defining an acceptable level of contamination. The statistics below the graphs show the number of samples that exceed this blank threshold and the probability that the slope of correlation is not different to 0. If a sample has a < 5% probability that the slope is not different to 0, then there is evidence of systematic carryover from samples into the blank. Simon (2014) recommends that there should be no systematic carryover evident in blanks.

The duplicate plots show the minimum of a sample pair against the maximum of that pair. All data are therefore plotted above the red 1:1 line. For each type of duplicate (sample, crush, pulp) there is a threshold for acceptable repeatability. This uses the hyperbolic method recommended by Simon (2014) and all the parameters for allowable variability defined therein. This method allows for greater variability nearer the detection limit. The variability at high concentrations ranges from tight precision in pulp duplicates and more variability at higher stages of sample reduction. The variability limits for gold are higher than for other elements.

A significant component of this review is to define the way that acQuire exports data where there is more than one data source for an interval, whether that means the interval was resampled or the pulp was reassayed. In order to define this, the QA/QC of each method must be reviewed and ranked. Once this is complete, acQuire can export the most appropriate data for each sample for each element. In order to identify them, acQuire exports these with the suffix “BESTEL”, and through the course of this document, this derived column is referred to as the BESTEL column.

11.3.1.1 Silver

There are 21 silver methods including the historical data, which is attributed as Ag_UNK_UNK_opt. For the purpose of this review, all methods reported in oz/t are converted into ppm; these then carry both the initial and converted units in their column name, for instance Ag_UNK_UNK_opt_gpt. Of the 20 methods that have laboratory, method, and unit attribution, there are a range of detection limits and assumed precisions (geochemical methods vs. assay methods) and with a range of supporting QA/QC data. These are summarized in Table 11-1, which has the method detection limit, the number of samples with a valid result, the number of samples greater than the detection limit, the number of samples greater than three times the detection limit, the number of associated standards and blanks, and the number of duplicates (including laboratory duplicates). Also in this table are the number of occurrences in the previous BESTEL-derived column.

Table 11-1: Summary of the Available Silver Data in the Schaft Creek Database

Method	DL	No. Samples	No. > DL	No. > 3DL	No. in BESTEL	Standards + Blanks	Duplicates*
Ag_1DX1_ACME_ppm	0.1	2444	1528	824	2091	107	208
Ag_1EX_ACME_ppm	0.1	12368	10454	7946	12364	1498	2024
Ag_1F04_ACME_ppb	0.002	111	111	111	111	0	0
Ag_1F06_ACME_ppb	0.002	10	10	10	10	0	3
Ag_7AR2_ACME_gpt	2	37	16	2	10	3	3
Ag_7TD2_ACME_gpt	2	21854	4069	305	6922	1673	2865
Ag_AQ200_ppm	0.1	2110	548	236	57	204	101
Ag_AQ250_ppb	0.002	115	115	115	0	5	5
Ag_AQ252_ppb	0.002	28	28	28	28	0	0
Ag_FA_LOR_gpt	0.1	1089	1061	947	1089	55	27
Ag_G613_ACME_gpt	?	6	0	0	2	0	2
Ag_ICP_IPL_ppm	0.5	7717	2494	1120	2182	363	151
Ag_MA200_ppm	0.1	67	65	57	2	0	0
Ag_MA370_gpt	2	2121	39	1	2115	201	101
Ag_MEICP61_ALS_gpt	0.5	14	6	2	0	3	1
Ag_MEICP61a_ALS_gpt	1	443	234	34	136	58	12
Ag_MEICP61a_ALS_ppm	1	479	227	25	285	9	6
Ag_MEMS61_ALS_ppm	0.02	926	922	829	612	0	0
Ag_MEMS62_ALS_ppm	0.02	642	639	616	0	67	17
Ag_OG62_ALS_gpt	1	278	148	14	0	0	1
Ag_UNK_UNK_opt	0.34286	8791	7870	6047	8795	0	0

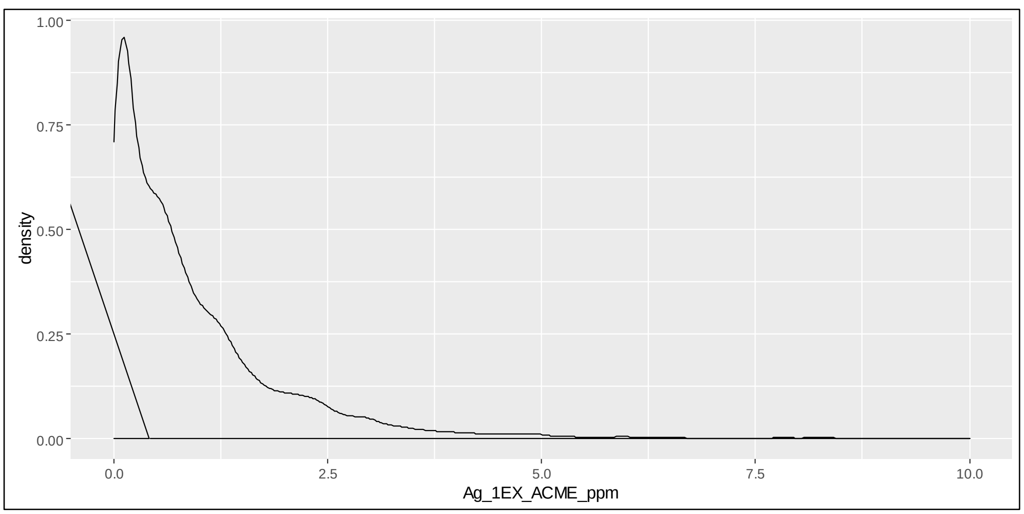
Table 11-1 shows that the detection limits range from 0.002 ppm to 2 ppm. To judge whether this is suitable, the distribution of primary samples must be reviewed in a robust, low detection limit method of analysis. The Ag_1EX_ACME_ppm method is a low detection limit geochemical method, providing analysis of a mixed acid digestion and which is shown to provide acceptable quality data. It has some 12,000 samples, and therefore, provides a good estimate of the whole population. This digestion is at times criticized as a method for silver because of the complex interferences from ZrO on an ICP-MS, but these interferences were well understood and were effectively corrected for by ACME at the time these data were collected.

Figure 11-1 shows the population density plot for silver by 1EX. Silver has a median of 0.4 ppm, a 95th percentile of 3 ppm and a 99th percentile of 5.9 ppm. This strongly suggests that the 1 ppm and 2 ppm detection limit methods will significantly limit the number of available valid analyses. The practicality of this is evident in Table 11-1 in the row for the Ag_7TD2_AMCE_gpt. There are nearly 22,000 valid analyses, but only 4,069 have a result of the detection limit or greater, and only 300 samples are greater than three times the detection limit. Figure 11-2 shows a statistically significant correlation between the results where there are comparable data by both methods. On the right is the same data but with only the samples at or below the detection shown. The majority of samples with a 2 ppm reported concentration by the assay method have a comparable low-level analysis. In fact, more than 90% of samples with a 2 ppm assay concentration have > 1 ppm reported by the low-level method and more than 80% have > 1.5 ppm reported. Therefore, very few samples with an above detection limit assay will result in an overestimation of silver concentration. It is therefore recommended that samples with a result of the detection limit or greater for these methods be included in the _BESTEL calculation. Samples with a result reported < 2 ppm should have an inputted value based on the regression models for silver.

Because the 1EX samples seem to have been selected with a strong bias towards providing good low-level analysis on low concentration samples, the removal of < 2 ppm 7TD Ag data affects only ~20 primary samples. All other samples either have a 1EX_ACME, ICP_IPL, or FA_LOR low-level analysis.

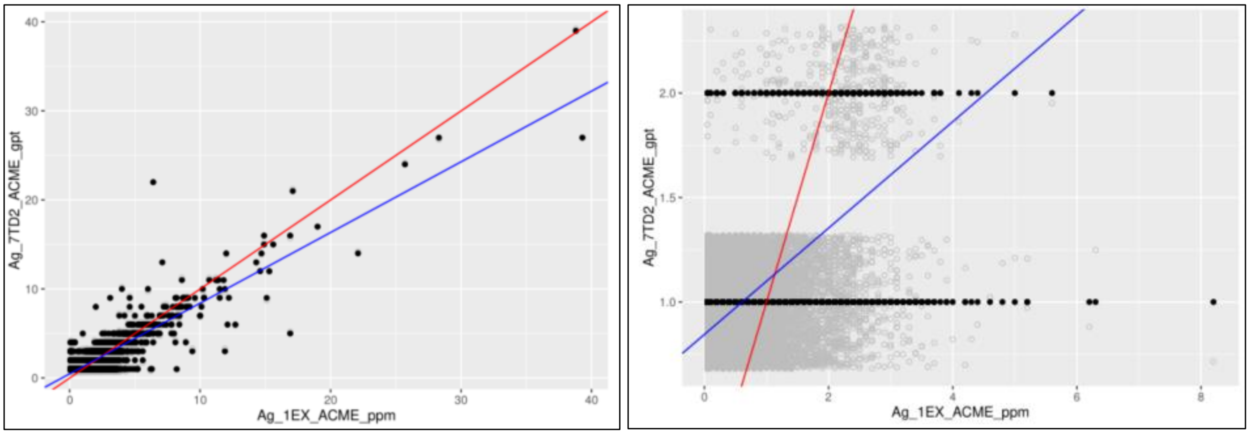
The MA370_ACME data and OG62_ALS data appear biased high, but all samples have an accompanying low-level method that provides more precise data, so the data can be excluded from the _BESTEL calculation with no consequence.

Figure 11-1: Population Density Plot for Ag_1EX_ACME_ppm, Presumed to be Representative of the Entire Population



The left graph in Figure 11-2 shows the full range of data, and the right graph shows the data at below detection (shown as 1 ppm by the assay method) and at detection (shown as 2 ppm by the assay method). The red line is a 1:1 line and the blue line is the standard major axis (SMA). The grey cloud is essentially a population density representation at the particular assay concentration.

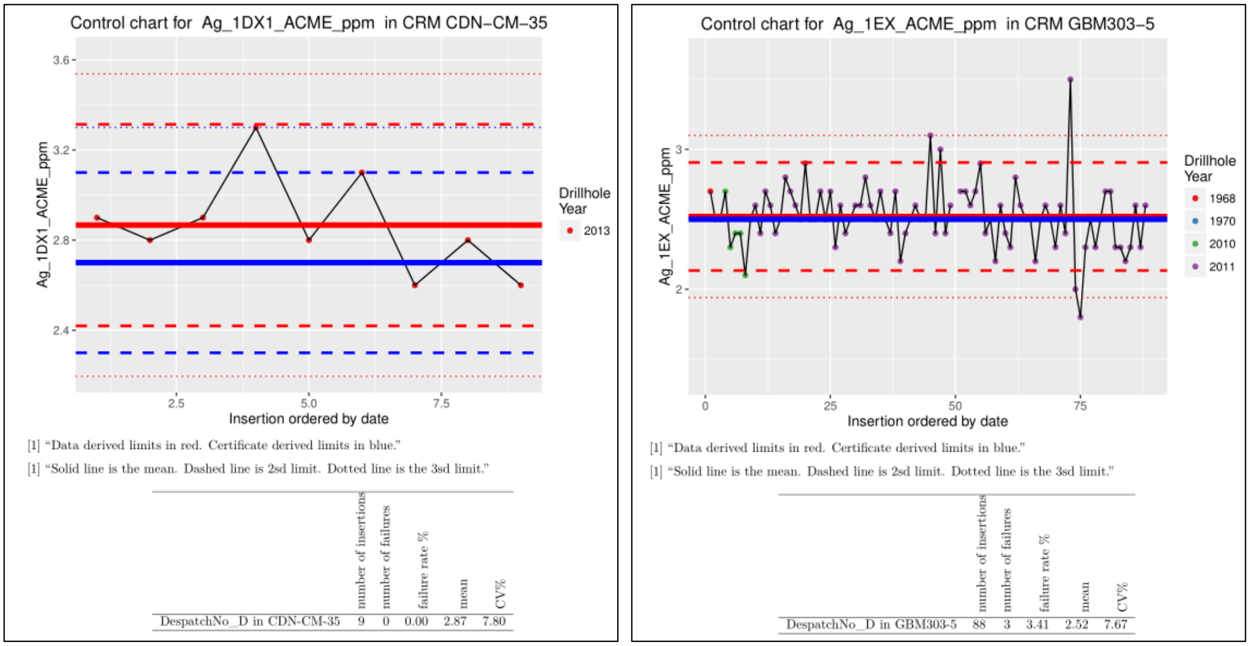
Figure 11-2: Scatter Plots of Silver by a Geochemical Method and an Assay Method



The remaining methods for silver can be grouped as ACME/BV (BV acquired ACME in 2014) aqua regia geochemical methods and mixed acid digestion methods, ALS mixed acid digestion methods, Loring assay data, and Independent Plasma Labs data.

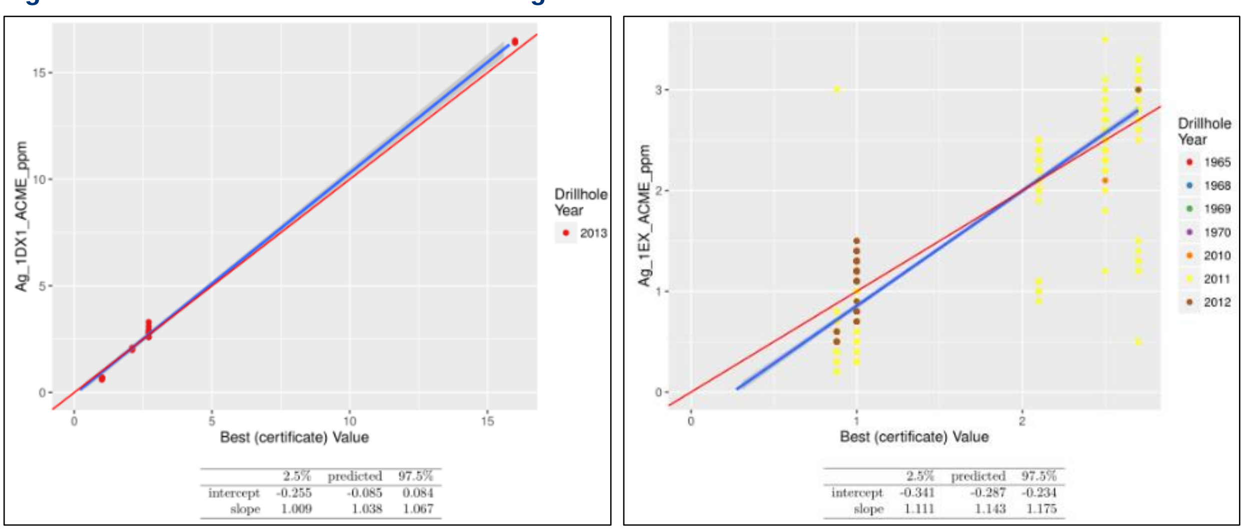
The ACME/BV data is primarily 1DX (aqua regia) and 1EX (4-acid digestion) data, with minor additional 1F## and AQ2## (aqua regia) data and is shown in Figure 11-3. There are limitations to the completeness of QA/QC assessment that can be completed for silver as the majority of CRMs do not have a certified value for silver. The CRMs for both the 1EX and 1DX methods have an acceptable CV% given the proximity to the method detection limits. There is also an acceptably low failure rate for these data in all cases.

Figure 11-3: Example Control Charts for the 1DX and 1EX Ag Data



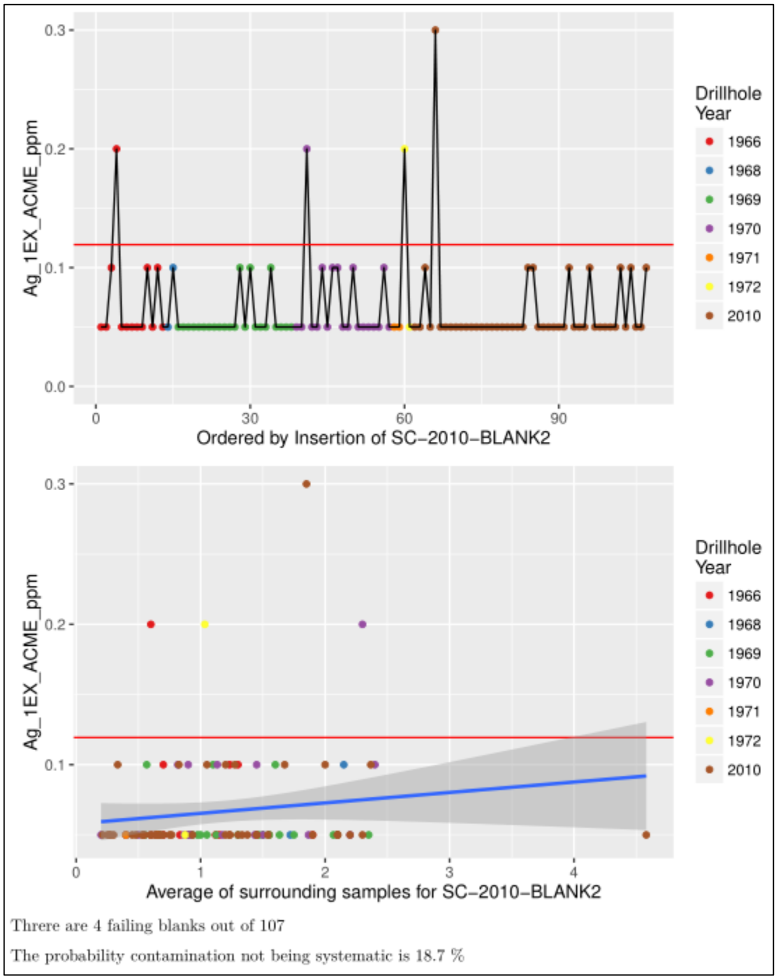
There is an acceptable bias in the 1DX data and the 1EX data (see Figure 11-4) from the samples with a CRM value for silver. The slope of regression is not significantly different to 1.00 if the low concentration CRM at 0.88 ppm is excluded (this is just a recommended value rather than a certified value). It is clear looking at the 1EX data that there are a number of outliers in these datasets and at least some of these are instances where the wrong CRM has been inserted.

Figure 11-4: Bias Plots for 1DX and 1EX Ag Data



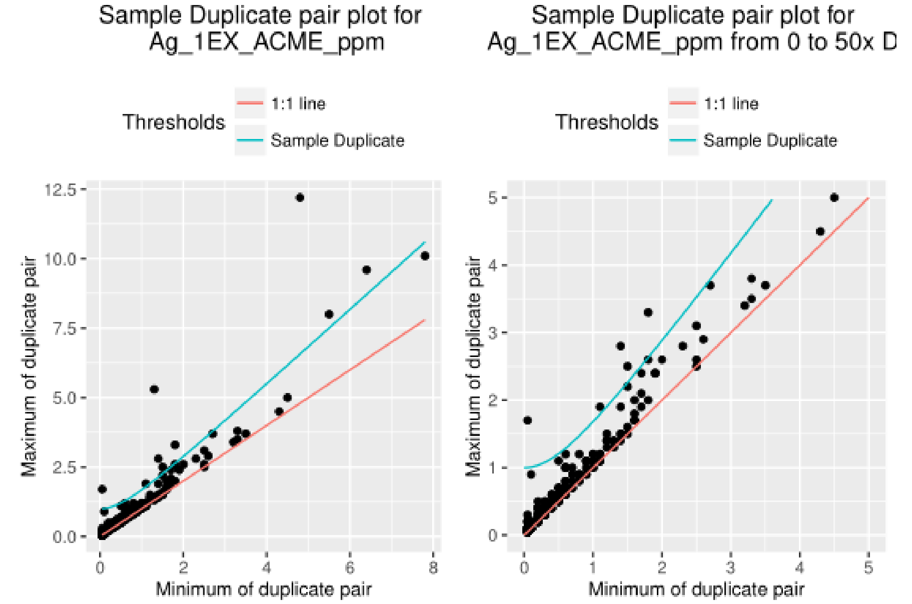
The blanks, summarized in Figure 11-5, show an acceptable failure rate and no evidence of systematic contamination.

Figure 11-5: Blank Plots for Ag by 1DX

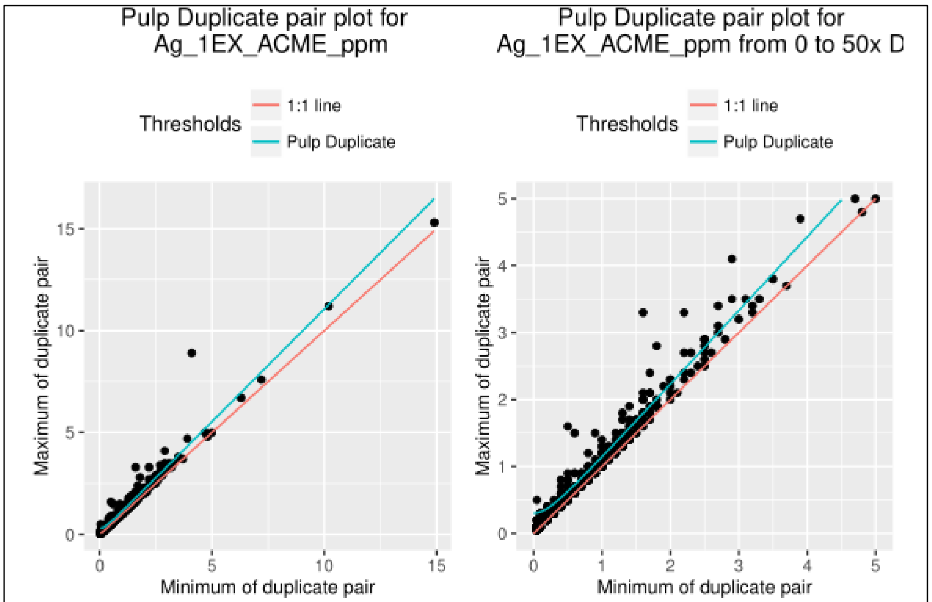


The sample duplicate plots for the 1DX method all show overall acceptable data. The 1EX data (see Figure 11-6) shows good repeatability of the overall sampling technique, but the pulp duplicates have a failure rate exceeding 10% (Figure 11-6). This suggests the analytical imprecision is a significant source of the overall variance, which is acceptable as a whole. If these data were being received on a live program, it would be recommended to consider changing to a higher precision method so that the analytical variance was minimized, but this does not make the overall sampling variance problematical.

Figure 11-6: Samples and Pulp Duplicate Pair Plots for Ag by 1EX



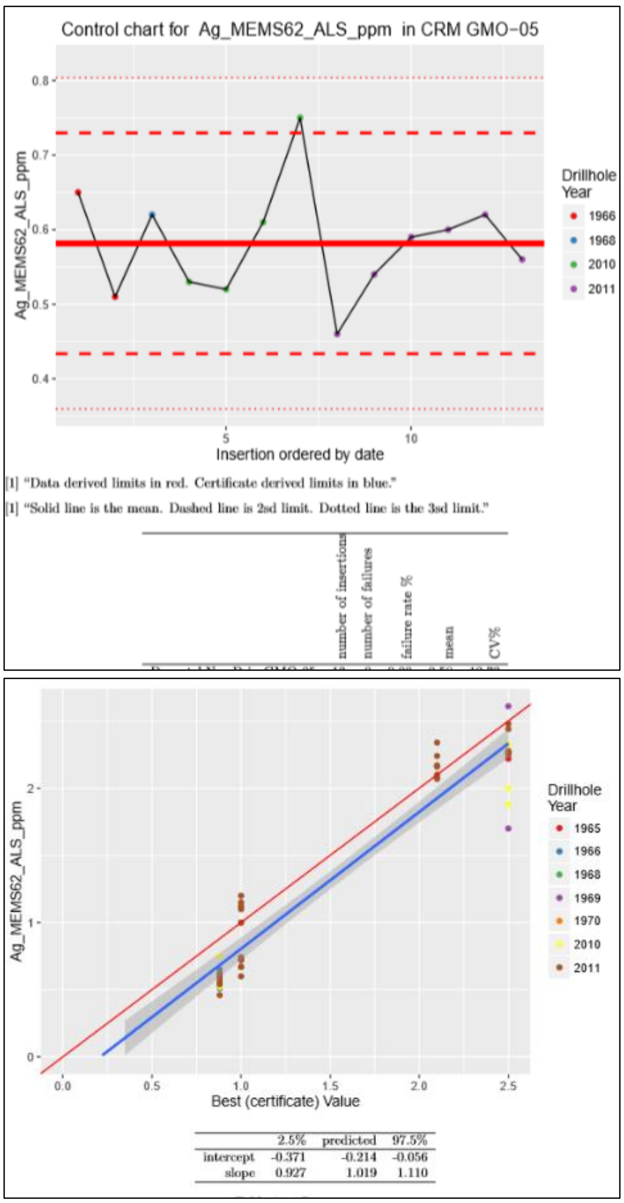
	Number of Pairs	Failing Pairs	Failure Rate (%)	Practical Detection Limit
Ag_1EX_ACME_ppm	224	9	4	0.100



	Number of Pairs	Failing Pairs	Failure Rate (%)	Practical Detection Limit
Ag_1EX_ACME_ppm	423	78	18	0.100

The ALS MEICP61, MEMS61, and MEMS62 data have limited QA/QC, but the data that is extant shows no evidence of poor data. The MEMS62 data (see Figure 11-7) in particular is well supported by a comprehensive QA/QC program. There are nearly 1,000 MEMS61 data points that have no QA/QC support. Given the necessity for data in this Project and familiarity with the quality of data produced by ALS at this time on other projects, these data are to be retained in the database with a low priority. The higher detection limit ME-ICP61a data can be excluded with no consequence as it is always accompanied by a lower level analytical method.

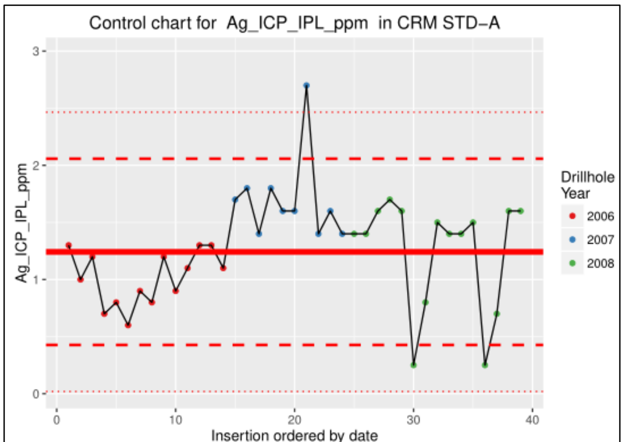
Figure 11-7: Control Charts and Bias Plot for ALS MEMS62 Silver Data



The Ag_FA_LOR_ppm data has a reported detection limit of 0.1 ppm, but is less precise than preferred. The CGS-3 data has a coefficient of variation (CV = standard deviation/mean) of 13 at 1.5 ppm, which strongly suggests that the practical detection limit was higher than 0.1 ppm. The CRM used for this generation of data have no best value for silver, so there is no way to assess the accuracy of these data. The blanks show no evidence of contamination and the small population of duplicates make the assessment of quality tentative, but would technically be classified as unacceptable according to the strict criteria set by AMEC. However, one less failure would make the failure rate acceptable according to the recommendations of Simon (2014). Considering all these factors together, the recommendation is to include these data with a low priority.

The Ag_ICP_IPL_ppm data does not show acceptable precision on any of the CRM (Figure 11-8). This suggests that at the range of concentration values for the standards (0 ppm to 2 ppm), the precision is poor. There is also no ability to assess accuracy as attempts to find certified values for the "STD-X" series has not been fruitful. This poor precision is a consequence of the 0.5 ppm detection limit, remembering that the median for the 1EX data was 0.4 ppm. There is a subset of samples in these data that were also analysed for the Ag 7TD assay package at ACME. While this assay data was itself not ideal for understanding the distribution of silver in deposit with a 2 ppm detection limit, CRMs show it to be unbiased at higher concentrations. Figure 11-9 shows that there is no bias between these datasets so it can therefore be inferred that the Ag_ICP_IPL_ppm data is most likely accurate. The duplicates are acceptable at a detection limit of 0.5 and there is no evidence of contamination. Therefore, the recommendation is to include these data in the final dataset with a low priority.

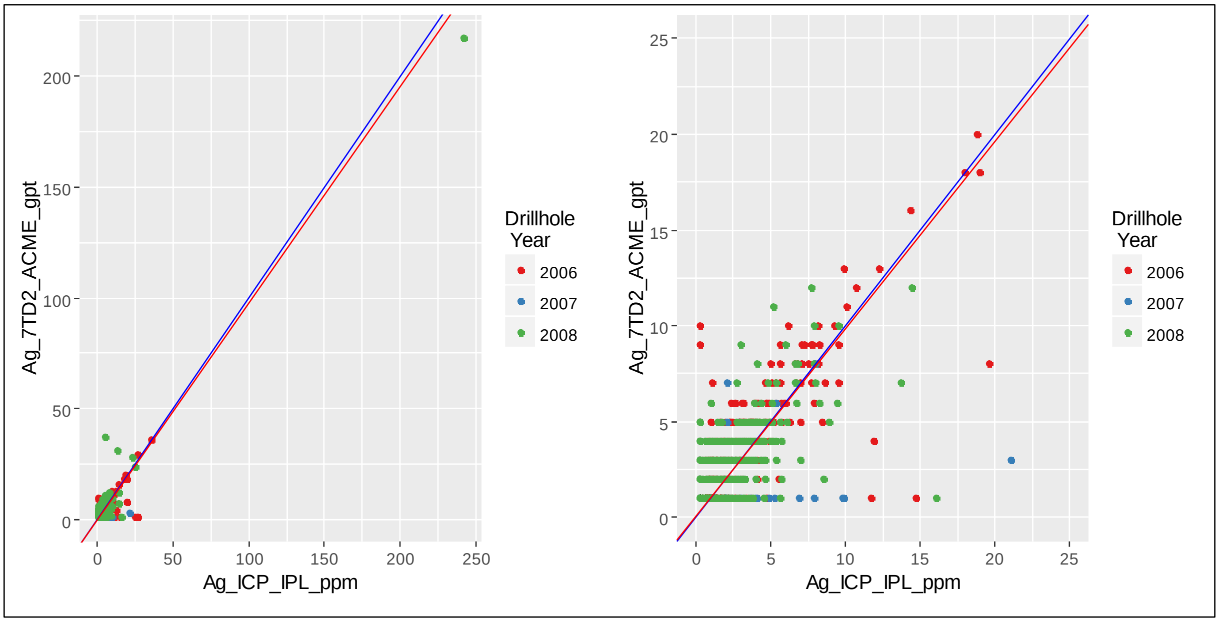
Figure 11-8: Control Chart for Ag_ICP_IPL_ppm for STD-A



[1] "Data derived limits in red. Certificate derived limits in blue."
[1] "Solid line is the mean. Dashed line is 2sd limit. Dotted line is the 3sd limit."

	number of insertions	number of failures	failure rate %	mean	CV%
DespatchNo_D in STD-A	39	1	2.56	1.24	32.82

Figure 11-9: Correlation Between Ag_ICP_IPL_ppm and Ag_7RD_ACME_gpt Data for the Full Range of Points (and Right) Ranged From 0 to 25 ppm



Also of note is the G612 gravimetric fire assay data for six analyses of four samples. These six analyses have such high silver that they appear inconsistent with the remainder of the silver datasets. The sample intervals that include G612 data should be checked in the geological logs to support their inclusion.

11.3.1.2 Arsenic

There are 15 arsenic methods in the database for arsenic, and like silver, there are variable detection limits and intended precisions. The methods and associated statistics are shown in Table 11-2. Like silver, in order to assess the suitability of various methods, a population that is most likely to be representative of the uncensored distribution of the dataset must be reviewed. Like silver, the best method for this will be the As_1EX_ACME_ppm data. There are critics of using mixed acid digestion data for arsenic because of potential volatility issues, but these issues are most pronounced in high total sulphide, low silicate geological matrices (it is a pronounced problem in NiS mines for instance) but is less of an issue for typical porphyry samples.

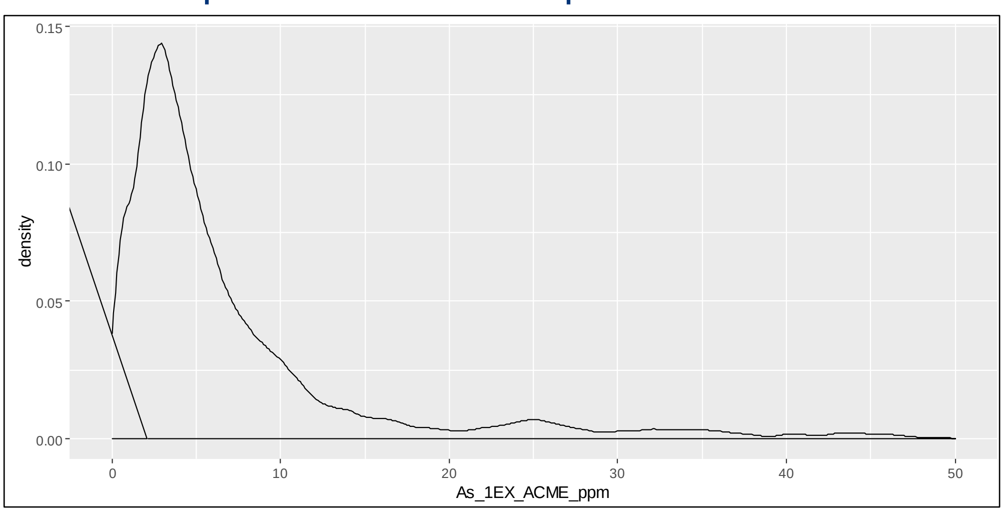
At the time of writing this Technical Report, arsenic was not intended to be included in the resource model, but should be reviewed given the potential penalty costs.

Table 11-2: Summary of the Available Arsenic Data in the Schaft Creek Database

Method	DL	No. Samples	No. > DL	No. > 3DL	No. in BESTEL	Standards + Blanks	Duplicates*
As_1DX1_ACME_ppm	0.5	2446	2323	1905	2114	107	208
As_1EX_ACME_ppm	1	8654	8105	5305	8654	1215	1636
As_1F04_ACME_ppm	0.1	111	111	111	111	0	0
As_1F06_ACME_ppm	0.1	10	10	10	10	0	3
As_7AR2_ACME_pct	100	37	0	0	1	3	3
As_7TD2_ACME_pct	200	21855	175	7	5574	1673	2865
As_AQ200_ppm	0.5	2110	1577	670	2110	204	101
As_AQ250_ppm	0.1	115	109	106	0	5	5
As_AQ252_ppm	0.1	28	25	24	5	0	0
As_ICP_IPL_ppm	5	7717	1058	701	6907	363	561
As_MA200_ppm	1	67	61	35	2	0	0
As_MA370_pct_ppm	200	2121	0	0	0	201	101
As_MEICP61_ALS_ppm	5	14	6	3	4	3	1
As_MEICP61a_ALS_ppm	50	922	81	8	586	67	18
As_MEMS61_ALS_ppm	0.2	926	914	899	877	0	0

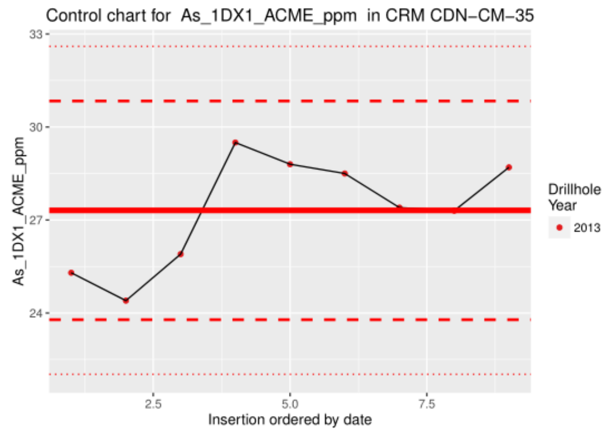
The As_1EX_ACME_ppm data is presumed to be representative of the uncensored population as a whole. There is a median of 4 ppm, a 95th percentile of 36 ppm, and a 99th percentile of 107 ppm. The population density is shown in Figure 11-10. This suggests that the methods with detection limits of 100 ppm or greater are unsuitable for understanding the distribution of arsenic at Schaft Creek. In the event that arsenic is to be included in the resource model, the potential to use the few over-detection limit samples in the high detection limit assay methods should be reviewed in order to constrain high arsenic domains, but is not needed at this stage.

Figure 11-10: Population Density Plot for As_1EX_ACME_ppm, Presumed to be Representative of the Entire Population



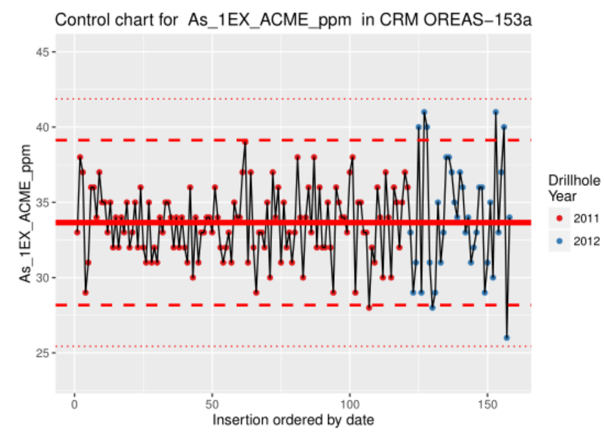
The ACME/BV data, which constitutes a substantial proportion of the samples in the dataset, both show sufficiently precise data to be considered acceptable (Figure 11-11), but both show a negative bias (Figure 11-12) presumably because of incomplete digestion in aqua regia and slight volatility in a mixed acid digestion. Like the silver data, the overall sample duplicates are acceptable, but the pulp duplicates have a higher failure rate than would generally be considered acceptable. Again, the implication of this is that the analytical imprecision makes a disproportionately high contribution to the overall variance (which is acceptable as a whole).

Figure 11-11: Control Charts for the 1DX and 1EX Methods for Arsenic



[1] "Data derived limits in red. Certificate derived limits in blue."
[1] "Solid line is the mean. Dashed line is 2sd limit. Dotted line is the 3sd limit."

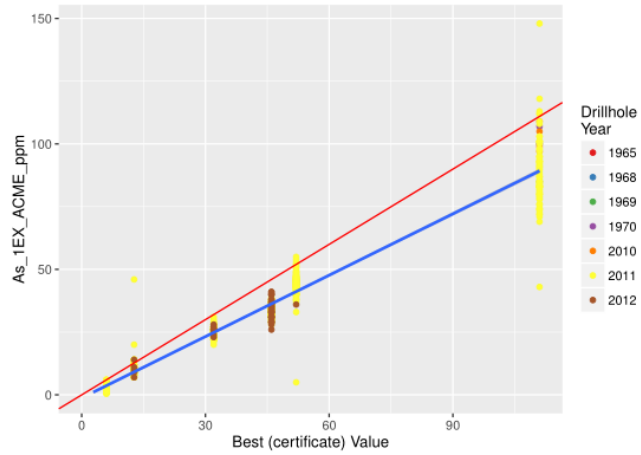
	number of insertions	number of failures	failure rate %	mean	CV%
DespatchNo_D in CDN-CM-35	9	0	0.00	27.31	6.46



[1] "Data derived limits in red. Certificate derived limits in blue."
[1] "Solid line is the mean. Dashed line is 2sd limit. Dotted line is the 3sd limit."

	number of insertions	number of failures	failure rate %	mean	CV%
DespatchNo_D in OREAS-153a	158	0	0.00	33.65	8.14

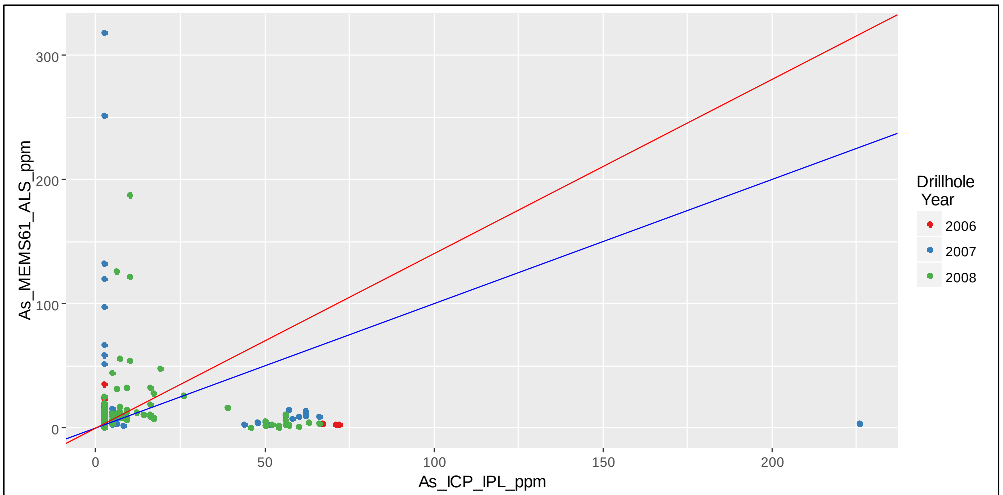
Figure 11-12: Bias Plot for the 1EX Data for Arsenic



	2.5%	predicted	97.5%
intercept	-1.826	-1.229	-0.633
slope	0.803	0.815	0.828

The As_ICP_IPL_ppm data, like the silver data, needs additional evaluation. The 7TD data for arsenic has a 200 ppm detection limit and is therefore unsuitable to make comparisons against, but there is a smaller subset of samples also with ALS MEICP61 data against which a comparison can be made (Figure 11-13). While these samples have insufficient CRMs to assess their accuracy, at the time ALS offered a reasonable data quality, whereas IPL were regarded as more questionable. The bimodal population for the IPL data is unrealistic given the populations indicated by other methods. The interpretation of these data is that there is some fundamental flaw in the arsenic IPL data. Therefore, the recommendation is not to include these data in the BESTEL arsenic column.

Figure 11-13: Scatter Plot of As_ICP_IPL_ppm against As_MEMS61_ALS_ppm



The two ALS methods at a low detection limit for arsenic have essentially no QA/QC support. The recommendation is to keep them in the database with a low priority.

11.3.1.3 Gold

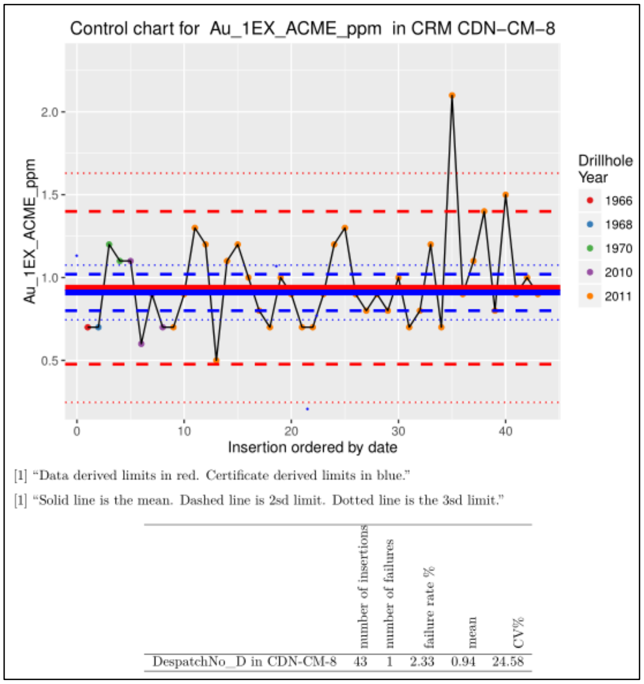
There are 17 different gold methods (Table 11-3) in the Schaft Creek database. However, some of the methods that are included and which contribute to the BESTEL column, are grossly unsuitable for the quantification of gold. In particular, the 1DX1, 1EX, 1F04, AQ200, AQ250, and MA200 are all based on a < 1 g sample, and the AQ252 and 1F06 data are weak aqua regia digestions and not suitable for the quantification of gold either. The MA200 and 1EX data are 4-acid digestions, and it is extremely unusual for a laboratory to report gold by this method because it is strongly unsuitable. Therefore, approximately half the methods can immediately be removed from the QA/QC assessment. As an example of why these data are unsuitable, Figure 11-14 shows the control chart by gold reported by the 1EX, 0.25 g, mixed acid digestion method. The CV% far exceeds 5 and the data ranges far beyond the certificate limits for the CRM.

The removal of all these methods removes only ~320 primary assays from the database, as the vast majority of samples have a high quality fire assay accompanying the multielement analysis that also reported gold. It, therefore, has a minor effect on the fidelity of the overall database for gold.

Table 11-3: Summary of the Available Gold Data in the Schaft Creek Database

Method	DL	No. Samples	No. > DL	No. > 3DL	No. in BESTEL	Standards + Blanks	Duplicates*
Au_1DX1_ACME_ppb	0.0005	2446	2150	1934	84	107	208
Au_1EX_ACME_ppm	0.1	8654	2771	1272	373	1215	1663
Au_1F04_ACME_ppb	0.0002	111	106	102	0	0	0
Au_1F06_ACME_ppb	0.001	10	10	10	1	0	3
Au_AA26_ALS_gpt	0.01	640	521	365	642	65	17
Au_AQ200_ppb	0.0005	2110	1419	880	56	204	101
Au_AQ250_ppb	0.0004	115	113	110	0	5	5
Au_AQ252_ppb	0.0001	28	28	28	5	0	0
Au_FA_LOR_gpt	0.01	1089	1066	983	891	55	27
Au_FA430_ppm	0.005	2110	686	233	2110	205	101
Au_FAAAS_LOR_gpt	0.01	5734	5140	3827	1488	270	201
Au_G601_ACME_gpt	0.005	8425	6074	4792	8408	129	714
Au_G610_ACME_gpt	0.005	13447	10812	9220	12910	1544	2172
Au_G612_ACME_gpt	0.9	4	4	4	2	0	2
Au_ICP21_ALS_gpt	0.001	280	247	214	0	0	1
Au_MA200_ppm	0.1	67	29	15	2	0	0
Au_UNK_UNK_opt	0.003429	8808	8783	8528	8808	0	0

Figure 11-14: Control Chart for Gold Reported by the 1EX ACME Method



However, in all but the G612 ACME method, all fire assay methods provide precise (Figure 11-15) and accurate data (Figure 11-16). While there are occasional sample mix-ups, there are no other failures for gold by fire assay methods.

Figure 11-15: Control Charts for Gold for Two Selected CRMs

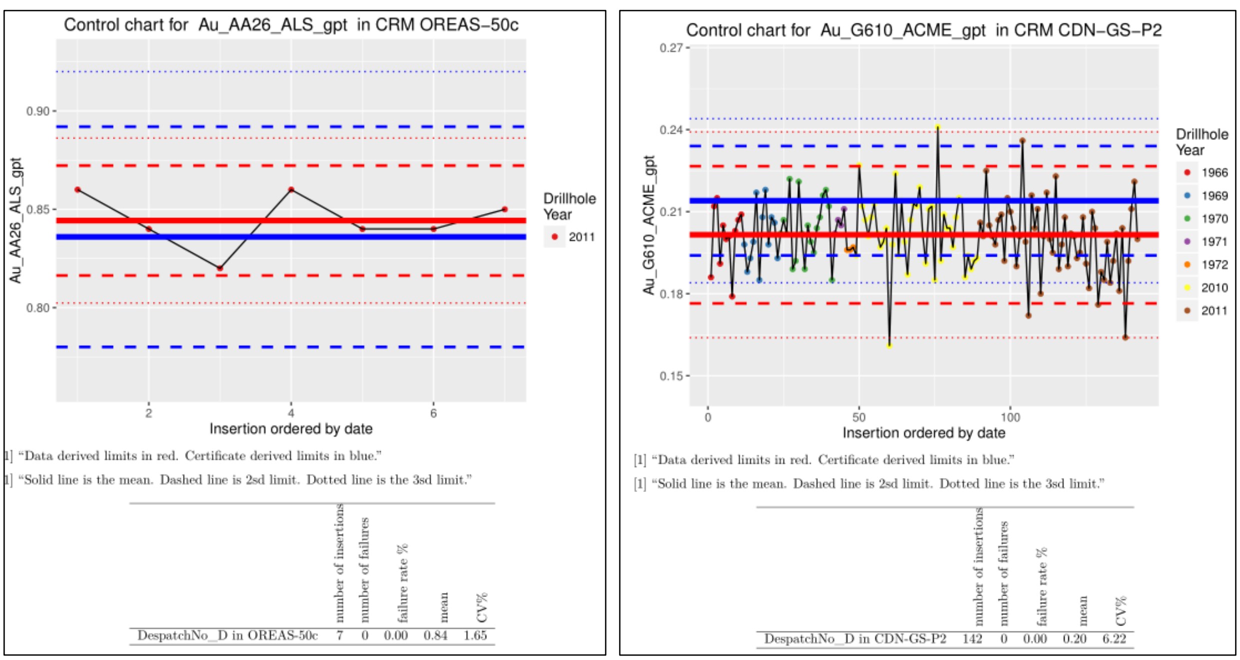
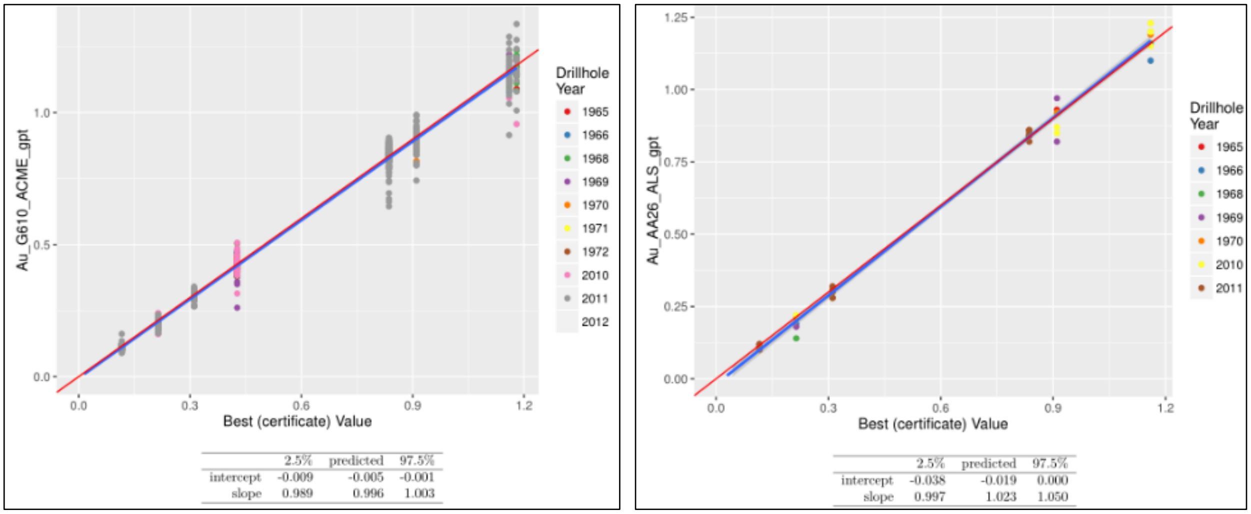


Figure 11-16: Bias Plots for the G610 ACME and AA26 ALS Data Fire Assay Methods



The only exception to the fire assay gold data quality is that Au_FAAAS_LOR_gpt does not have any CRMs with known values, so the accuracy of this method cannot be assessed.

Also of note is the G612 gravimetric fire assay data for four samples. These four samples have such high gold that they appear inconsistent with the remainder of the gold datasets. The sample intervals with a G612 data should be checked in the geological logs for a justification for their inclusion.

11.3.1.4 Copper

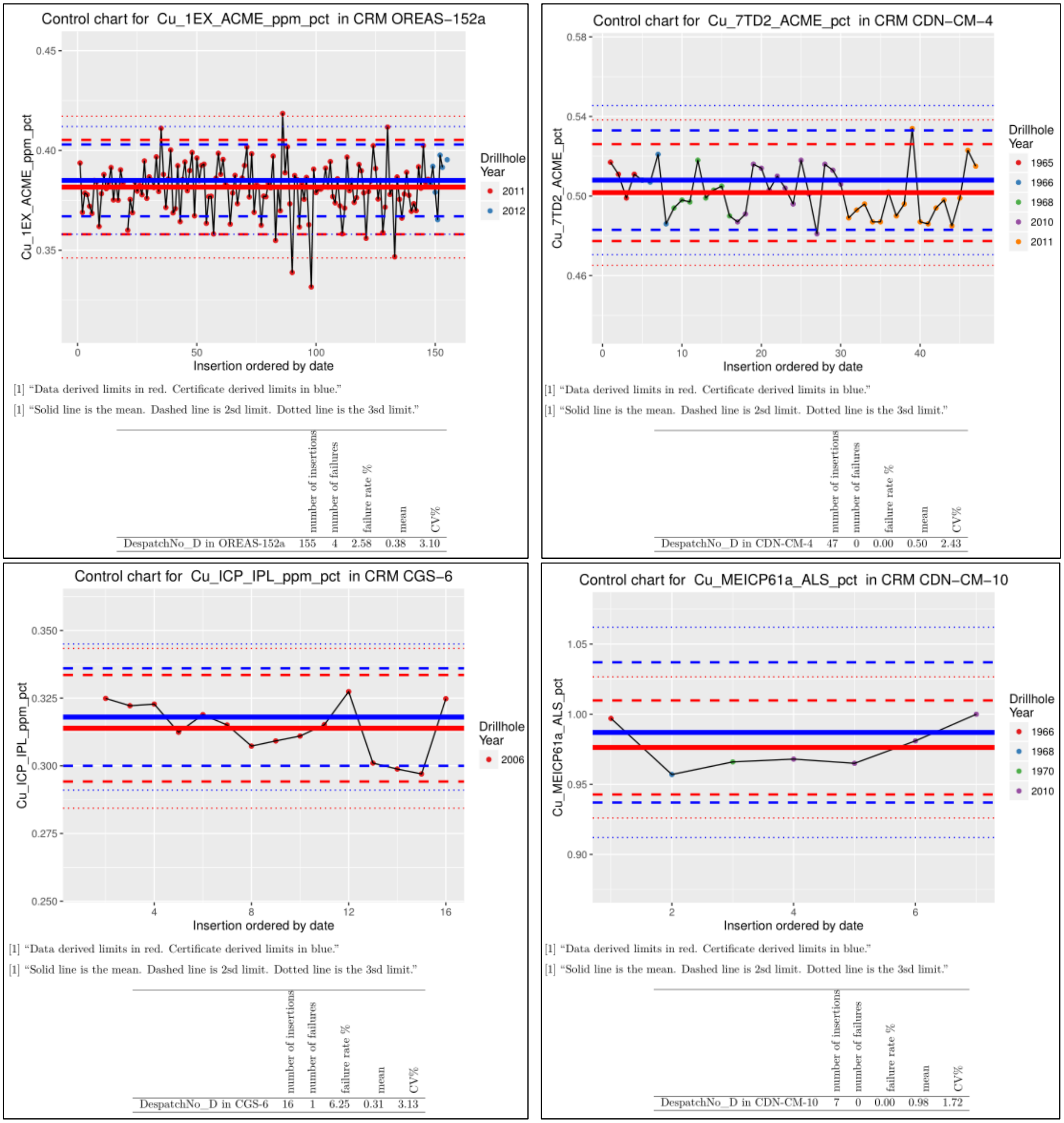
There are 20 copper methods in the database, including assay and geochemical methods, as well as mixed acid and aqua regia methods (Table 11-4). Again, there is variable QA/QC support for these methods.

Table 11-4: Summary of the Available Copper Data in the Schaft Creek Database

Method	DL	No. Samples	No. > DL	No. > 3DL	No. in BESTEL	Standards + Blanks	Duplicates*
Cu_1DX1_ACME_ppm	0.00001	2435	2435	2435	84	107	207
Cu_1EX_ACME_ppm	0.00001	8576	8574	8574	357	1203	1646
Cu_1F04_ACME_ppm	0.00001	111	111	111	0	0	0
Cu_1F06_ACME_ppm	0.00001	10	10	10	1	0	3
Cu_7AR2_ACME_pct	0.02	37	36	33	1	3	3
Cu_7TD2_ACME_pct	0.001	21855	20556	19466	21854	1673	2865
Cu_AQ200_ppm	0.00001	2106	2106	2098	57	204	101
Cu_AQ250_ppm	0.0001	115	115	115	0	5	5
Cu_AQ252_ppm	0.0001	27	27	27	0	0	0
Cu_ASY_IPL_Pct	0.01	1679	1345	1293	393	109	84
Cu_FA_LOR_pct	0.001	1089	1089	1089	846	55	27
Cu_ICP_IPL_ppm	0.0001	7693	7588	7447	1789	363	537
Cu_MA200_ppm	0.00076	63	62	62	2	0	0
Cu_MA370_pct	0.001	2121	1612	1260	2120	201	101
Cu_MEICP61_ALS_pct	0.001	14	14	12	4	3	1
Cu_MEICP61a_ALS_pct	0.001	642	634	608	80	67	17
Cu_MEICP61a_ALS_ppm	0.001	280	251	230	0	0	1
Cu_MEMS61_ALS_ppm	0.0001	926	926	925	275	0	0
Cu_OG62_ALS_pct	0.001	278	266	230	0	0	1
Cu_UNK_UNK_pct	0.001	18249	18224	18026	18282	0	0

There is a singular circumstance for copper in this dataset in that of the 20 methods available, there is not a strong justification to exclude the data from any one method. Some methods have more QA/QC support than others, but there is generally good quality data for copper in the entire database. Apart from the occasional insertion of the incorrect standard, all control charts show good precision in the data (Figure 11-17). These results also show no significant bias in any method (Figure 11-18).

Figure 11-17: Selected Control Charts for a Variety of Copper Methods in the Database



[1] "Data derived limits in red. Certificate derived limits in blue."
[1] "Solid line is the mean. Dashed line is 2sd limit. Dotted line is the 3sd limit."

	number of insertions	number of failures	failure rate %	mean	CV%
DespatchNo_D in OREAS-152a	155	4	2.58	0.38	3.10

[1] "Data derived limits in red. Certificate derived limits in blue."
[1] "Solid line is the mean. Dashed line is 2sd limit. Dotted line is the 3sd limit."

	number of insertions	number of failures	failure rate %	mean	CV%
DespatchNo_D in CDN-CM-4	47	0	0.00	0.50	2.43

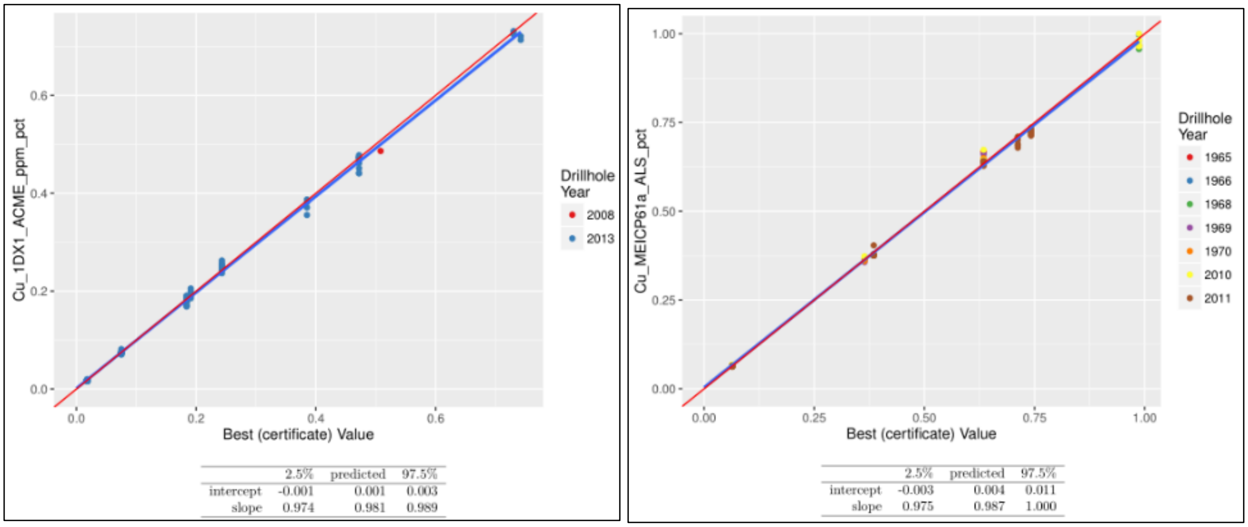
[1] "Data derived limits in red. Certificate derived limits in blue."
[1] "Solid line is the mean. Dashed line is 2sd limit. Dotted line is the 3sd limit."

	number of insertions	number of failures	failure rate %	mean	CV%
DespatchNo_D in CGS-6	16	1	6.25	0.31	3.13

[1] "Data derived limits in red. Certificate derived limits in blue."
[1] "Solid line is the mean. Dashed line is 2sd limit. Dotted line is the 3sd limit."

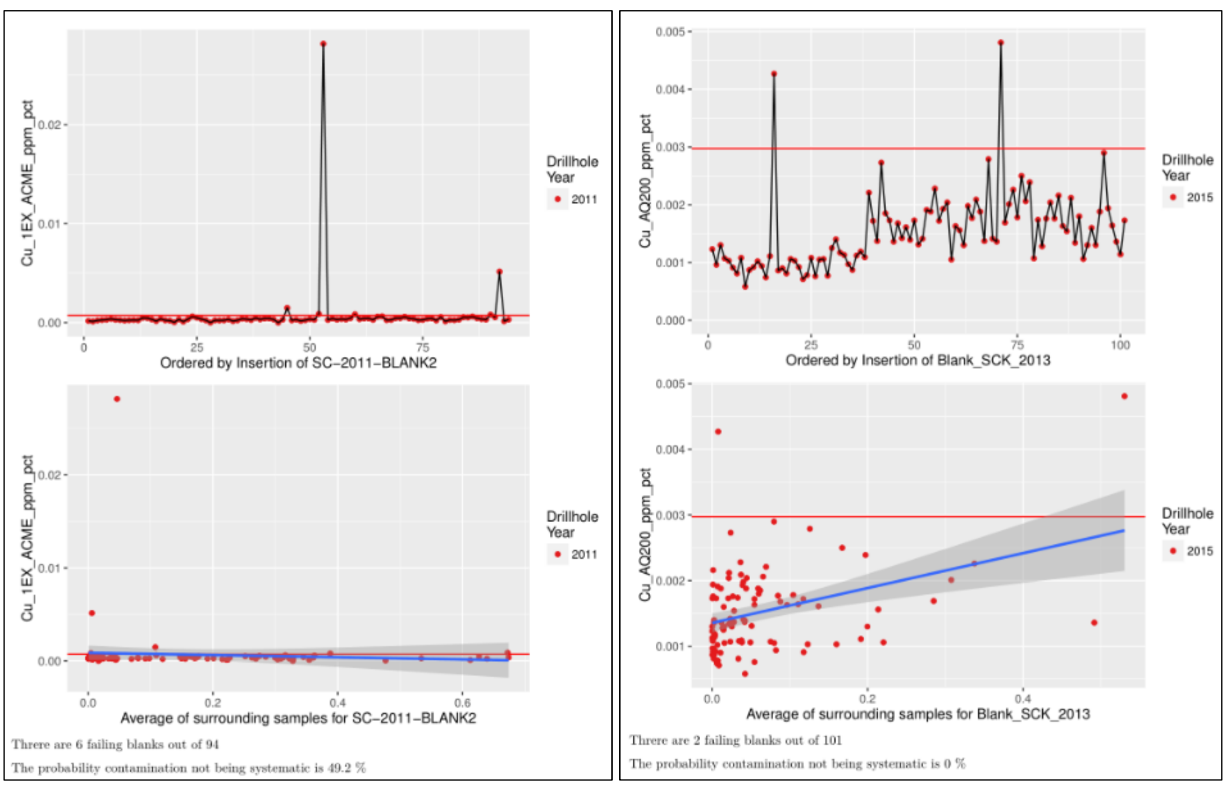
	number of insertions	number of failures	failure rate %	mean	CV%
DespatchNo_D in CDN-CM-10	7	0	0.00	0.98	1.72

Figure 11-18: Selected Bias Plots for Cu



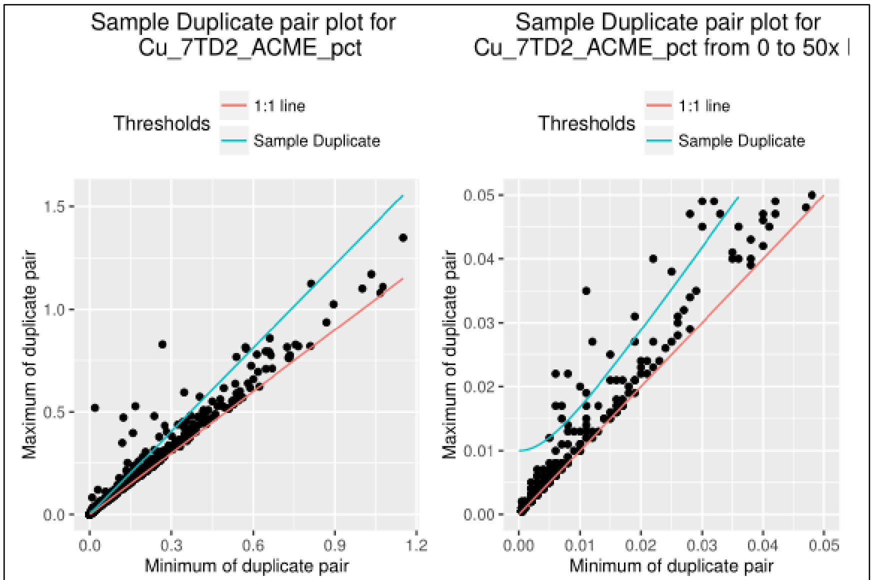
The blanks for copper show no or very minor contamination (Figure 11-19). In the right example plots in Figure 11-19, the null hypothesis that the slope is not different to 0 must be rejected, implying some systematic contamination, but the level of contamination is < 0.25% on average. This is therefore acceptable.

Figure 11-19: Selected Blank Plots for Cu

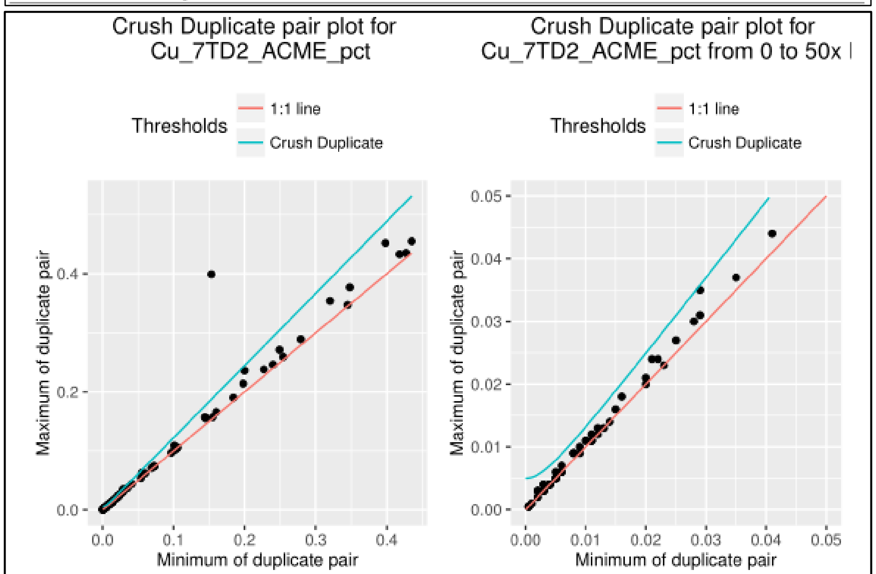


The duplicates show the only problem for the copper data (Figure 11-20). In several of the datasets, the overall sample duplicate failure rate exceeds 10%. However, in all the subsequent stages of sample reduction (crushing, pulverization, etc.) the duplicate failure rates are acceptable. This strongly suggests that the common practice of ¼ core sample duplicates is not suitable for the Project and this practice should be discontinued in the future. The copper data as a whole should nevertheless be considered acceptable.

Figure 11-20: Selected Duplicate Control Charts for Cu



	Number of Pairs	Failing Pairs	Failure Rate (%)	Practical Detection Limit
Cu_7TD2_ACME_pct	493	56	11	0.001



	Number of Pairs	Failing Pairs	Failure Rate (%)	Practical Detection Limit
Cu_7TD2_ACME_pct	86	1	1	0.001

11.3.1.5 Molybdenum

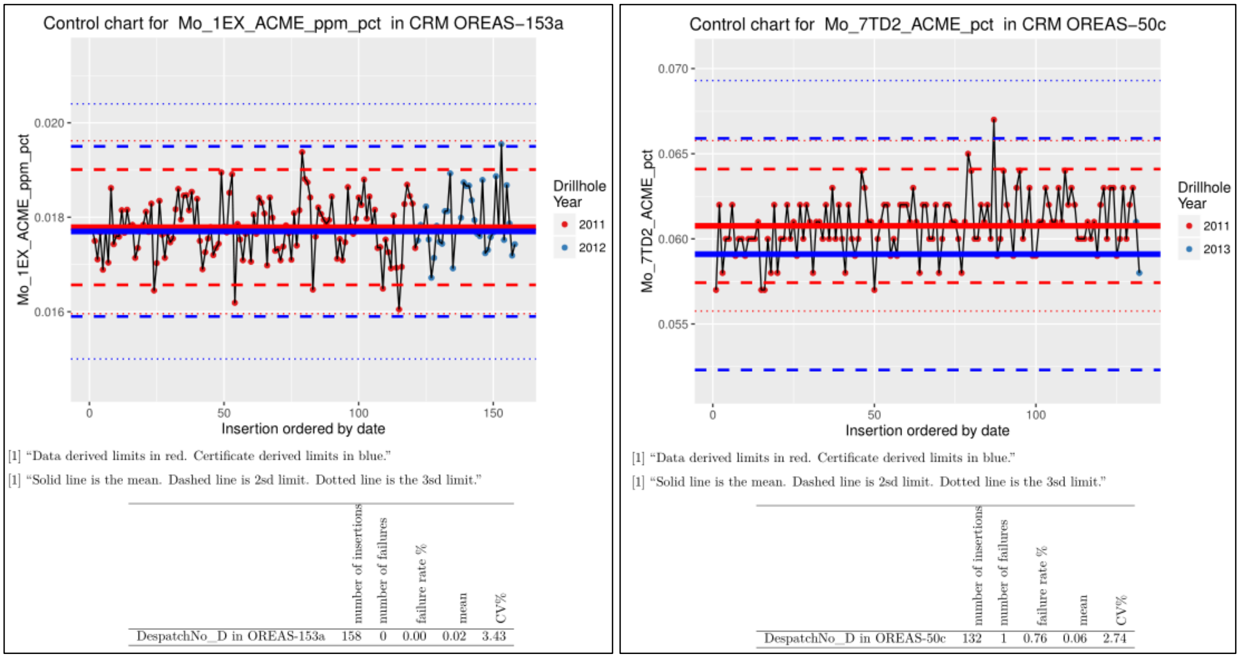
There are 20 molybdenum methods in the dataset, with a range of methods and precisions (Table 11-5). It is notable that there will be a methodological difference in the bias when comparing aqua regia and mixed acid digestion data. The molybdenum will not fully extract into aqua regia but is metastable in solution in an aqua regia, particularly after dilution for analysis. It is anticipated that the aqua regia data will demonstrate a negative bias.

Table 11-5: Summary of the Available Molybdenum Data in the Schaft Creek Database

Method	DL	No. Samples	No. > DL	No. > 3DL	No. in BESTEL	Standards + Blanks	Duplicates*
Mo_1DX1_ACME_ppm	0.00001	2445	2435	2201	84	107	280
Mo_1EX_ACME_ppm	0.00001	8650	8642	8387	8654	1215	1661
Mo_1F04_ACME_ppm	0.000001	111	106	79	0	0	0
Mo_1F06_ACME_ppm	0.000001	10	9	2	1	0	3
Mo_7AR2_ACME_pct	0.001	37	30	28	1	3	3
Mo_7TD2_ACME_pct	0.001	21855	13645	10803	13557	1673	2865
Mo_AQ200_ppm	0.00001	2108	2104	1840	57	204	101
Mo_AQ250_ppm	0.000001	115	109	105	0	5	5
Mo_AQ252_ppm	0.00001	28	27	25	0	0	0
Mo_ASY_IPL_pct	0.0006	1007	962	886	764	0	0
Mo_ICP_IPL_pct	0.0001	2292	2279	2133	214	116	152
Mo_ICP_IPL_ppm	0.0001	5404	5189	5014	1947	247	388
Mo_MA200_ppm	0.00001	67	65	58	2	0	0
Mo_MA370_pct	0.001	2121	371	185	2120	201	101
Mo_MEICP61_ALS_pct	0.0001	14	12	7	4	3	1
Mo_MEICP61a_ALS_pct	0.001	642	454	335	80	67	17
Mo_MEICP61a_ALS_ppm	0.001	280	157	115	0	0	1
Mo_MEMS61_ALS_ppm	0.000001	926	913	821	275	0	0
Mo_OG62_ALS_pct	0.001	278	163	115	0	0	1
Mo_UNK_UNK_pct	0.0001	18124	16909	13156	18135	0	0

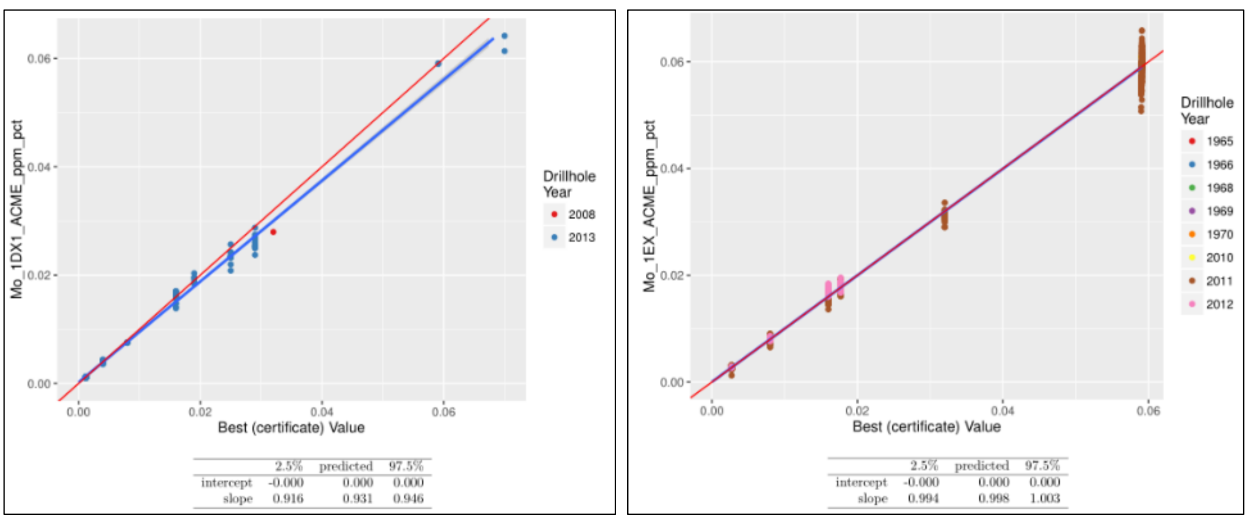
The data for molybdenum is generally good as was the case for copper. While molybdenum exhibits a slightly higher CV% than was seen for copper, this is generally attributable to the concentration of the sample being nearer the detection limit for the method. There are no CRMs that are not performing at a level of precision that would indicate that it was unsuitable for resource estimation (Figure 11-21).

Figure 11-21: Control Charts for Selected Molybdenum Methods



The aqua regia methods generally exhibit a negative bias of between 5% and 10% of the certified values (Figure 11-22). This is expected and is a function of the chemistry of the method. The 4-acid digestion data generally exhibits no or an acceptably small bias.

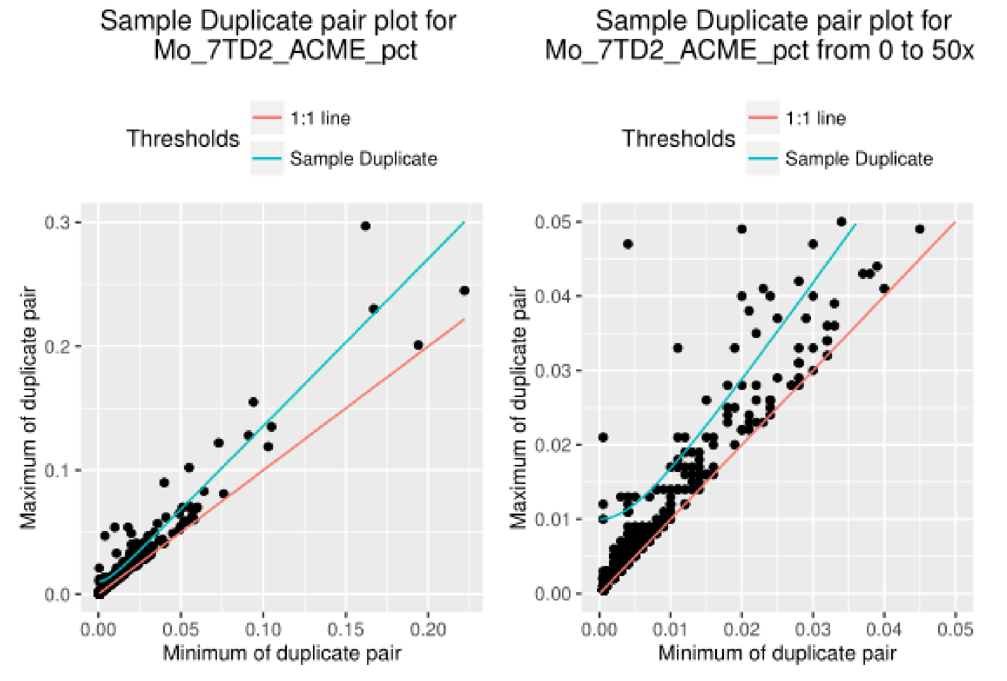
Figure 11-22: Bias Plots for an Aqua Regia Method (Left) and a 4-acid Digestion (Right)



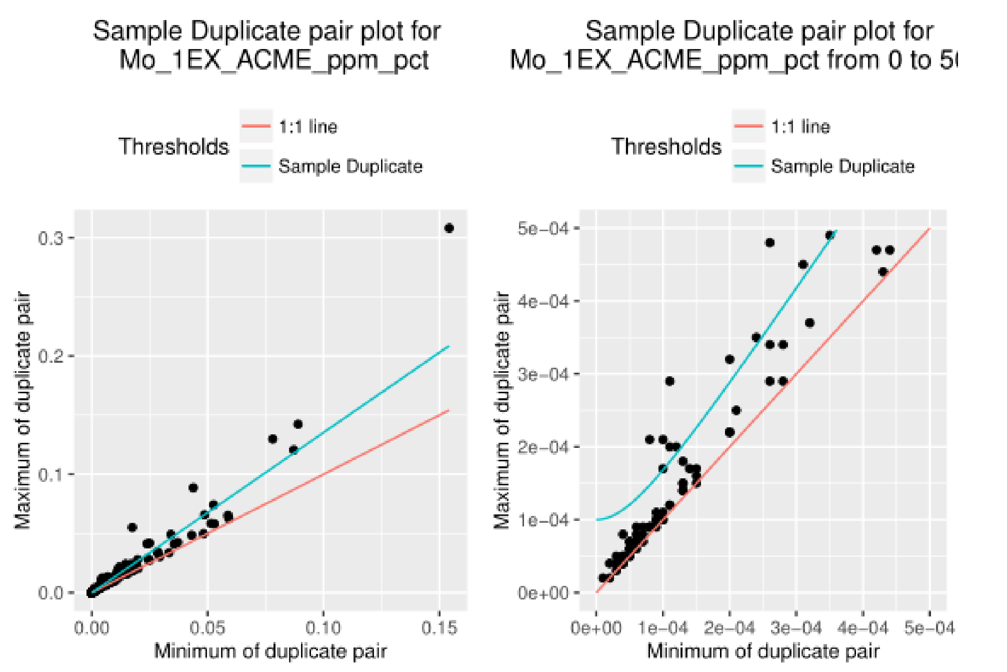
The blanks for molybdenum exhibit an acceptable failure rate and generally no evidence of systematic contamination. Where there is systematic contamination, it is generally a very low level of carryover. This is significant as molybdenite has a tendency to smear on the bowl during pulverization. The duplicate data is better for the assay data than for the geochemical methods (Figure 11-23). The 1EX data for instance has a higher failure rate than the 7TD data. The 1EX data has a failure rate of greater than 30%, which is alarming given that the assay data has an 8% failure rate. That suggests that the error is independent of the sampling. The subsequent stages of the 1EX digestion exhibit the same sort of failure rate, suggesting that the issue is related to subsampling at a late stage as the CRMs indicate that there is good analytical precision. Non-assay data should therefore be deprioritised relative to the assay data.

If the Schaft Creek JV was to remount a significant drill program, the source of this error should be determined, whether it is methodological or related to subsampling for digestion.

Figure 11-23: Selected Duplicate Plots for Selected Mo Methods



	Number of Pairs	Failing Pairs	Failure Rate (%)	Practical Detection Limit
Mo_7TD2_ACME_pct	493	37	8	0.001



	Number of Pairs	Failing Pairs	Failure Rate (%)	Practical Detection Limit
Mo_1EX_ACME_ppm_pct	192	66	34	0.000

11.3.1.6 Rhenium

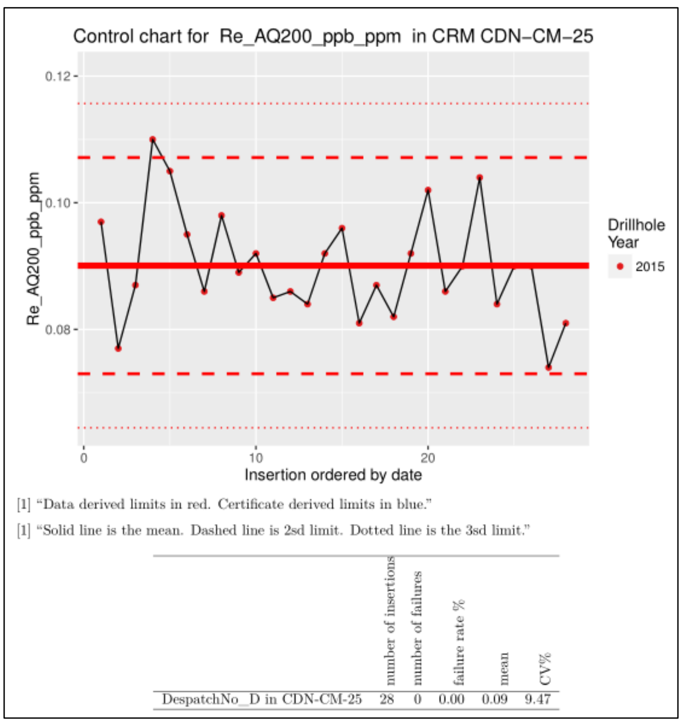
There is only a small subset of samples with a valid rhenium analysis (Table 11-6). There is also only one method with valid QA/QC included, which shows acceptable precision, but the standards for that method have no certification for rhenium (Figure 11-24). Therefore, there is no way of assessing the accuracy of the rhenium data.

At the time of this QA/QC review, there was no intention to include rhenium in the resource estimate, but an assessment of quality was requested.

Table 11-6: Summary of the Available Rhenium Data in the Schaft Creek Database

Method	DL	No. Samples	No. > DL	No. > 3DL	No. in BESTEL	Standards + Blanks	Duplicates*
Re_1F04_ACME_ppb	0.001	111	46	25	111	0	0
Re_1F06_ACME_ppb	0.001	10	5	2	10	0	3
Re_AQ200_ppb	0.01	2105	156	78	2105	201	101
Re_AQ250_ppb	0.001	115	114	104	0	5	5
Re_AQ252_ppb	0.001	28	22	21	28	0	0
Re_MA200_ppm	0.005	67	57	49	65	0	0
Re_MEMS61_ALS_ppm	0.002	926	699	569	906	0	0

Figure 11-24: Control Chart for Rhenium using the AQ200 Method

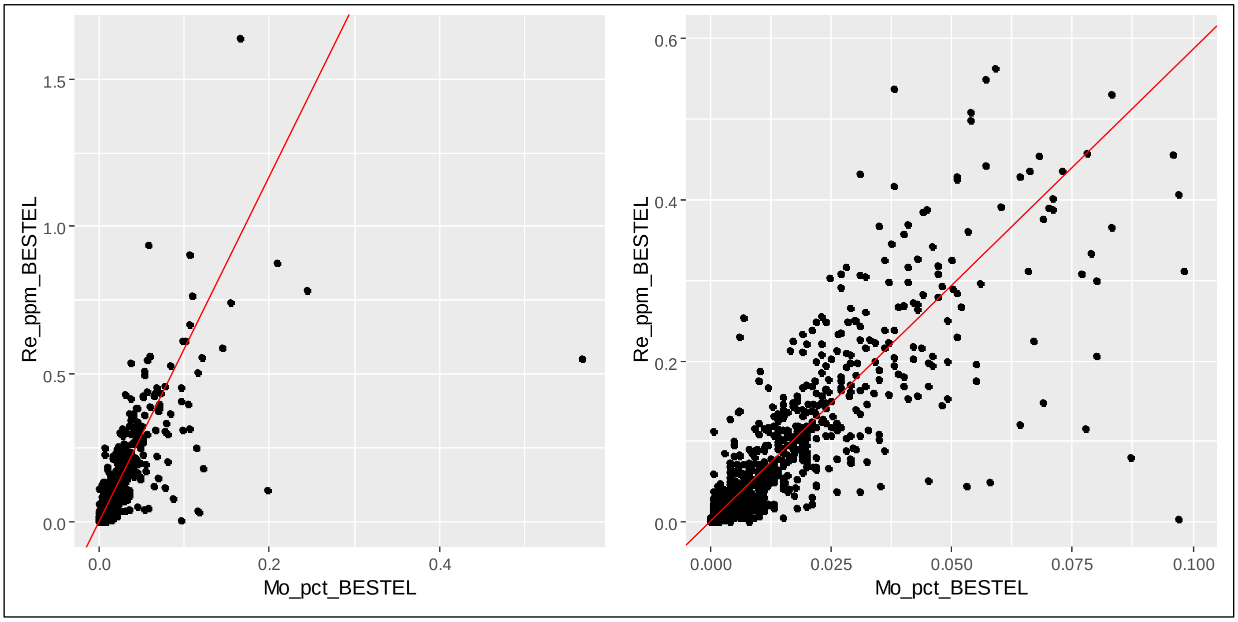


Note: This is the only CRM with more than a few insertions for rhenium in the entire database.

There is not enough blank or duplicate data to conclusively assess rhenium data quality, but the limited data suggests that it is not biased.

It is notable that there is generally a good correlation between molybdenum and rhenium for Mo < 0.2% (Figure 11-25). The statistics for the SMA shown in red are also provided below.

Figure 11-25: Correlation of Rhenium with Molybdenum for the Full Range of Data (Left) and Zoomed into Near the Origin (Right)



Note: SMA in red.

```
Coefficients:
                elevation     slope
estimate    -0.0004765358 5.884923
lower limit -0.0020112563 5.782979
upper limit  0.0010581846 5.988665

H0 : variables uncorrelated
R-squared : 0.7443584
P-value : < 2.22e-16
```

11.3.1.7 Sulphur

There are 14 different sulphur methods (Table 11-7) in the dataset, but at present, only the "TOT_S" infrared combustion data are included in the BESTEL column. Most sulphates and sulphides will be dissolved in an aqua regia digestion or a 4-acid digestion. The latter may exhibit some volatility for sulphur but that is generally negligible in low total sulphur environments and can be evaluated. Some sulphates (barite in particular) have limited solubility, but there is little recorded evidence for their presence at the Property. If the sulphur by these methods can be validated, it provides an additional 30,000 valid analytical results.

At the time of writing this Technical Report, there was no indication that sulphur would be estimated into the block model.

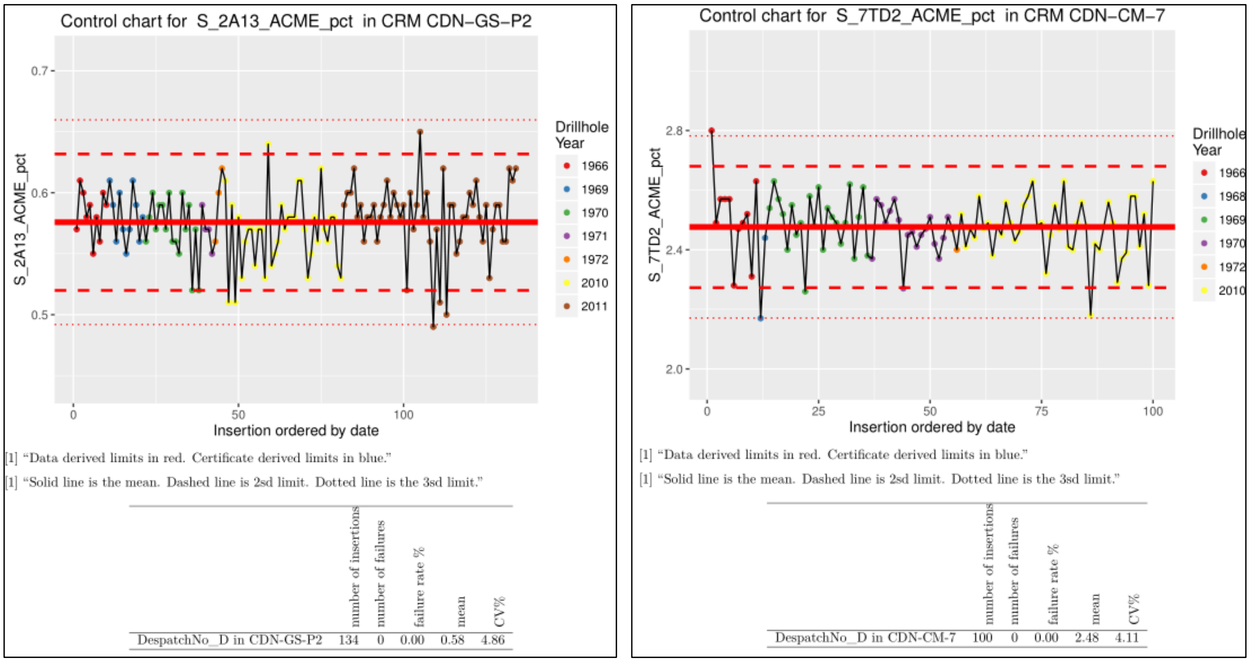
Table 11-7: Summary of the Available Sulphur Data in the Schaft Creek Database

Method	DL	No. Samples	No. > DL	No. > 3DL	No. in BESTEL	Standards + Blanks	Duplicates*
S_1DX1_ACME_pct	0.05	2209	1781	1392	0	105	194
S_1EX_ACME_pct	0.1	8654	6364	4152	0	1215	1663
S_1F06_ACME_pct	0.1	10	4	1	0	0	3
S_7AR2_ACME_pct	0.05	37	34	33	0	0	0
S_7TD2_ACME_pct	0.05	21855	18231	14893	0	1674	2865
S_AQ200_pct	0.05	2105	483	229	0	201	101
S_MA200_pct	0.1	67	62	47	0	0	0
S_MA370_pct	0.05	2121	506	229	0	201	101
S_MEICP61_ALS_pct	0.04	14	13	12	0	3	1
S_MEICP61a_ALS_pct	0.05	922	860	621	0	67	18
S_MEMS61_ALS_ppm_pct	0.01	926	899	769	0	0	0
TotS_2A13_ACME_pct	0.02	15153	13697	12532	14913	1586	1705
TotS_IR08_ALS_pct	0.01	926	898	766	650	0	0
S_TC000_pct	0.02	2260	1095	575	0	209	106

The sulphur data shows excellent precision by all methods. The vast majority of CRMs show excellent precision (Figure 11-26) when above the detection limit, the exception being 7TD2_ACME_pct in CRM CDN−GS−P2, which has an atypically high CV%. The bias plots (Figure 11-27) also show no appreciable bias by any method (note that the 0.6% S point on the 7TD graph shown is a recommended value, not a certified value) and there is no evidence of contamination. Every duplicate population has < 10% failure rate at all levels of duplication.

There is no reason to exclude the additional 30,000 samples in the calculation of a _BESTEL column.

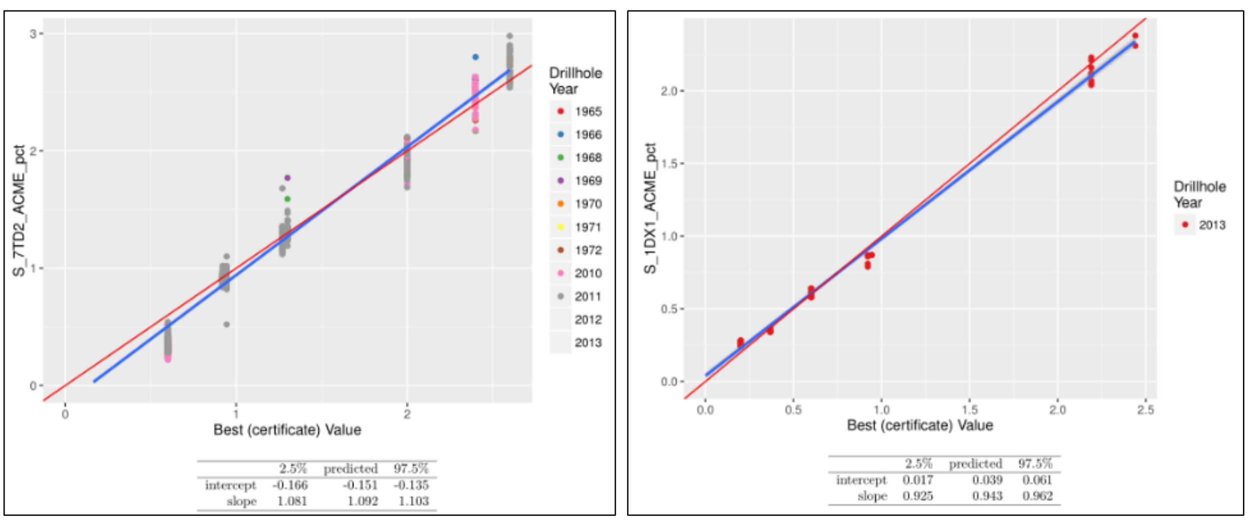
Figure 11-26: Control Charts for (Left) an Infrared Combustion Method and (Right) a Mixed Acid Digestion



	number of insertions	number of failures	failure rate %	mean	CV%
DespatchNo_D in CDN-GS-P2	134	0	0.00	0.58	4.86

	number of insertions	number of failures	failure rate %	mean	CV%
DespatchNo_D in CDN-CM-7	100	0	0.00	2.48	4.11

Figure 11-27: Bias Plots for S by (Left) a Total Digestion Method and (Right) an Aqua Regia Method



	2.5%	predicted	97.5%
intercept	-0.166	-0.151	-0.135
slope	1.081	1.092	1.103

	2.5%	predicted	97.5%
intercept	0.017	0.039	0.061
slope	0.925	0.943	0.962

11.3.2 Historical Data

The Schaft Creek JV assessed the historical data by comparing results from twinned drill holes (drilled by Copper Fox) and nearest neighbour (NN) samples and from a small subset of reassays taken by Copper Fox on different sampling intervals compared to the original sampling. AMEC provided a set of 3 m composite samples from historical and modern drill holes and provided nearest sample comparisons. The distance between points ranged from < 1 m to > 100 m. For the purposes of this review, for a sample to be considered a twin, the maximum difference that could be considered was assumed to be 20 m but preferably a much closer sample spacing would be required for a twinned sample comparison. For each element and generation, plots and summary statistics were produced for a 5 m maximum distance, as well as a 10 m and 20 m maximum distance.

For each comparison, the Schaft Creek JV prepared three different plots. The top plot shows the number of sample pairs that are within a certain distance. The middle plot shows a scatterplot of historical data versus nearest modern data with the points coloured by distance. There is a 1:1 line shown in blue and the red line shows the SMA regression forced through the origin for the data limited to the maximum sample spacing. The thinner red lines show the 95% confidence interval in the slope of the SMA regression. The pink line shows the SMA when not forced through 0 and is more in line with the analysis performed by AMEC. The bottom plot shows the slope of the SMA correlation (forced through the origin) in red and the 95% confidence interval in the blue and green lines. If this range contains 1.0, at any sample spacing then at that distance, there is insufficient evidence the data is biased and that a correction needs to be made. Likewise, the purple line shows the probability that the data is no different from 1. When this line drops below 0.05, then the data is biased and there is good evidence that a correction needs to be made.

In addition to this, for each comparison, the SMA coefficients and uncertainties are printed firstly for the SMA forced through the origin, and subsequently for the SMA with the intercept also calculated. Both of these have the probability of a correlation shown and the probability that the slope =1 shown. In the event that AMEC recommended a correction to a historical dataset previously, then the probability that the same slope would be predicted was also evaluated.

In some cases, there are resampling and modern analyses on different sample intervals for a subset of historical holes. Quantile-quantile (QQ) plots are shown for these sampling intervals to allow for comparisons of the populations of data and an estimate of bias.

11.3.2.1 Asarco Data

The Asarco generation drilling has no primary data for silver or gold; however, the copper and molybdenum data were reviewed. AMEC previously recommended the correction below be applied to the molybdenum data and that no correction be made to the copper data.

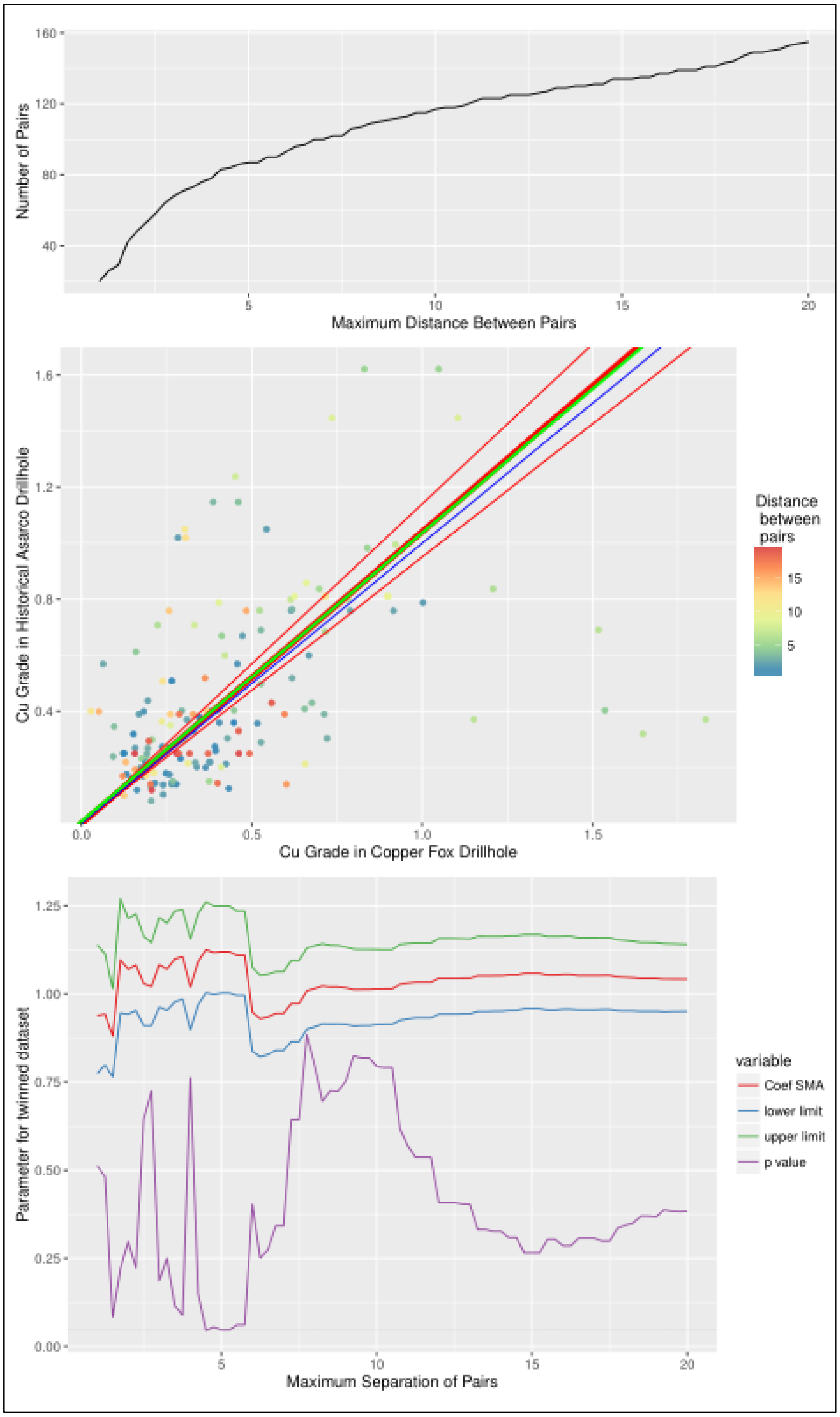
Asarco

- Mo – correction y = (x - 0.0039) / 0.8605
  The correction is applied up to a maximum grade of 0.03% Mo. At higher grades, no correction is applied, as the corrected grades would be higher than the original grades.

Copper in the twinned samples shows that at nearly all spacings between 0 m and 20 m, there is insufficient evidence for the need of a correction (Figure 11-28). For a very small set of maximum sample distances between 4 m and 6 m, there is > 95% chance that the slope is not 1. At 4 m maximum sample spacing, there are more than 80 samples, and the SMA regression through this population is not statistically different to 1. At 6 m maximum sample spacing, there are 90 samples, and the SMA regression through this population is again not statistically different to 1; however, at 5 m, there is a suggestion of bias between the datasets. There is no industry standard on what constitutes a twinned sample but consensus seems to be between 5 m and 10 m as a maximum distance. Given that at most sample spacings < 10 m, there is a < 95% probability that the slope is not 1; there is no basis for a copper correction based on these data. This is in agreement with AMEC recommendations from the previous resource estimate.

However, the QQ plots for copper of the resampling completed by Copper Fox show that the populations match at low concentrations (up to 0.4% Cu), but that at higher concentration, there is a clear departure from the near-linear trend (Figure 11-31). This would affect some 1,500 assays across the entire dataset, with the bias becoming more pronounced at higher concentrations. This has been debated amongst the project team as it indicates a strong bias at high concentrations. However, there was at least 30 years between the original drilling and the resampling, during which time, the samples had been exposed to the atmosphere. It is possible that intense oxidation of the core may have resulted in a mass increase becoming more pronounced in higher sulphide material and in a higher bias in higher grade samples. Because the bias is not indicated in the twinned dataset, it has been concluded that there is no justification for a correction based on these data.

Figure 11-28: Summary Plots for Copper in the Asarco Data as Compared to the Copper Fox Data for Sample Pairs up to 20 m Apart



The molybdenum data is far more complicated (Figure 11-29 and Figure 11-30). At 5 m, there is a slope averaging around 0.8 but statistically different from 1, implying that a correction is warranted. However, at distances of 7.5 m and greater, there is no statistically different slope of the SMA regression from 1 and no correction is warranted. This raises questions about what the correct course of action should be. The AMEC correction slope is within error of the SMA coefficients at a 5 m sample spacing. The SMA coefficients are shown below for an unconstrained SMA (top) and an SMA forced through the origin (below).

```
Coefficients:
              elevation     slope
estimate    0.005936893 0.6337891
lower limit 0.003494934 0.5338013
upper limit 0.008378852 0.7525059

Coefficients (no intercept included):
            elevation     slope
estimate            0 0.8504140
lower limit        NA 0.7491039
upper limit        NA 0.9654255
```

The recommendation assumes that 90 pairs at 5 m is probably representative and that a correction for the slope should be used, but not the intercept in this case, as there is an issue for data around detection limit in the historical data. This makes the data around the intercept unreliable, and therefore, an intercept correction is not supported by the quantized data. The appropriate correction would therefore be to use the SMA forced through the origin:

Mo_Asarco = Mo_UNK_UNK_pct/0.8504.

However, this would result in an increased molybdenum grade, which AMEC was consistently reluctant to recommend historically. The QQ plots for the molybdenum data show the same bias as the twinned data and confirm the recommended correction, which is shown as a green line.

Figure 11-29: Summary Plots for Mo in the Asarco Data as Compared to the Copper Fox Data for Sample Pairs up to 5 m Apart

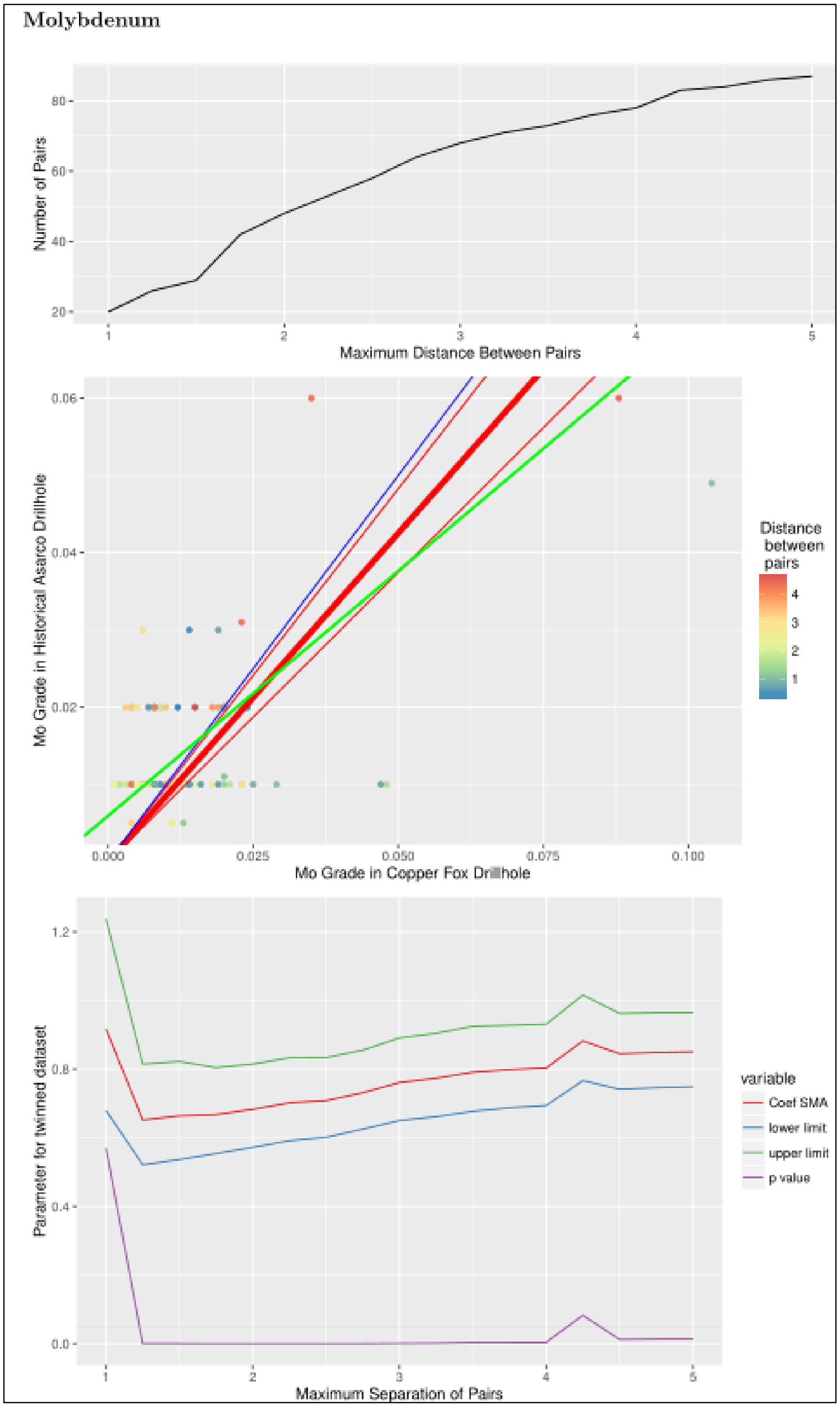


Figure 11-30: Summary Plots for Mo in the Asarco Data as Compared to the Copper Fox Data for Sample Pairs up to 20 m Apart

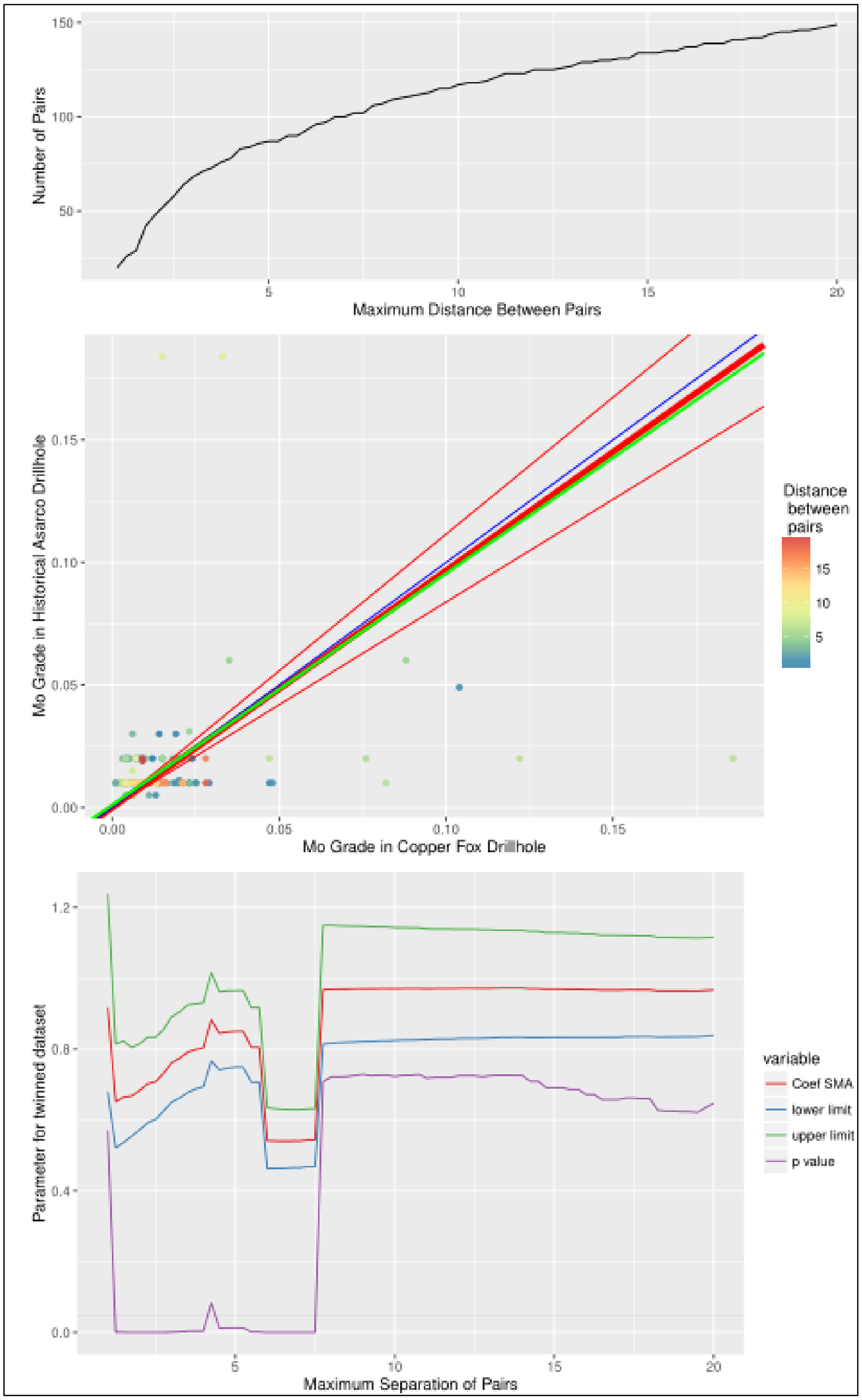
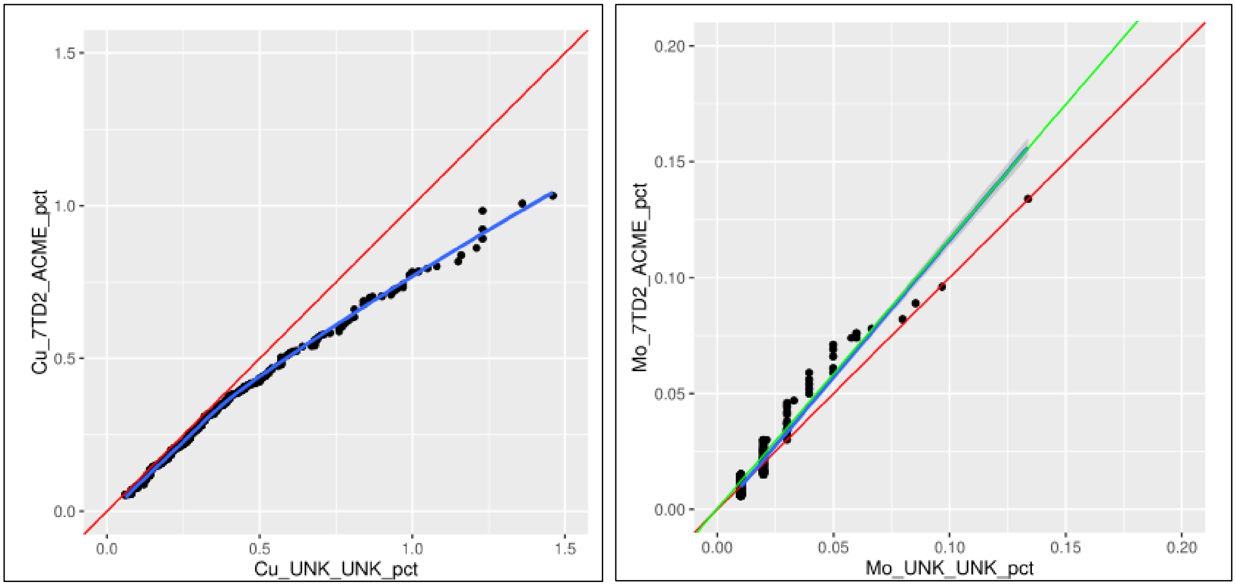


Figure 11-31: QQ plots for Resampling of Asarco Generation Drilling



Note: The recommended correction is shown as a green line.

11.3.2.2 Hecla Data

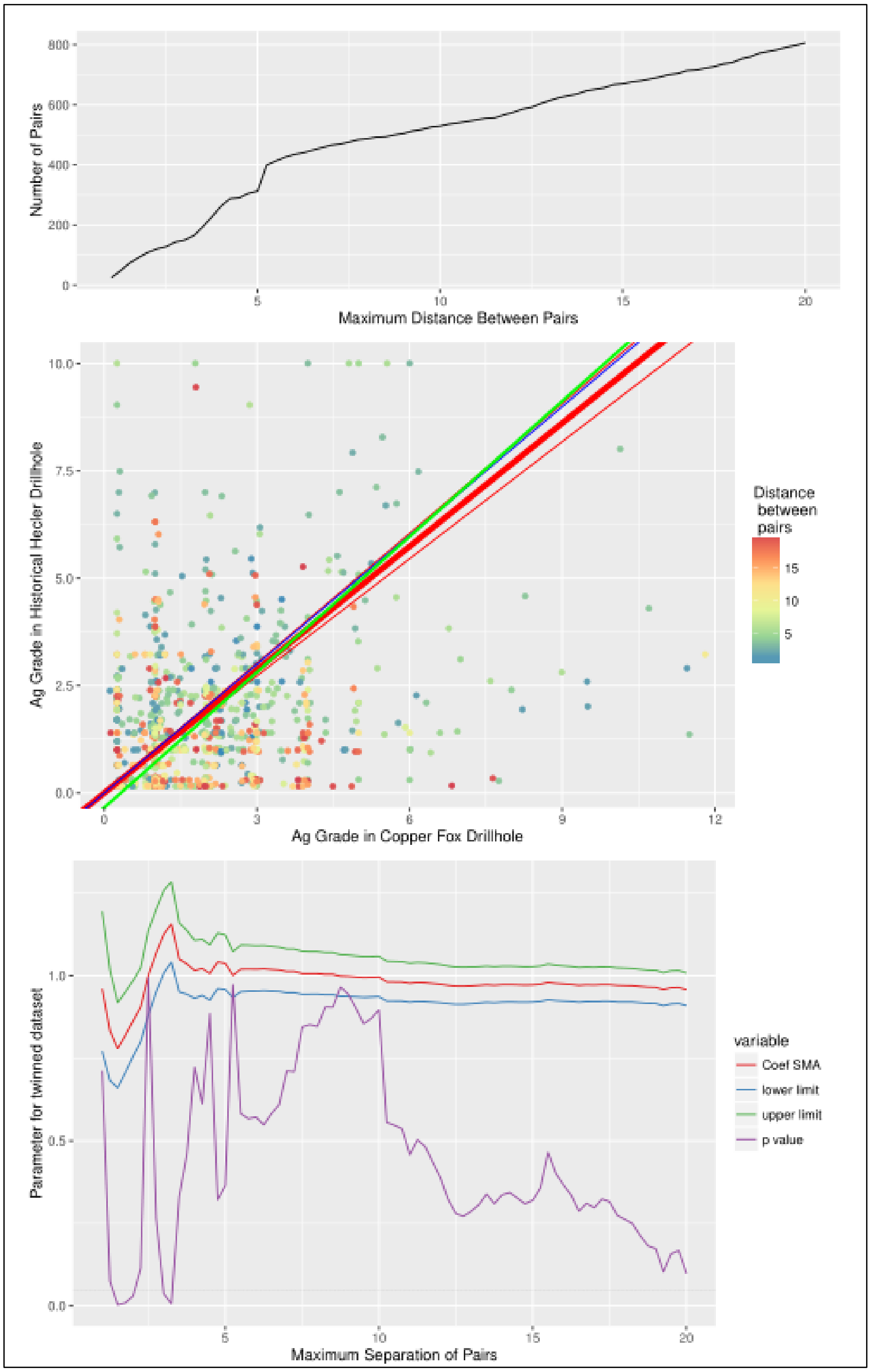
AMEC compared silver, gold, copper, and molybdenum data from the Hecla drilling program in support of a previous resource estimate. There are more than 200 sample pairs within 5 m of each other and over 600 within 20 m. For the Hecla data, there are reassays using the 7TD2 assay method at ACME on recut intervals that were different to the primary assays. These reassays were not taken on barren material and only exist for six holes, but they can be used to compare populations of data for these elements, but for copper and molybdenum only. Where these data are not available, only the NN comparisons are relied upon. Previously, AMEC made the following recommendations for corrections to the raw data:

- Cu – correction y = (x+0.0114) / 1.0659
- Au –correction y = (x + 0.0022) / 1.1271
- Ag – correction y = (x – 0.2685) / 0.9418
  The correction is applied up to a maximum of 4.6 g/t Ag. At higher grades, no correction is applied as the corrected grades would be higher than the original grades. Grades below 0.27 g/t are re-set to zero.

The Schaft Creek JV's analysis of the twinned samples is that the silver comparison data does not support a correction, which is not in line with previous recommendations. The summary statistics for the SMA are shown below for sample comparisons at a maximum of 5 m. The origin lies within error of the elevation parameters, and there is a > 90% chance that the SMA slope is not different to 1. This observation that there is no apparent bias is mimicked at all potential sample spacings up 20 m (Figure 11-32).

```
Coefficients:
             elevation     slope
estimate     0.1776835 0.9935611
lower limit -0.2023746 0.8915230
upper limit  0.5577417 1.1072778

H0 : variables uncorrelated
R-squared : 0.0530017
P-value : 3.9208e-05

------------------------------------------------------------
H0 : slope not different from 1
Test statistic : r= -0.006638 with 311 degrees of freedom under H0
P-value : 0.90689
```

Figure 11-32: Paired Sample Comparison Plots for Silver in the Hecla Data for all Sample Spacings up to 20 m



The gold comparison (Figure 11-33) data shows that sample pairs between 0 m and 5 m have a 3% probability of not being different to 1, so it would be typical to justify that a correction is warranted. The previous AMEC recommendation was for a correction of y = (x + 0.0022) / 1.1271. This can also be evaluated (below). There is a 64% probability that the slope is not different to the 1.1271 recommended by AMEC; however, there is a complication with the intercept. The lower and upper estimates for the intercept straddle 0 (i.e., there is not good evidence that the slope does not go through the origin).

```
Coefficients:
              elevation    slope
estimate    -0.01668429 1.103483
lower limit -0.07012727 1.007173
upper limit  0.03675869 1.209002

H0 : variables uncorrelated


R-squared : 0.3281777
P-value : < 2.22e-16

------------------------------------------------------------
H0 : slope not different from 1.1271
Test statistic : r= -0.02583 with 311 degrees of freedom under H0
P-value : 0.64895
```

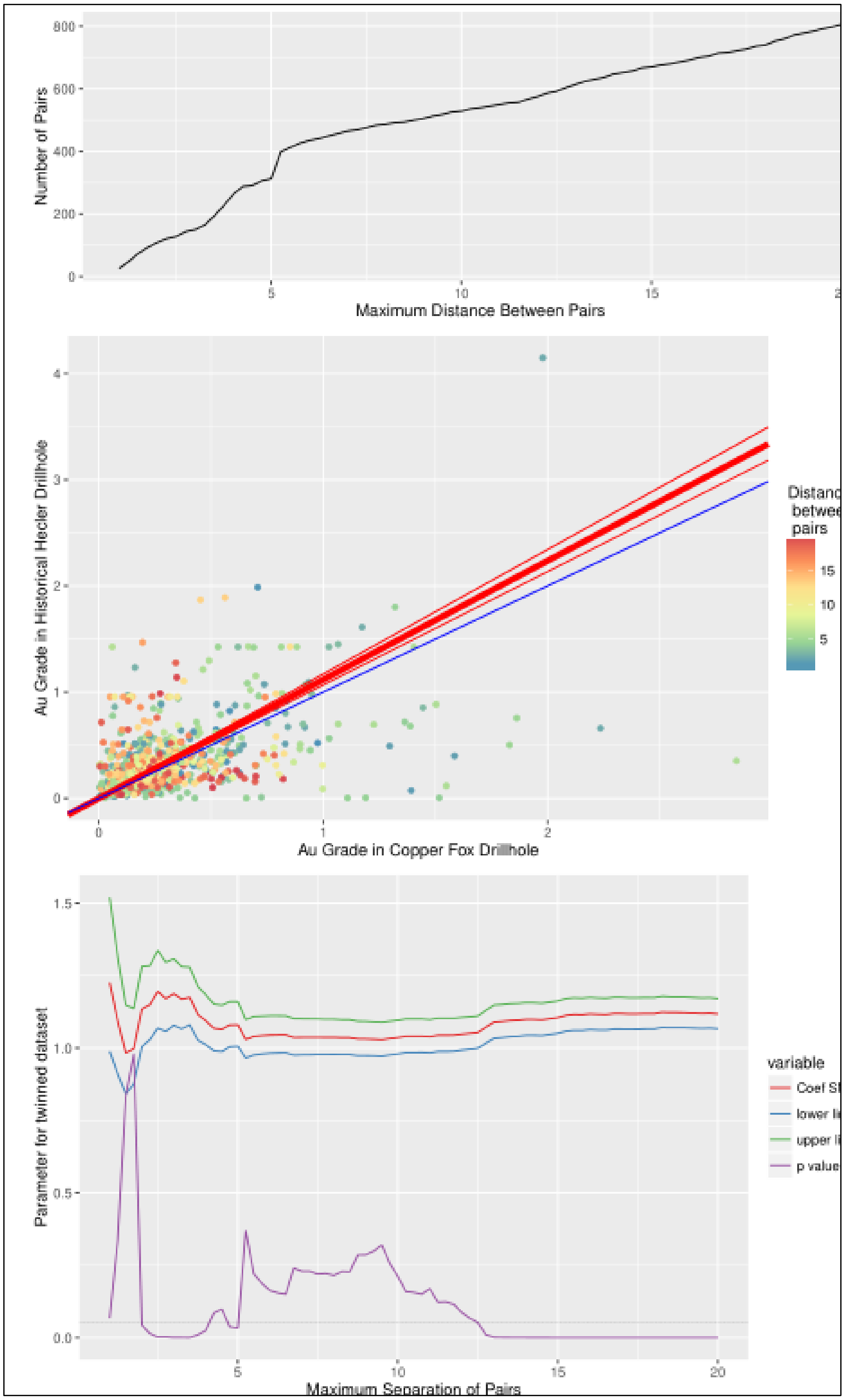
Because of this, the Schaft Creek JV recommends that the slope forced through the origin is used for the correction. This still has a < 5% chance of accepting the null hypothesis that the slope is not different to 1 and is a simpler correction of:

Au_Hecla = Au_UNK_UNK_pct/1.0806.

```
Coefficients (no intercept included):
            elevation    slope
estimate            0 1.080614
lower limit        NA 1.006467
upper limit        NA 1.160224

H0 : variables uncorrelated
R-squared : 0.592102
P-value : < 2.22e-16

------------------------------------------------------------
H0 : slope not different from 1
Test statistic : r= 0.1206 with 311 degrees of freedom under H0
P-value : 0.032892
```

There is very little difference between this correction and the one recommended by AMEC in the previous resource estimate.

Figure 11-33: Paired Sample Comparison Plots for Gold in the Hecla Data for all Sample Spacings up to 20 m



AMEC recommended a sample correction of around 6% to the data for copper. The QQ plot for the six holes with resampling shows no bias between assays and reassays (Figure 11-34).

When initially reviewing the NN data, it seems difficult to support the correction. If samples are limited to being within 5 m of each other. The slope of the SMA forced through the origin is predicted to be 1.33 and has a negligible probability not being different to either 1 or 1.06. This discrepancy is at least in part explained by the presence of regular high copper samples, which were removed from the dataset by AMEC as outliers. These outliers are clear in (Figure 11-35).

```
Coefficients (no intercept included):
            elevation    slope
estimate            0 1.325695
lower limit        NA 1.247355
upper limit        NA 1.408956

H0 : variables uncorrelated
R-squared : 0.6425972
P-value : < 2.22e-16

------------------------------------------------------------
H0 : slope not different from 1
Test statistic : r= 0.4312 with 371 degrees of freedom under H0
P-value : < 2.22e-16
```

Once the outliers are removed from the dataset, the SMA slope is much more reasonable and the results are almost identical to what AMEC recommended (Figure 11-36). The limitation, however, is that there is a 22% chance that the slope is not different to 1, which does not meet the standard 95% criteria for assessing certainty. In this case, it may not seem significant, but there seems to be insufficient evidence to support the correction and the data should be accepted as is.

```
Coefficients:
               elevation     slope
estimate    -0.008953563 1.0616166
lower limit -0.060094138 0.9636584
upper limit  0.042187012 1.1695325

H0 : variables uncorrelated
R-squared : 0.1223891
P-value : 6.8681e-12

------------------------------------------------------------
H0 : slope not different from 1
Test statistic : r= 0.06373 with 361 degrees of freedom under H0
P-value : 0.22576
```

On the other side of this argument is that it equally cannot be argued that the correction factor applied by AMEC is incorrect. Thus, if it was desired to maintain consistency, there is an argument for that also. This correction reduces the grade of the historical samples, so it does not run the risk of overestimating grade, and thus, does not have dire consequences.

However, the QQ plots of the resampled material show no bias between the primary assays and the resampled data (Figure 11-34). Therefore, the recommendation is to reject a correction of these data.

Figure 11-34: QQ Plots for Copper in the Resampled Hecla Holes

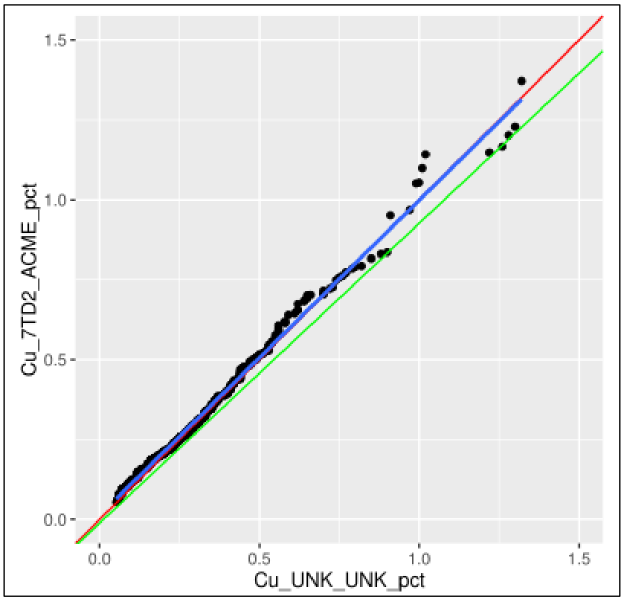
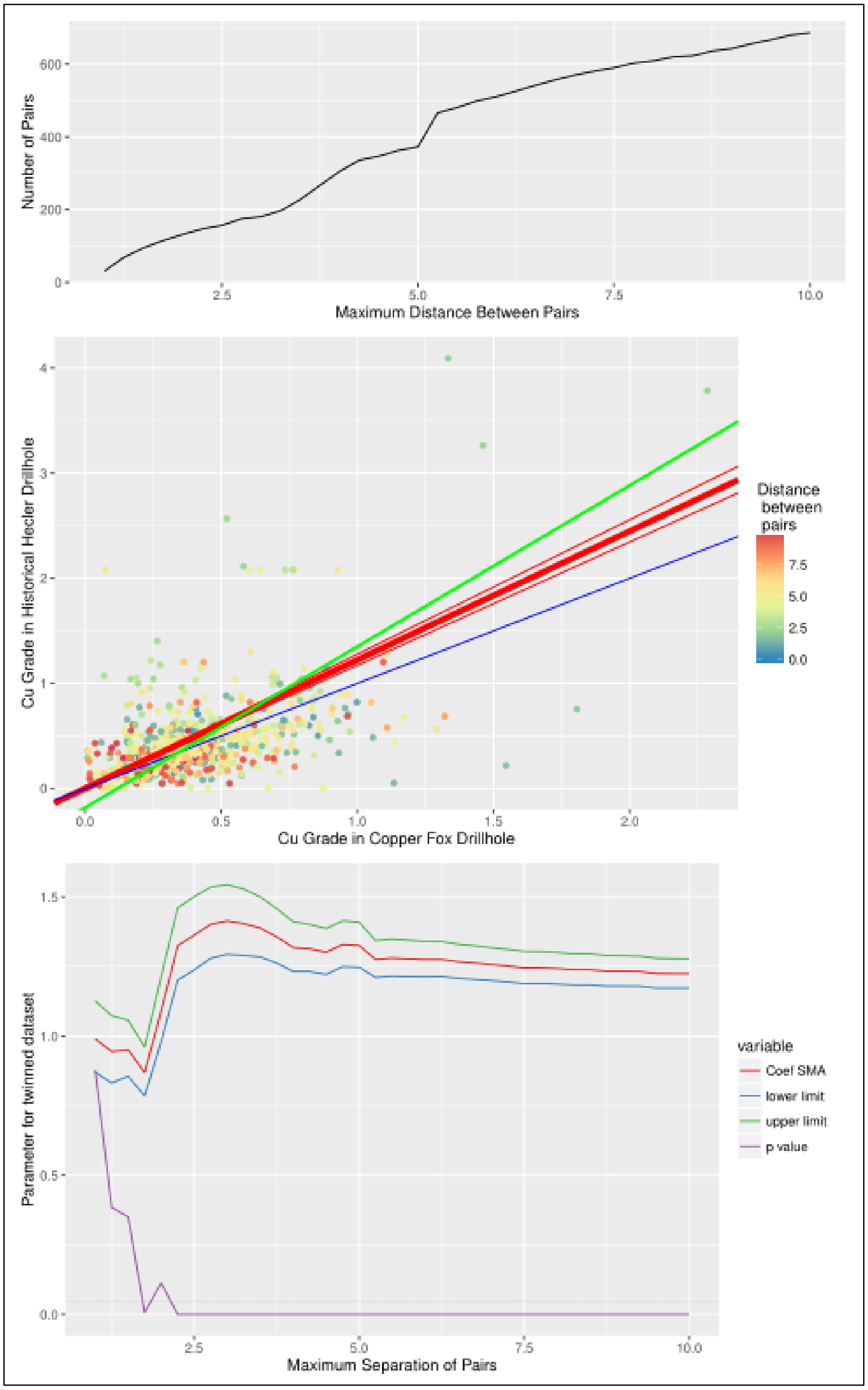
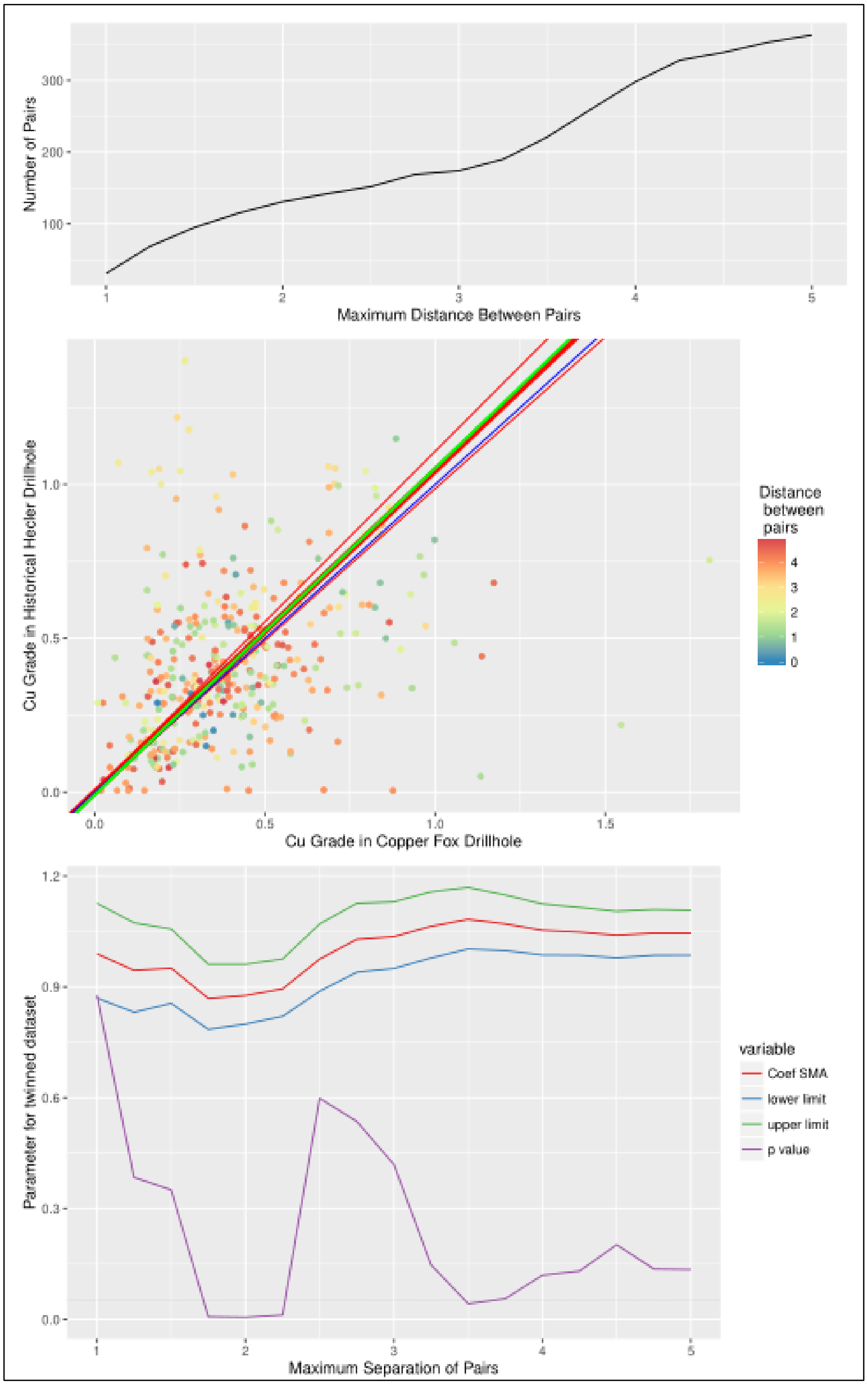


Figure 11-35: Uncensored Copper Duplicated Data



Note: There are regular samples > 2% Cu in the historical assays that are not reflected in the more modern assays.

Figure 11-36: Paired Sample Comparison Plots for Censored Copper in the Hecla Data for All Sample Spacings up to 5 m



The molybdenum data once the outliers are removed from the historical datasets shows no bias between historical and modern data, which supports the AMEC recommendation to make no change to this dataset (Figure 11-37 and Figure 11-38).

```
Coefficients:
                elevation     slope
estimate    -0.0006118044 0.9869365
lower limit -0.0037741405 0.8954924
upper limit  0.0025505317 1.0877184

H0 : variables uncorrelated
R-squared : 0.1665657
P-value : 3.6905e-15

------------------------------------------------------------
H0 : slope not different from 1
Test statistic : r= -0.0144 with 340 degrees of freedom under H0
P-value : 0.79071
```

Figure 11-37: Paired Sample Comparison Plots for Censored Molybdenum in the Hecla Data for All Sample Spacings up to 5 m

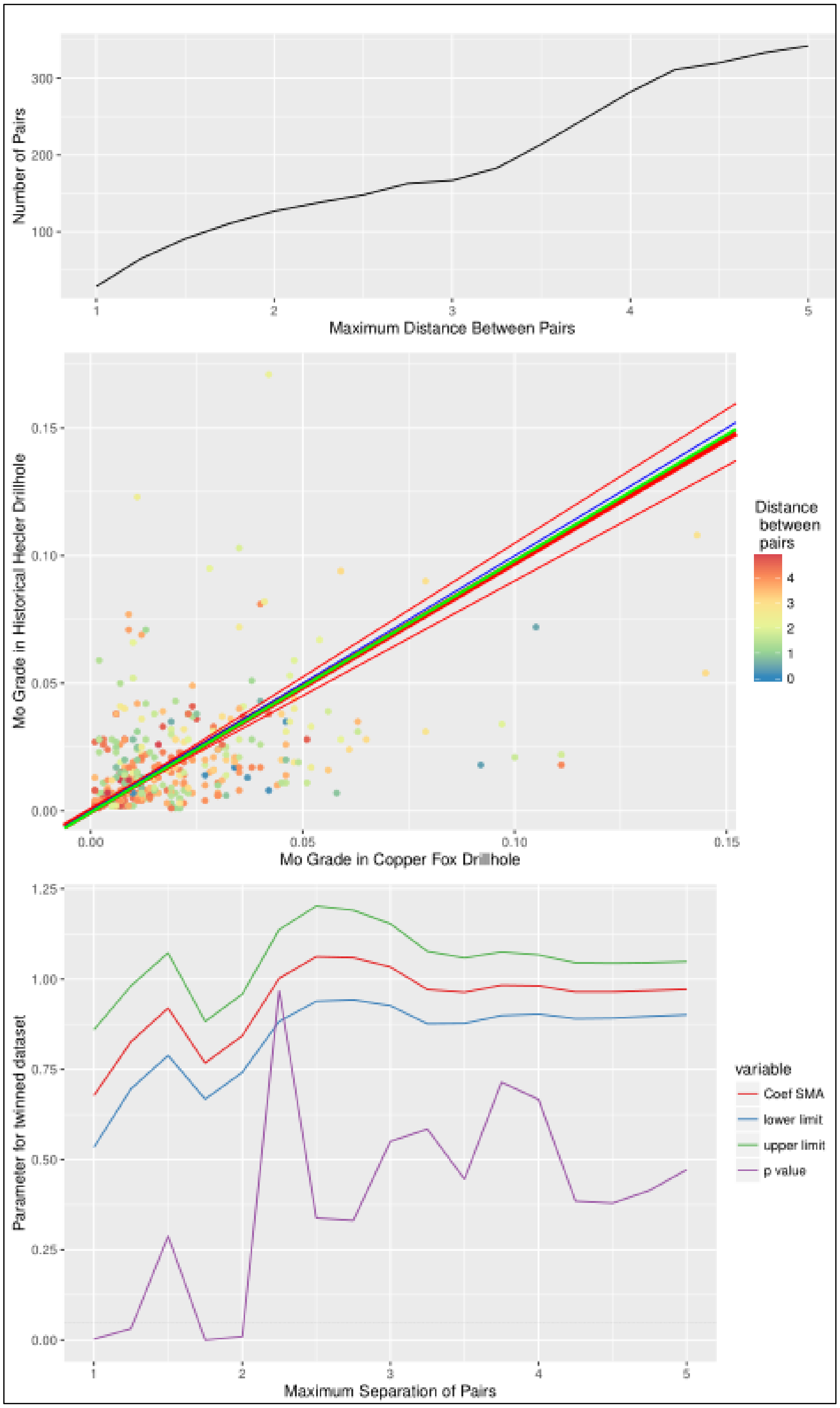
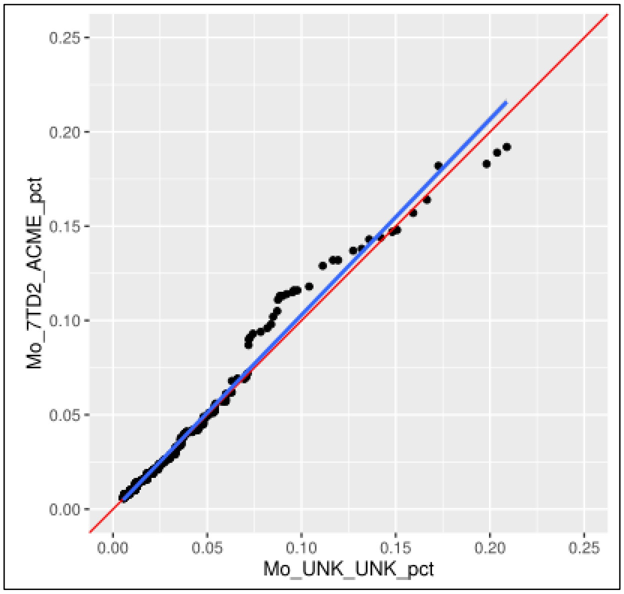


Figure 11-38: QQ Plots for Molybdenum in the Resampled Hecla Holes



11.3.2.3 Paramount Data

There are no Paramount twinned samples within 20 m, thus there is no way to assess the accuracy of these data. A subset of Paramount holes were reassayed on different sample intervals by Copper Fox using the 7TD package from ACME. The QQ plots comparing these datasets show no bias in the historical assays for copper and a low bias in the historical molybdenum assays (Figure 11-39). In the first case, this can be used as evidence to support the inclusion of the historical copper assays in the final dataset. For molybdenum, the level of bias that is evident is typical of the difference between an aqua regia and a 4-acid digestion. This methodological difference has not been corrected for elsewhere in this dataset so it should not be corrected for here.

The QQ plots for silver and gold are based on more limited data but show general agreement between data sources. Accepting the historical data is a conservative course of action in these cases (Figure 11-40).

Figure 11-39: QQ plots for Primary Assays and Resampling of Cu and Mo Data From the Paramount Generation of Drilling

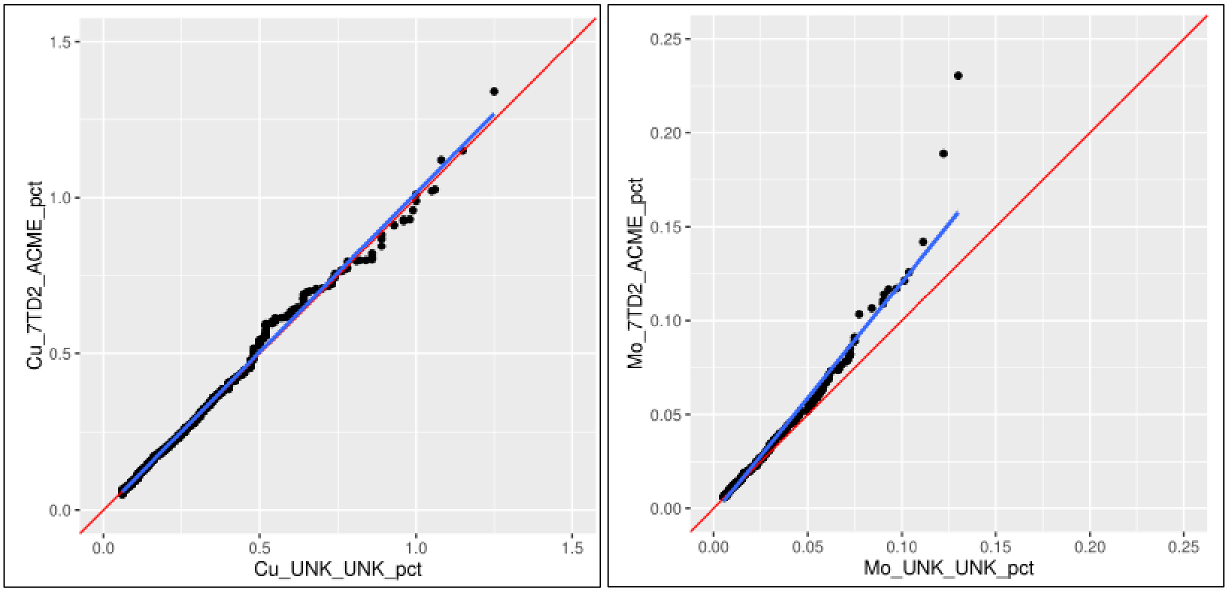
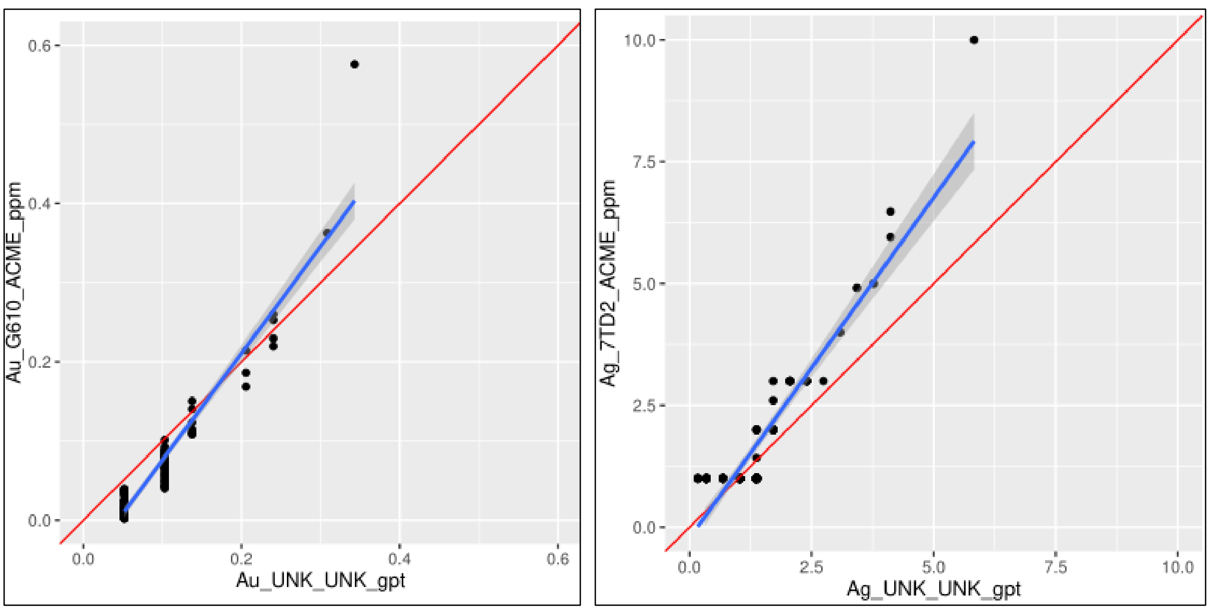


Figure 11-40: QQ Plots for Primary Assays and Resampling of Au and Ag Data From the Paramount Generation of Drilling



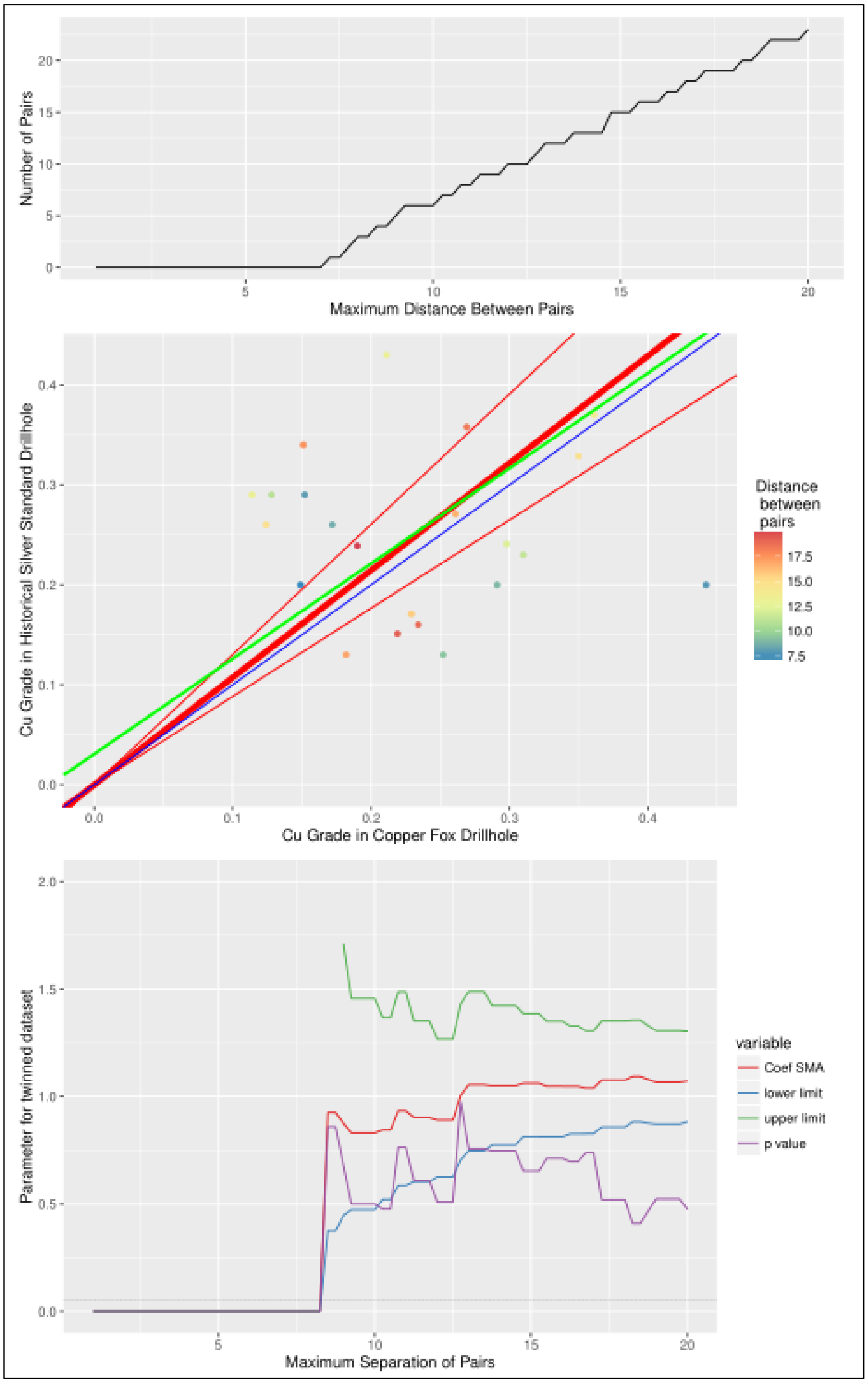
11.3.2.4 Silver Standard Data

There is no silver or gold data from Silver Standard campaign for comparison. Even within 20 m, there are only 20 NN points for comparison for copper and molybdenum, with none within 7 m. Because there are so few samples, the error in the slopes of the SMA are large. AMEC made no recommendations for corrections to these data.

In the case of copper, a slope and an error can be calculated that show that the slope is not statistically different to 1 (Figure 11-41). However, the crucial factor is that there is an 86% probability that the data are uncorrelated at both 10 m and 20 m sample spacing. If the null hypothesis that the data are uncorrelated is accepted, then nothing significant can be said about the slope. There cannot be any correction based on these data, and the modern data to support the accuracy of the Silver Standard copper data cannot be used.

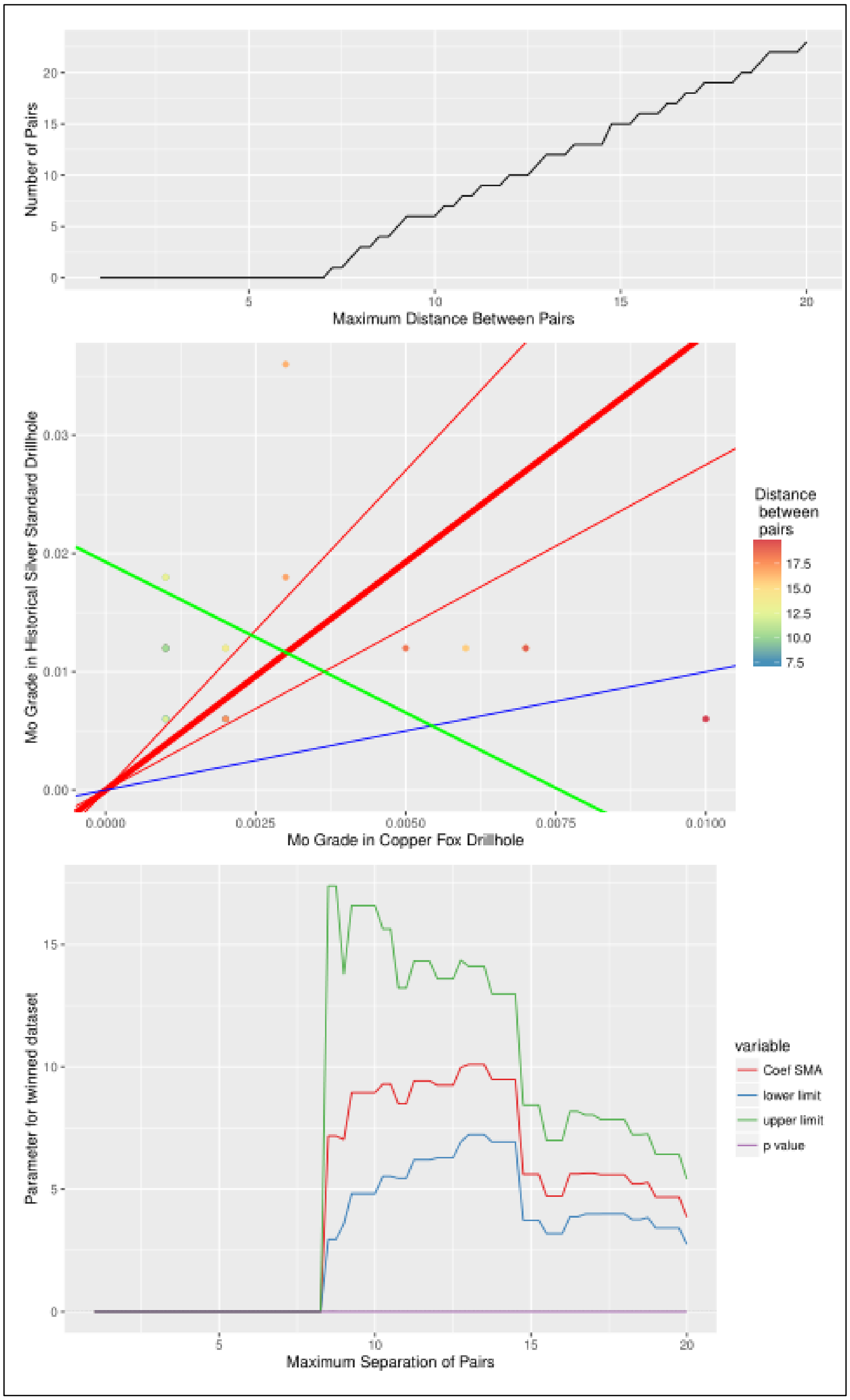
```
Coefficients:
              elevation     slope
estimate     0.03080624 0.9527499
lower limit -0.07976738 0.6140723
upper limit  0.14137985 1.4782174

H0 : variables uncorrelated
R-squared : 0.001358712
P-value : 0.86739

------------------------------------------------------------
H0 : slope not different from 1
Test statistic : r= -0.0484 with 21 degrees of freedom under H0
P-value : 0.82642
```

Figure 11-41: Paired Sample Comparison Plots for Copper in the Silver Standard Data for All Sample Spacings up to 20 m



There is likewise no statistically significant correlation between the historical and modern molybdenum data. The molybdenum data has the added complication that the historical data is within error of the detection limit for all comparison points within 20 m (Figure 11-42).

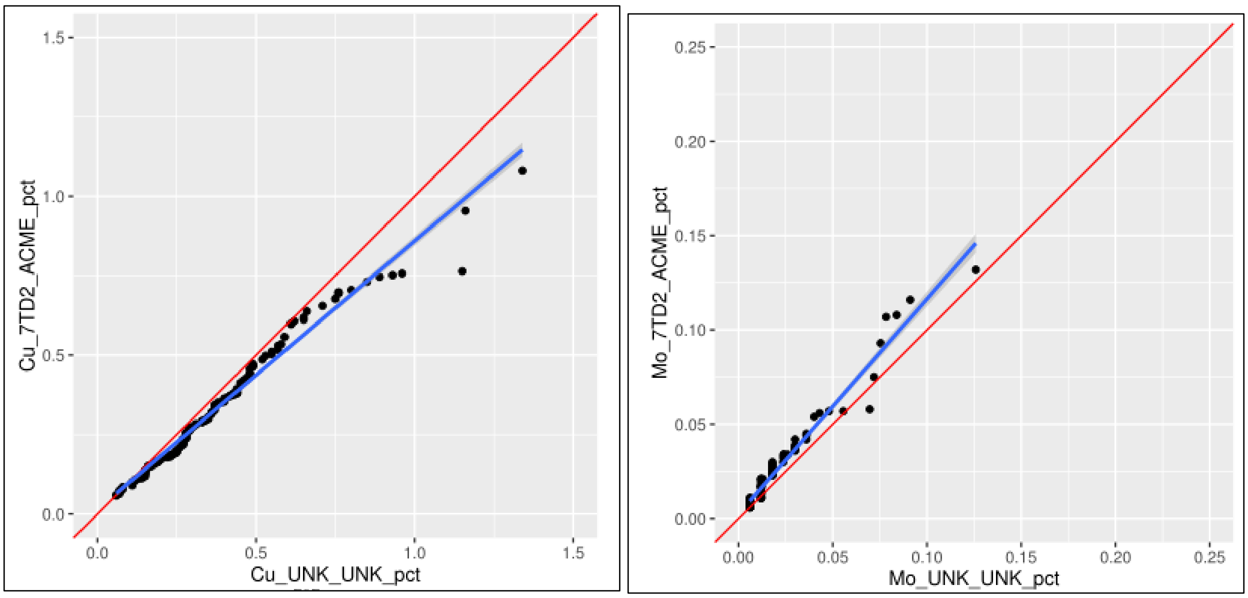
Figure 11-42: Paired Sample Comparison Plots for Molybdenum in the Silver Standard Data for All Sample Spacings up to 20 m



There are only two holes with modern reassays. They show no significant bias for copper or molybdenum (Figure 11-43). The slight high bias in historical assays for copper evident affects only approximately 10 samples in the Silver Standard dataset and can readily be ignored. It may well be that this is caused by the same oxidation as was evident in the Asarco generation data.

As was the case with the Paramount generation data, the bias in molybdenum is most readily explained by the historical use of aqua regia and the use of a mixed acid digestion in the reassays. It is recommended that these data can be accepted without correction in the resource estimate.

Figure 11-43: QQ Plots for Primary Assays and Resampling of Copper and Molybdenum from the Silver Standard Generation of Drilling



11.3.2.5 Teck Data (1980s data only)

The comparison of paired data with the older Teck data has no samples between 0 m and 3 m. There are approximately 80 samples within 5 m but more than 400 within 10 m. The only AMEC recommended change was for silver, as documented below. There was also no resampling of the historical core by Copper Fox, so consequently, no comparisons can be made with reassays.

- Ag – correction x = (y – 1.5064) / 0.5047
  The correction is applied up to a maximum of 3.04 g/t Ag. At higher grades, no correction is applied so that the corrected grades are not increased by applying the correction. Grades below 1.51 g/t Ag are re-set to zero.

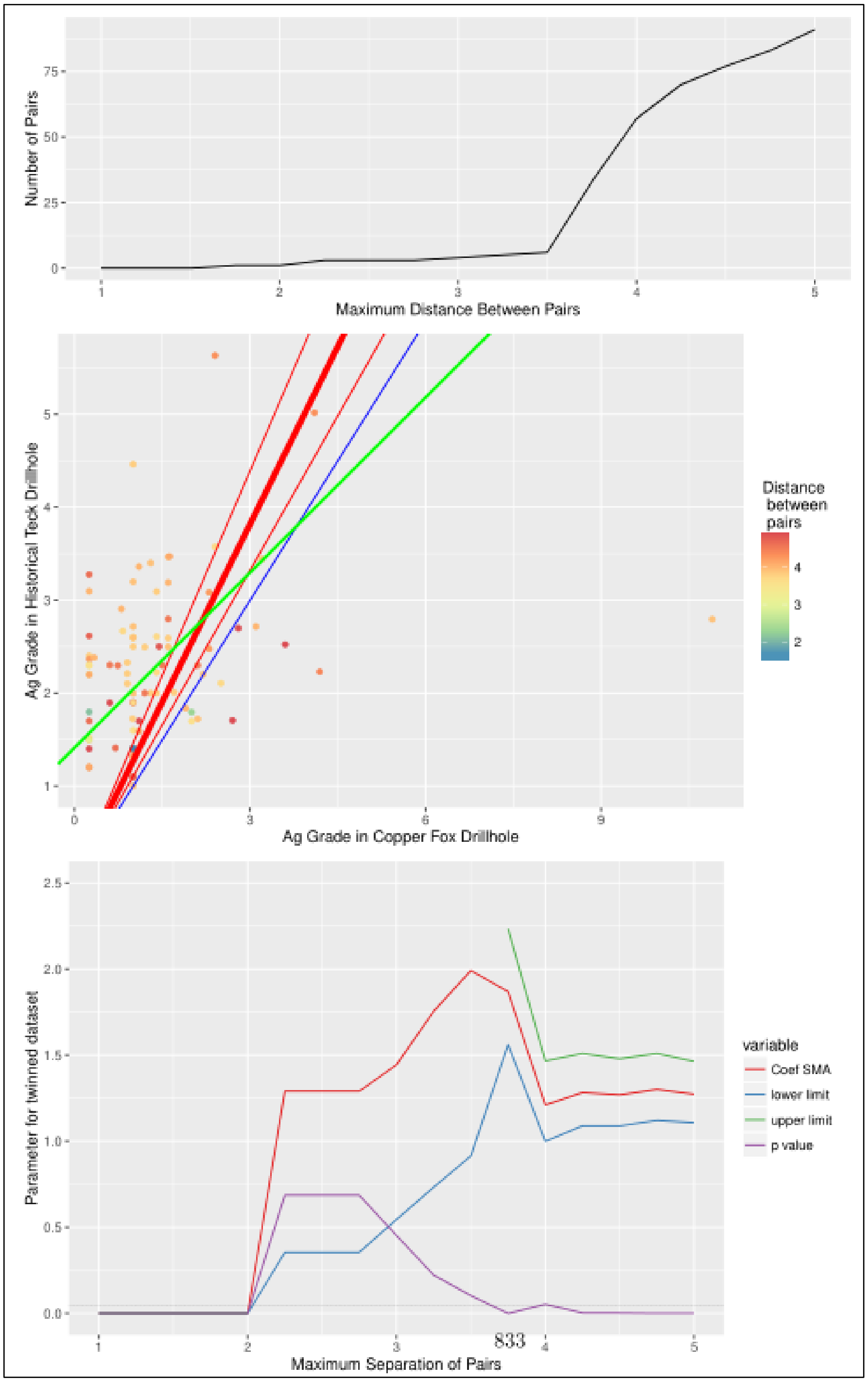
The historical Teck silver data is so extremely poor that it should be discarded (Figure 11-44). A comparison of points within 5 m shows that there is a very high predicted intercept in the SMA data. The predicted intercept and slope are similar to but not in total agreement with the AMEC recommendation.

```
Coefficients:
            elevation     slope
estimate     1.413590 0.6279874
lower limit  1.144311 0.5142203
upper limit  1.682870 0.7669245

H0 : variables uncorrelated
R-squared : 0.08742958
P-value : 0.004433

------------------------------------------------------------
H0 : slope not different from 1
Test statistic : r= -0.4506 with 89 degrees of freedom under H0
P-value : 7.3904e-06
```

If the much larger 10 m sample dataset is used, a high intercept is maintained (below), but the predicted slope for the SMA correlation changes from 0.51 to 1.41. This drastic inconsistency tends to suggest that it is not understood what the historical Teck silver data is showing relative to twinned samples, and so, it cannot be corrected for. The Schaft Creek JV recommendation is to reject these data.

```
Coefficients:
            elevation    slope
estimate    1.0167188 1.410502
lower limit 0.7423447 1.288877
upper limit 1.2910929 1.543603

H0 : variables uncorrelated
R-squared : 0.1667533
P-value : < 2.22e-16

------------------------------------------------------------
H0 : slope not different from 1
Test statistic : r= 0.3587 with 395 degrees of freedom under H0
P-value : 1.692e-13
```

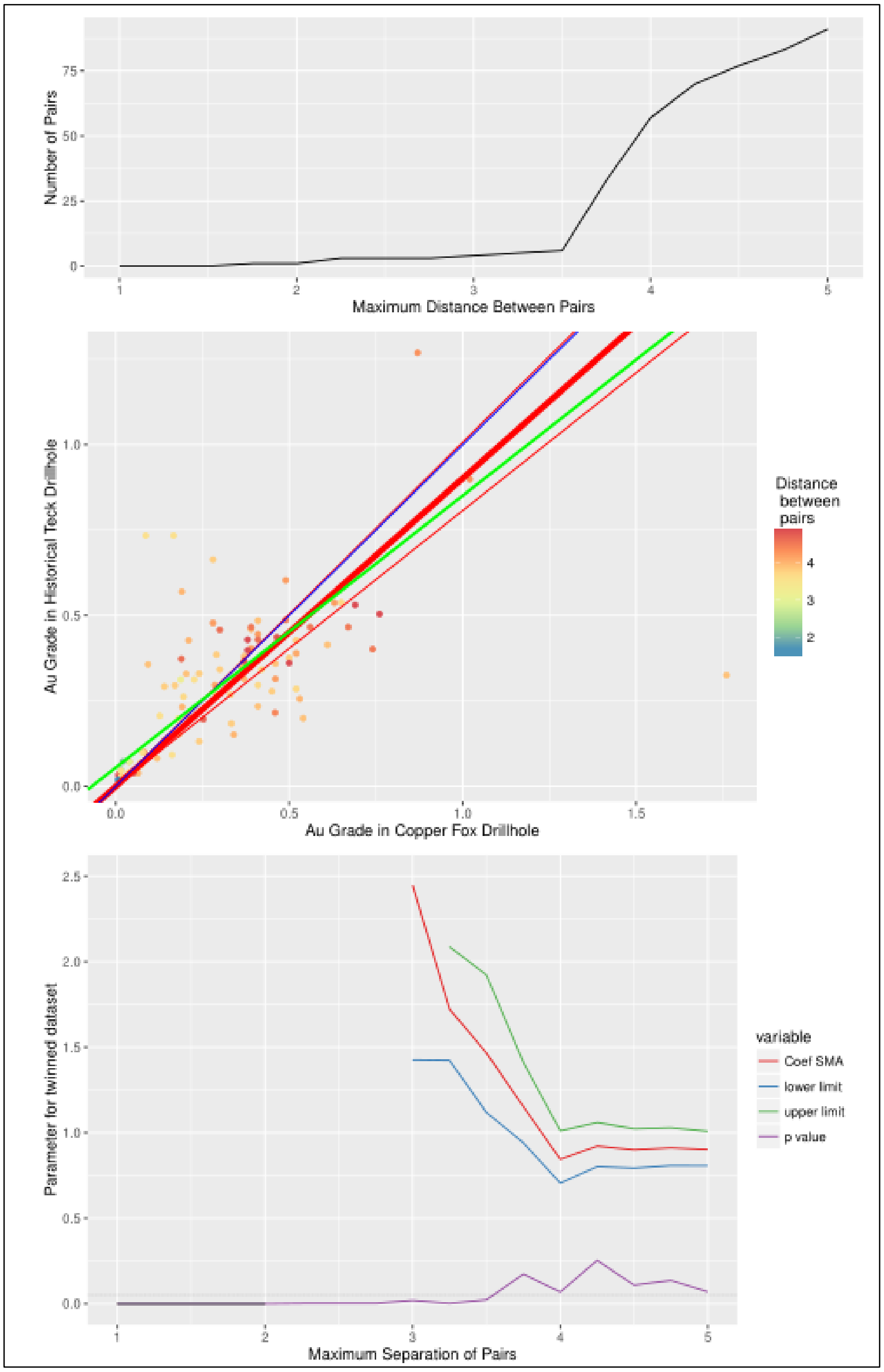
Figure 11-44: Paired Sample Comparison Plots for Silver in the Teck Data for All Sample Spacings up to 5 m



Gold in the historical data shows a reasonable correlation but with a bias. The unrestricted SMA shows a strong bias but that the intercept includes 0. Therefore, it is recommended to evaluate the bias using the SMA model forced through the intercept. If this is done (below), then the predicted slope correction is 0.9. However, this does not meet the 95% confidence that the slope is not different to 1. Therefore, the null hypothesis must be accepted and no correction made, in line with the previous recommendation from AMEC (Figure 11-45).

```
Coefficients (no intercept included):
            elevation     slope
estimate            0 0.9017426
lower limit        NA 0.8071517
upper limit        NA 1.0074188

H0 : variables uncorrelated
R-squared : 0.7188209
P-value : < 2.22e-16

------------------------------------------------------------
H0 : slope not different from 1
Test statistic : r= -0.1918 with 89 degrees of freedom under H0
P-value : 0.068569
```

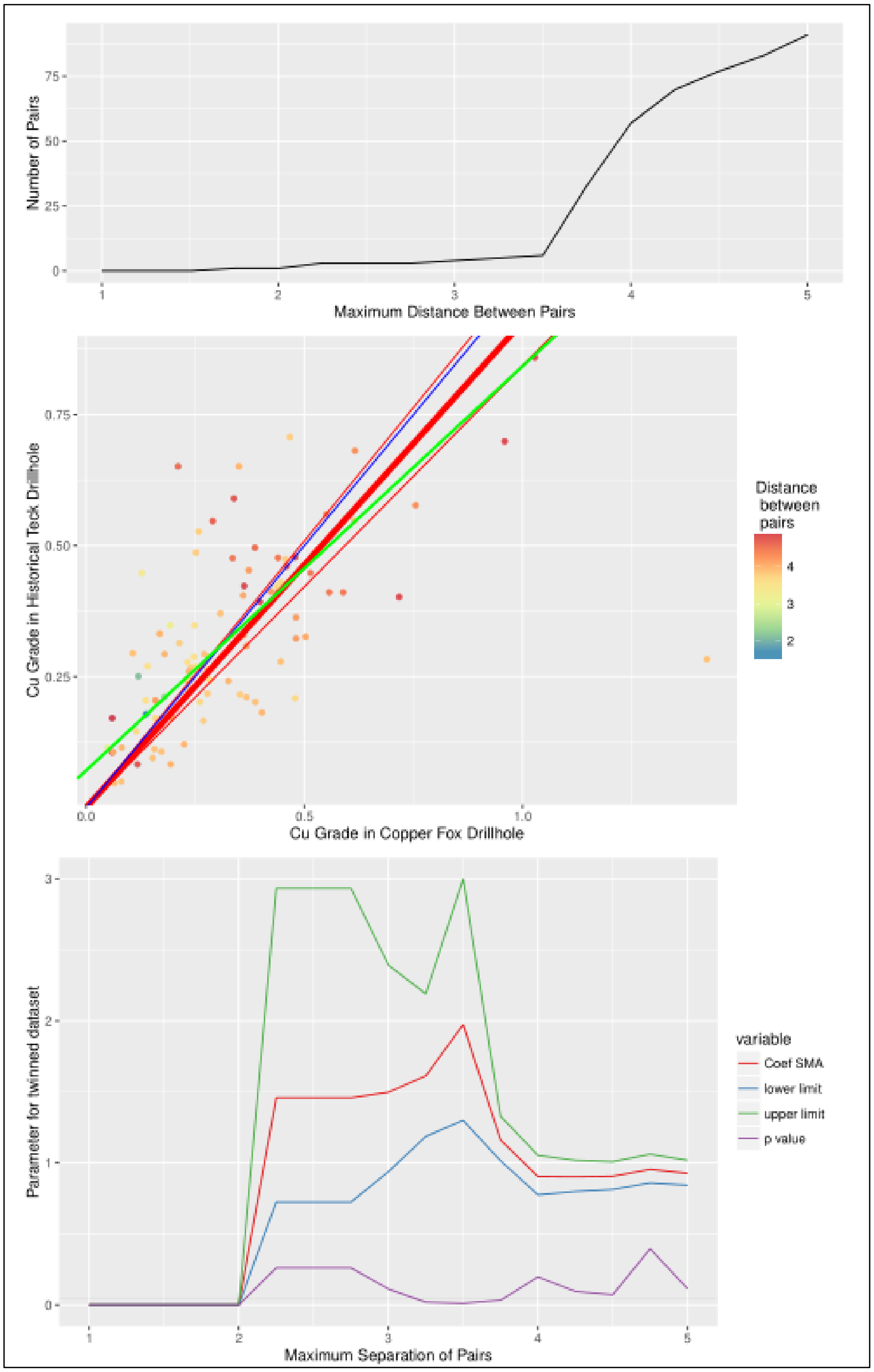
Figure 11-45: Paired Sample Comparison Plots for Gold in the Teck Data for All Sample Spacings up to 5 m



For copper, a similar chain of logic applies. If the SMA with no intercept is used, then it does not meet the 95% confidence limit that is required in the slope being different to 1 (Figure 11-46). The original data must be accepted. While it may appear somewhat biased, the data is of acceptable quality without correction.

```
Coefficients (no intercept included):
            elevation     slope
estimate            0 0.9265678
lower limit        NA 0.8432100
upper limit        NA 1.0181661

H0 : variables uncorrelated
R-squared : 0.7967488
P-value : < 2.22e-16

------------------------------------------------------------
H0 : slope not different from 1
Test statistic : r= -0.167 with 89 degrees of freedom under H0
P-value : 0.11366
```

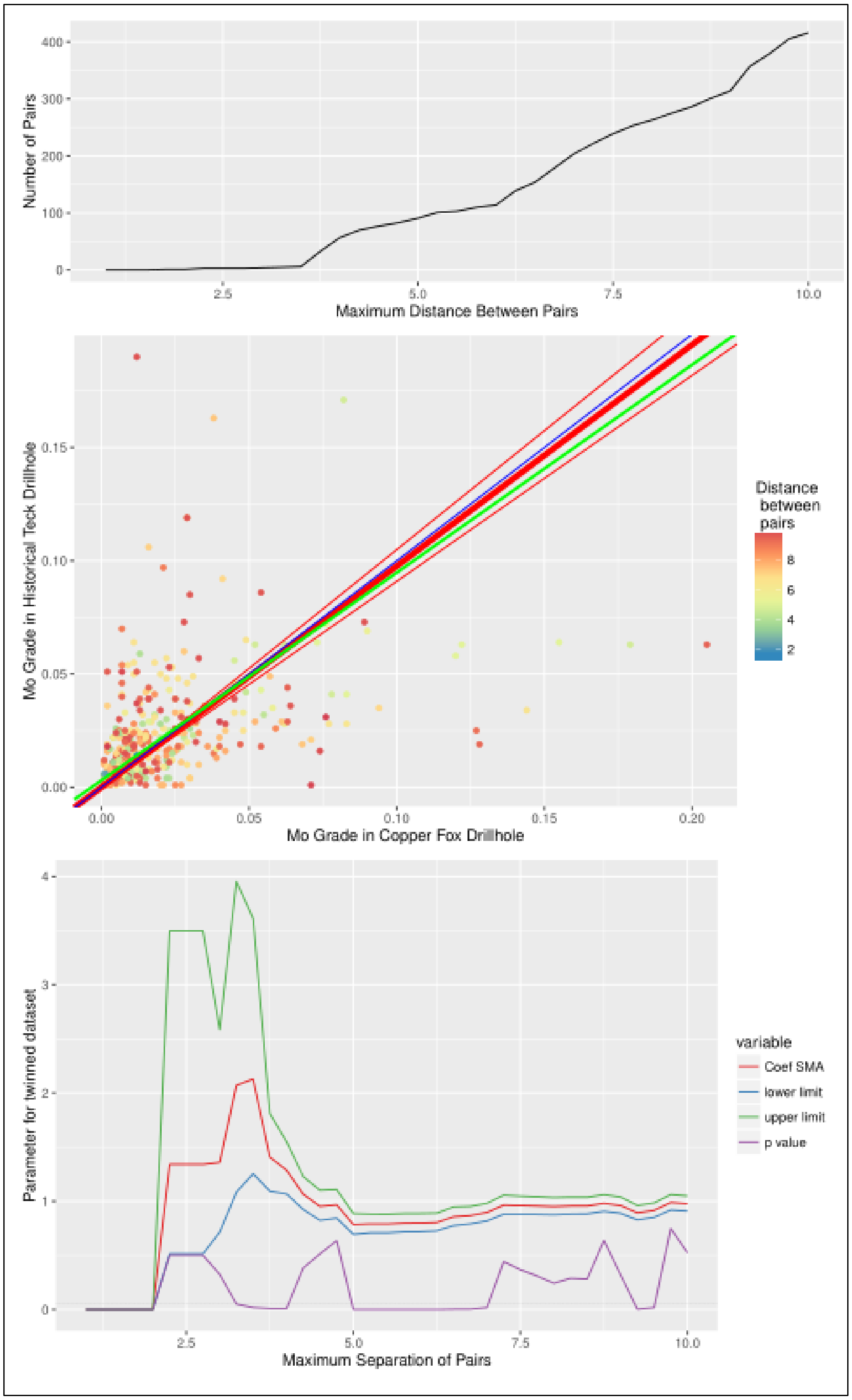
Figure 11-46: Paired Sample Comparison Plots for Cu in the Teck Data for All Sample Spacings up to 5 m



The molybdenum data (Figure 11-47) is virtually the same as for copper in the Teck 1980s data. In this case, the 10 m dataset was chosen because of a larger number of outliers in the 5 m dataset. The more robust dataset makes the effect of these outliers less significant. But again, the slope of regression fails to meet the 95% confidence interval that would suggest a correction should be made. Therefore, the data must be accepted as it is in line with AMEC's recommendations.

```
Coefficients (no intercept included):
            elevation     slope
estimate            0 0.9769554
lower limit        NA 0.9093472
upper limit        NA 1.0495901

H0 : variables uncorrelated
R-squared : 0.4466921
P-value : < 2.22e-16

------------------------------------------------------------
H0 : slope not different from 1
Test statistic : r= -0.03133 with 414 degrees of freedom under H0
P-value : 0.52396
```

Figure 11-47: Paired Sample Comparison Plots for Molybdenum in the Teck Data for All Sample Spacings up to 5 m



11.3.3 Recommendations for Further Work

There are two pieces of outstanding work to be completed for the next generation of study:

- The geological logs from samples listed in Table 11-8 need to be reviewed to see if there is any justification for the gravimetric fire assays on these samples that seem out of line with the remainder of the dataset.
- The Schaft Creek JV noted that in the Hecla drilling, there were multiple samples of very high copper grade that were not reflected in the NN samples from twinned drill holes. All these samples came from H72CH101 and H72CH091. A review of the drill logs, assays, and historical core for these holes would determine whether these samples were erroneous.

Table 11-8: Gravimetric Fire Assay Samples to be Validated

SAMPLEID	DUPLICATENO	HOLEID	DH_YEAR	RETURNDATE	SAMPFROM	SAMPTO	Ag_G613_ACME_gpt	Au_G612_ACME_gpt
611008	DUP	2010CF397	2010		47.8	49.8	662	
611008	PRIMARY	2010CF397	2010	3-Jan-11	47.8	49.8	686	
1053597	DUP	2011CF415	2011		75	77	2027	
1053597	PRIMARY	2011CF415	2011	7-Dec-11	75	77	2006	
1054879	PRIMARY	2011CF411	2011	19-Aug-11	206	209	1689	
1579183	PRIMARY	2012CF430	2012	14-Sep-12	130.75	132.15	414	
1053547	DUP	2011CF413	2011		560.5	562.5		18.6
1053547	PRIMARY	2011CF413	2011	23-Nov-11	560.5	562.5		20.2
1579000	DUP	2012CF427	2012		657	659		6.1
1579000	PRIMARY	2012CF427	2012	16-Aug-12	657	659		7.2

11.3.4 Corrections to Historical Data

There are a number of corrections recommended to be made in the exporting process for the acQuire database for the historical data in the Element_UNK_UNK_units columns for copper, molybdenum, silver, and gold. These are summarized below by each historic drilling program.

11.3.4.1 Asarco Generation Data

There are only primary copper and molybdenum data. No correction is recommended for copper (in line with previous resource estimates). The Schaft Creek JV recommends a correction for molybdenum that is slightly different to the previous AMEC recommendation.

Mo_corrected = Mo_UNK_UNK_pct/0.8504.

11.3.4.2 Hecla Generation Data

Previously, AMEC recommended a correction for silver, gold, and copper. The Schaft Creek JV recommends that there is insufficient evidence for a correction for silver or copper, both of which were relatively minor corrections in the AMEC dataset. The recommended correction for gold is marginally different to the correction recommended by AMEC.

Au_corrected = Au_UNK_UNK_pct/1.0806.

11.3.4.3 Paramount Generation Data

No correction is recommended in these data, in line with previous work.

11.3.4.4 Silver Standard Generation Data

No data exists for gold or silver. No correction to copper or molybdenum is warranted.

11.3.4.5 Teck 1980s Generation Data

As discussed, the Teck silver data is recommended for exclusion from the resource estimation process. This is inconsistent with the recommendations of AMEC used previously. No correction is warranted for copper, molybdenum, or gold.

11.3.5 BESTEL Comparisons

The new BESTEL column derives data from a different source compared to the previous _BESTEL calculations. Table 11-9 below shows the number of each method in the old and in the new _BESTEL calculations. The differences in total silver are due almost entirely to the exclusion of the 5312 Teck 1980s generation data.

Table 11-9: BESTEL Comparison

Method	DL	No. Samples	No. in old BESTEL	No. in new BESTEL
Ag_1DX1_ACME_ppm	0.1	2444	2091	2318
Ag_1EX_ACME_ppm	0.1	12368	12364	12368
Ag_1F04_ACME_ppb	0.002	111	111	0
Ag_1F06_ACME_ppb	0.002	10	10	1
Ag_7AR2_ACME_gpt	2	37	10	10
Ag_7TD2_ACME_gpt	2	21854	6922	2665
Ag_AQ200_ppm	0.1	2110	57	2004
Ag_AQ250_ppb	0.002	115	0	0
Ag_AQ252_ppb	0.002	28	28	5
Ag_FA_LOR_gpt	0.1	1089	1089	1089
Ag_G613_ACME_gpt	?	6	2	0
Ag_ICP_IPL_ppm	0.5	7717	2182	7258
Ag_MA200_ppm	0.1	67	2	5
Ag_MA370_gpt	2	2121	2115	0
Ag_MEICP61_ALS_gpt	0.5	14	0	0
Ag_MEICP61a_ALS_gpt	1	443	136	0
Ag_MEICP61a_ALS_ppm	1	479	285	0
Ag_MEMS61_ALS_ppm	0.02	926	612	639
Ag_MEMS62_ALS_ppm	0.02	642	0	154
Ag_OG62_ALS_gpt	1	278	0	0
Ag_UNK_UNK_opt	0.34286	8791	8795	3483
Total			36811	31999

table continues…

Method	DL	No. Samples	No. in old BESTEL	No. in new BESTEL
As_1DX1_ACME_ppm	0.5	2446	2114	2325
As_1EX_ACME_ppm	1	8654	8654	8654
As_1F04_ACME_ppm	0.1	111	111	0
As_1F06_ACME_ppm	0.1	10	10	10
As_7AR2_ACME_pct	100	37	1	0
As_7TD2_ACME_pct	200	21855	5574	0
As_AQ200_ppm	0.5	2110	2110	2004
As_AQ250_ppm	0.1	115	0	0
As_AQ252_ppm	0.1	28	5	5
As_ICP_IPL_ppm	5	7717	6907	0
As_MA200_ppm	1	67	2	5
As_MA370_pct_ppm	200	2121	0	0
As_MEICP61_ALS_ppm	5	14	4	4
As_MEICP61a_ALS_ppm	50	922	586	0
As_MEMS61_ALS_ppm	0.2	926	877	925
Total			26955	13932
Method	DL	No. Samples	No. in old BESTEL	No. in new BESTEL
Au_1DX1_ACME_ppb	0.0005	2446	84	0
Au_1EX_ACME_ppm	0.1	8654	373	0
Au_1F04_ACME_ppb	0.0002	111	0	0
Au_1F06_ACME_ppb	0.001	10	1	0
Au_AA26_ALS_gpt	0.01	640	642	88
Au_AQ200_ppb	0.0005	2110	56	0
Au_AQ250_ppb	0.0004	115	0	0
Au_AQ252_ppb	0.0001	28	5	0
Au_FA_LOR_gpt	0.01	1089	891	891
Au_FA430_ppm	0.005	2110	2110	2001
Au_FAAAS_LOR_gpt	0.01	5734	1488	1517
Au_G601_ACME_gpt	0.005	8425	8408	8425
Au_G610_ACME_gpt	0.005	13447	12910	13447
Au_G612_ACME_gpt	0.9	4	2	0
Au_ICP21_ALS_gpt	0.001	280	0	8
Au_MA200_ppm	0.1	67	2	0
Au_UNK_UNK_opt	0.003429	8808	8808	8808
Total			35780	35185

table continues…

Method	DL	No. Samples	No. in old BESTEL	No. in new BESTEL
Cu_1DX1_ACME_ppm	0.00001	2435	84	237
Cu_1EX_ACME_ppm	0.00001	8576	357	367
Cu_1F04_ACME_ppm	0.00001	111	0	0
Cu_1F06_ACME_ppm	0.00001	10	1	1
Cu_7AR2_ACME_pct	0.02	37	1	1
Cu_7TD2_ACME_pct	0.001	21855	21854	21855
Cu_AQ200_ppm	0.00001	2106	57	4
Cu_AQ250_ppm	0.0001	115	0	0
Cu_AQ252_ppm	0.0001	27	0	0
Cu_ASY_IPL_Pct	0.01	1679	393	394
Cu_FA_LOR_pct	0.001	1089	846	846
Cu_ICP_IPL_ppm	0.0001	7693	1789	1885
Cu_MA200_ppm	0.00076	63	2	5
Cu_MA370_pct	0.001	2121	2120	2005
Cu_MEICP61_ALS_pct	0.001	14	4	4
Cu_MEICP61a_ALS_pct	0.001	642	80	84
Cu_MEICP61a_ALS_ppm	0.001	280	0	0
Cu_MEMS61_ALS_ppm	0.0001	926	275	294
Cu_OG62_ALS_pct	0.001	278	0	8
Cu_UNK_UNK_pct	0.001	18249	18282	18249
Total			46145	46239
Method	DL	No. Samples	No. in old BESTEL	No. in new BESTEL
Mo_1DX1_ACME_ppm	0.00001	2445	84	237
Mo_1EX_ACME_ppm	0.00001	8650	8654	365
Mo_1F04_ACME_ppm	0.000022	111	0	0
Mo_1F06_ACME_ppm	0.000186	10	1	1
Mo_7AR2_ACME_pct	0.001	37	1	1
Mo_7TD2_ACME_pct	0.001	21855	13557	21855
Mo_AQ200_ppm	0.00001	2108	57	4
Mo_AQ250_ppm	0.000354	115	0	0
Mo_AQ252_ppm	0.000058	28	0	0
Mo_ASY_IPL_pct	0.0006	1007	764	809
Mo_ICP_IPL_pct	0.0001	2292	214	285
Mo_ICP_IPL_ppm	0.0001	5404	1947	2194
Mo_MA200_ppm	0.00006	67	2	5
Mo_MA370_pct	0.001	2121	2120	2005
Mo_MEICP61_ALS_pct	0.0001	14	4	0
Mo_MEICP61a_ALS_pct	0.001	642	80	88
Mo_MEICP61a_ALS_ppm	0.001	280	0	0
Mo_MEMS61_ALS_ppm	0.000042	926	275	19
Mo_OG62_ALS_pct	0.001	278	0	8
Mo_UNK_UNK_pct	0.0006	18124	18124	18124
Total			45884	46000

table continues…

Method	DL	No. Samples	No. in old BESTEL	No. in new BESTEL
Re_1F04_ACME_ppb	0.001	111	111	111
Re_1F06_ACME_ppb	0.001	10	10	9
Re_AQ200_ppb	0.01	2105	2105	2001
Re_AQ250_ppb	0.001	115	0	0
Re_AQ252_ppb	0.001	28	28	28
Re_MA200_ppm	0.005	67	65	67
Re_MEMS61_ALS_ppm	0.002	926	906	926
Total			3225	3142
Method	DL	No. Samples	No. in old BESTEL	No. in new BESTEL
S_1DX1_ACME_pct	0.05	2209	0	0
S_1EX_ACME_pct	0.1	8654	0	315
S_1F06_ACME_pct	0.42	10	0	1
S_7AR2_ACME_pct	0.05	37	0	1
S_7TD2_ACME_pct	0.05	21855	0	7988
S_AQ200_pct	0.05	2105	0	1
S_MA200_pct	0.1	67	0	5
S_MA370_pct	0.05	2121	0	2005
S_MEICP61_ALS_pct	0.04	14	0	0
S_MEICP61a_ALS_pct	0.05	922	0	96
S_MEMS61_ALS_ppm_pct	0.01	926	0	0
TotS_2A13_ACME_pct	0.02	15153	14913	15052
TotS_IR08_ALS_pct	0.01	926	650	294
S_TC000_pct	0.02	2260	0	0
Total			15563	25758

11.3.6 Assay Recommendations

Based on the findings detailed above, Table 11-10 contains the recommendation of which methods from the Copper Fox and more recent data should be accepted and with what order or rejected in the cascading _BESTEL calculations.

Table 11-10: Analytical Method Recommendations

Method	Status	Order
Ag_1DX1_ACME_ppm	Accept	1
Ag_AQ250_ppb	Accept	2
Ag_AQ200_ppm	Accept	3
Ag_1EX_ACME_ppm	Accept	4
Ag_MEMS62_ALS_ppm	Accept	5
Ag_FA_LOR_gpt	Accept	6
Ag_AQ252_ppb	Accept	7
Ag_1F04_ACME_ppb	Accept	8
Ag_1F06_ACME_ppb	Accept	9
Ag_MA200_ppm	Accept	10
Ag_MEICP61_ALS_gpt	Accept	11
Ag_MEMS61_ALS_ppm	Accept	12
Ag_ICP_IPL_ppm	Accept	13
Ag_7AR2_ACME_gpt	Partial Accept ≥ DL	14
Ag_7TD2_ACME_gpt	Partial Accept ≥ DL	15
Ag_UNK_UNK_opt	Corrections Needed	16
Ag_G613_ACME_gpt	Reject	
Ag_MA370_gpt	Reject	
Ag_MEICP61a_ALS_gpt	Reject	
Ag_MEICP61a_ALS_ppm	Reject	
Ag_OG62_ALS_gpt	Reject	

Method	Status	Order
As_1DX1_ACME_ppm	Accept	1
As_AQ200_ppm	Accept	2
As_1EX_ACME_ppm	Accept	3
As_AQ250_ppm	Accept	4
As_1F04_ACME_ppm	Accept	5
As_AQ252_ppm	Accept	6
As_1F06_ACME_ppm	Accept	7
As_MA200_ppm	Accept	8
As_MEMS61_ALS_ppm	Accept	9
As_MEICP61_ALS_ppm	Accept	10
As_7AR2_ACME_pct	Reject	
As_7TD2_ACME_pct	Reject	
As_ICP_IPL_ppm	Reject	
As_MA370_pct_ppm	Reject	
As_MEICP61a_ALS_ppm	Reject	

table continues…

Method	Status	Order
Au_G610_ACME_gpt	Accept	1
Au_G601_ACME_gpt	Accept	2
Au_AA26_ALS_gpt	Accept	3
Au_FA430_ppm	Accept	4
Au_FA_LOR_gpt	Accept	5
Au_FAAAS_LOR_gpt	Accept	6
Au_ICP21_ALS_gpt	Accept	7
Au_UNK_UNK_opt	Corrections Needed	8
Au_1DX1_ACME_ppb	Reject	
Au_1EX_ACME_ppm	Reject	
Au_1F04_ACME_ppb	Reject	
Au_1F06_ACME_ppb	Reject	
Au_AQ200_ppb	Reject	
Au_AQ250_ppb	Reject	
Au_AQ252_ppb	Reject	
Au_G612_ACME_gpt	Reject	
Au_MA200_ppm	Reject	

Method	Status	Order
Cu_MA370_pct	Accept	1
Cu_7TD2_ACME_pct	Accept	2
Cu_1EX_ACME_ppm	Accept	3
Cu_AQ200_ppm	Accept	4
Cu_AQ250_ppm	Accept	5
Cu_7AR2_ACME_pct	Accept	6
Cu_MEICP61a_ALS_pct	Accept	7
Cu_FA_LOR_pct	Accept	8
Cu_ASY_IPL_Pct	Accept	9
Cu_ICP_IPL_ppm	Accept	10
Cu_1DX1_ACME_ppm	Accept	11
Cu_OG62_ALS_pct	Accept	12
Cu_MA200_ppm	Accept	13
Cu_AQ252_ppm	Accept	14
Cu_1F04_ACME_ppm	Accept	15
Cu_1F06_ACME_ppm	Accept	16
Cu_MEICP61_ALS_pct	Accept	17
Cu_MEICP61a_ALS_ppm	Accept	18
Cu_MEMS61_ALS_ppm	Accept	19
Cu_UNK_UNK_pct	Accept	20

table continues…

Method	Status	Order
Mo_7TD2_ACME_pct	Accept	1
Mo_MA370_pct	Accept	2
Mo_1EX_ACME_ppm	Accept	3
Mo_7AR2_ACME_pct	Accept	4
Mo_1DX1_ACME_ppm	Accept	5
Mo_AQ200_ppm	Accept	6
Mo_MEICP61a_ALS_pct	Accept	7
Mo_ASY_IPL_pct	Accept	8
Mo_ICP_IPL_pct	Accept	9
Mo_ICP_IPL_ppm	Accept	10
Mo_MA200_ppm	Accept	11
Mo_1F04_ACME_ppm	Accept	12
Mo_1F06_ACME_ppm	Accept	13
Mo_AQ250_ppm	Accept	14
Mo_AQ252_ppm	Accept	15
Mo_OG62_ALS_pct	Accept	16
Mo_MEICP61a_ALS_ppm	Accept	17
Mo_MEICP61_ALS_pct	Accept	18
Mo_MEMS61_ALS_ppm	Accept	19
Mo_UNK_UNK_pct	Corrections Needed	20

Method	Status	Order
Re_MEMS61_ALS_ppm	Accept	1
Re_1F04_ACME_ppb	Accept	2
Re_1F06_ACME_ppb	Accept	3
Re_AQ250_ppb	Accept	4
Re_AQ252_ppb	Accept	5
Re_MA200_ppm	Accept	6
Re_AQ200_ppb	Accept	7

table continues…

Method	Status	Order
TotS_2A13_ACME_pct	Accept	1
S_1DX1_ACME_pct	Accept	2
S_7TD2_ACME_pct	Accept	3
S_MA370_pct	Accept	4
S_1EX_ACME_pct	Accept	5
S_AQ200_pct	Accept	6
S_TC000_pct	Accept	7
S_MEICP61a_ALS_pct	Accept	8
TotS_IR08_ALS_pct	Accept	9
S_7AR2_ACME_pct	Accept	10
S_MA200_pct	Accept	11
S_1F06_ACME_pct	Accept	12
S_MEICP61_ALS_pct	Accept	13
S_MEMS61_ALS_ppm_pct	Accept	14

It is recommended that two corrections be applied to the historical datasets based on either twinned samples, resampling, or both. AMEC recommended making a correction to both these subsets of data also. AMEC also recommended other corrections previously. A review of these corrections indicates that there is insufficient evidence to support their use. The recommended corrections are shown below:

- The Asarco generation data justifies a correction to molybdenum:
  - Mo_corrected = Mo_UNK_UNK_pct/0.8504.
- The Hecla generation data justifies a correction to gold:
  - Au_corrected = Au_UNK_UNK_pct/1.0806.

The Teck 1980s generation data for silver has no QA/QC and no correlation with twinned samples. There is no reason to assume that these data are correct and they should be excluded from the resource estimate.

The new cascading _BESTEL calculation has consequences for the number of samples available for resource estimation. In the case of silver, there are approximately 5,000 fewer samples available than previously. This is almost entirely a consequence of the removal of the Teck 1980s generation data. There is a reduction in the number of gold samples available, but only 600 samples (< 2% of the dataset) as a consequence of the removal of specific unsuitable methods. The number of available copper and molybdenum samples are essentially unchanged.

All recommendations for changes were implemented in acQuire.

11.4 Density

11.4.1 Density from Previous Programs

From Caron et al. (2012):

> Ninety-six samples of full diameter drill core were selected from drill holes 2011CF407 through to 2011CF425 for specific gravity determination, averaging about five samples per drill hole. Samples were between 13 cm and 20 cm long and were selected by the geologist responsible as being representative of lithology, alteration, or mineralization. Samples were collected roughly every 100 m throughout each drill hole, or as dictated by changes in lithology/alteration/mineralization. A labelled wooden reference block was inserted in the core box from where the sample was collected. Samples were assigned a unique sample ID number and sent to the Acme laboratory for specific gravity measurement according to criteria specified by Copper Fox.
>
> The specific gravity samples were processed according to Acme laboratory code G813-WAX as follows: The core was first dried and weighed, then covered in wax to seal any fractures and to eliminate the porous nature of the core. The waxed core was then re-weighed to note the amount of wax, and then weighed in water. The specific gravity was then calculated as a ratio of the sample weight in air and the sample weight in water.

The QP is satisfied that the density determination procedures are industry standard and appropriate to provide data suitable for in situ bulk density estimation.

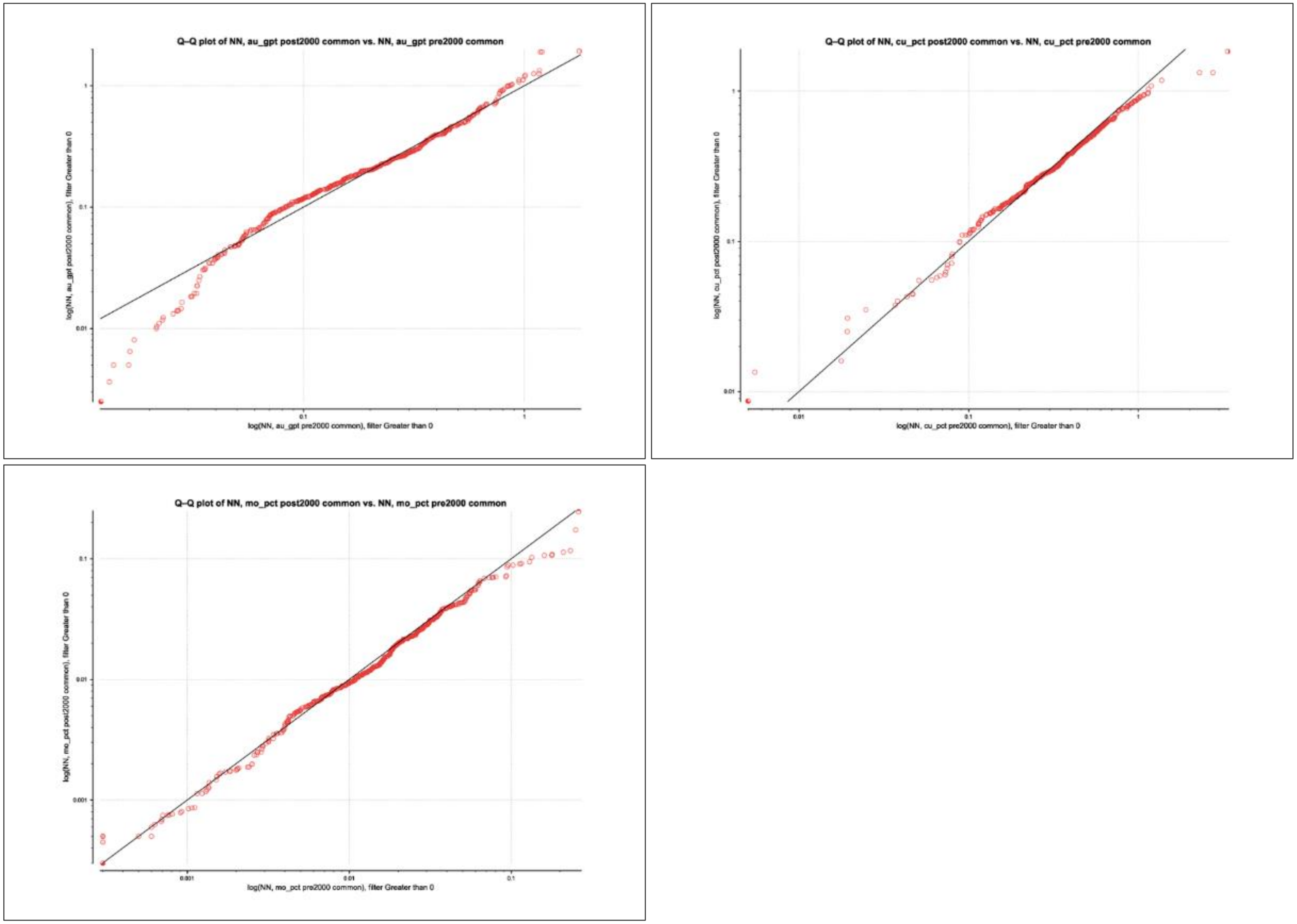
12.0 DATA VERIFICATION

12.1 Historic versus Current Drill Sampling Comparisons

The QP compared subsets of drill sampling from historic (pre-2000) and current (since 2000) campaigns to assess the risk of bias using Leapfrog EDGE™ software.

Subsets of assay data were prepared by locating historic and drillhole segments within 10 m of each other. The sampling data for historic and current sets within this common volume was composited to 6 m lengths, and the resulting 546 NNs were compared by means of QQ plots for gold, copper, and molybdenum (Figure 12-1). The red points represent the data and a black line representing the ideal condition where historic and current sample populations would be identical is shown for convenience.

Figure 12-1: Quantile Plots for Gold, Copper, and Molybdenum (Clockwise) Comparing Historic and Current Sampling Results



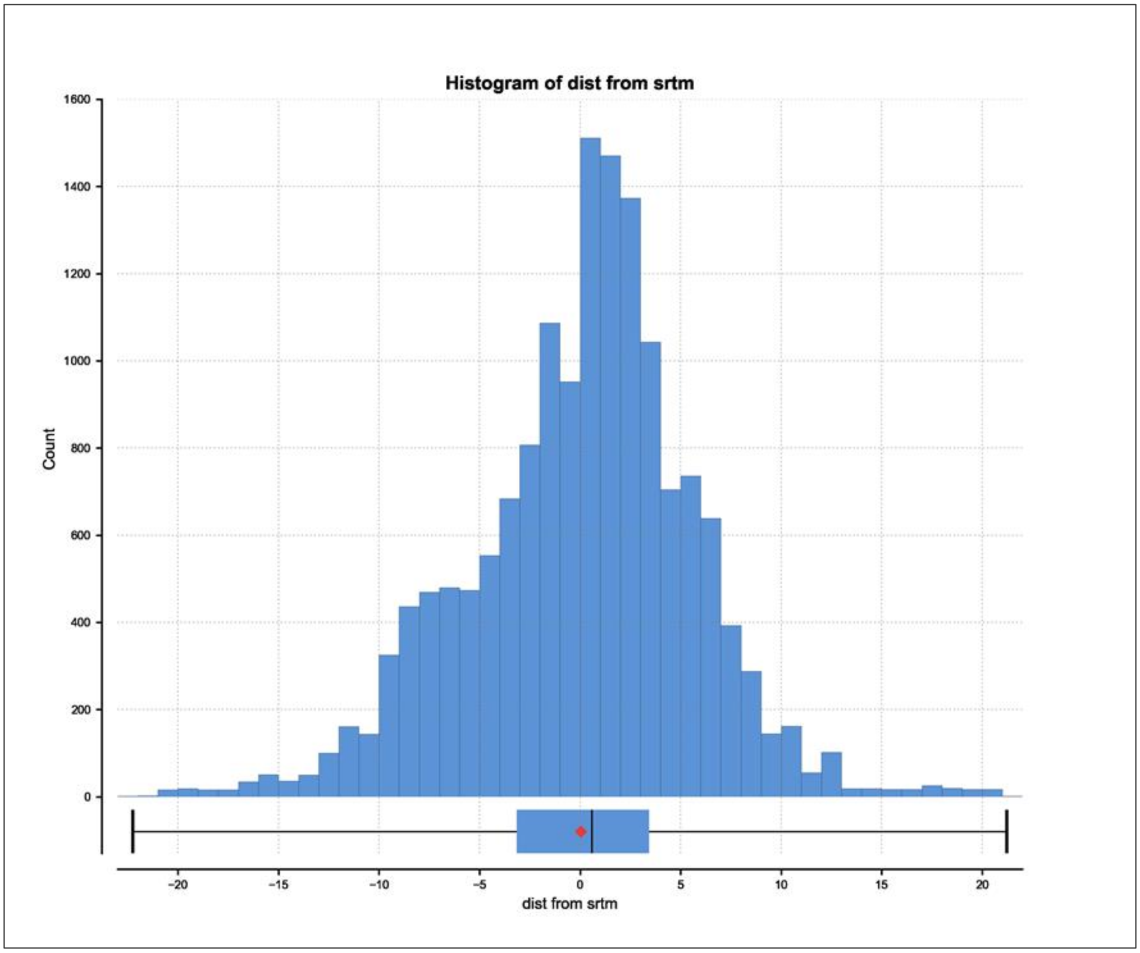
Historic and current sampling for all three elements within the comparable region appear to draw from the same populations with some statistical noise for low and very high-grade material. The QP is satisfied that there is no significant bias between the comparable historic and current sampling and assay data within the grade range of practical resource estimation for the assay data that has been retained in terms of the recommendations made in Section 11.3.6, “Assay Recommendations”.

12.2 Topography Verification

The provenance of the high definition topographic surface mesh (Dem_SCK) that was used for geological modelling and resource estimation is not documented. Tetra Tech compared this mesh with public domain data from NASA's SRTM.

The difference between the high definition topographic surface mesh and the SRTM mesh over the area covered by the resource block model was estimated using Leapfrog Geo™. The difference is summarised in Figure 12-2.

Figure 12-2: Histogram of the Difference Between Dem_SCK and SRTM Topography Data



The average difference is 0.038 m, and 90% of differences fall between -13 m and 13 m. The resource block model vertical dimension is 15 m, so the QP believes that the uncertainty in topography data does not present a material risk to the Project.

12.3 Site Visit Verifications

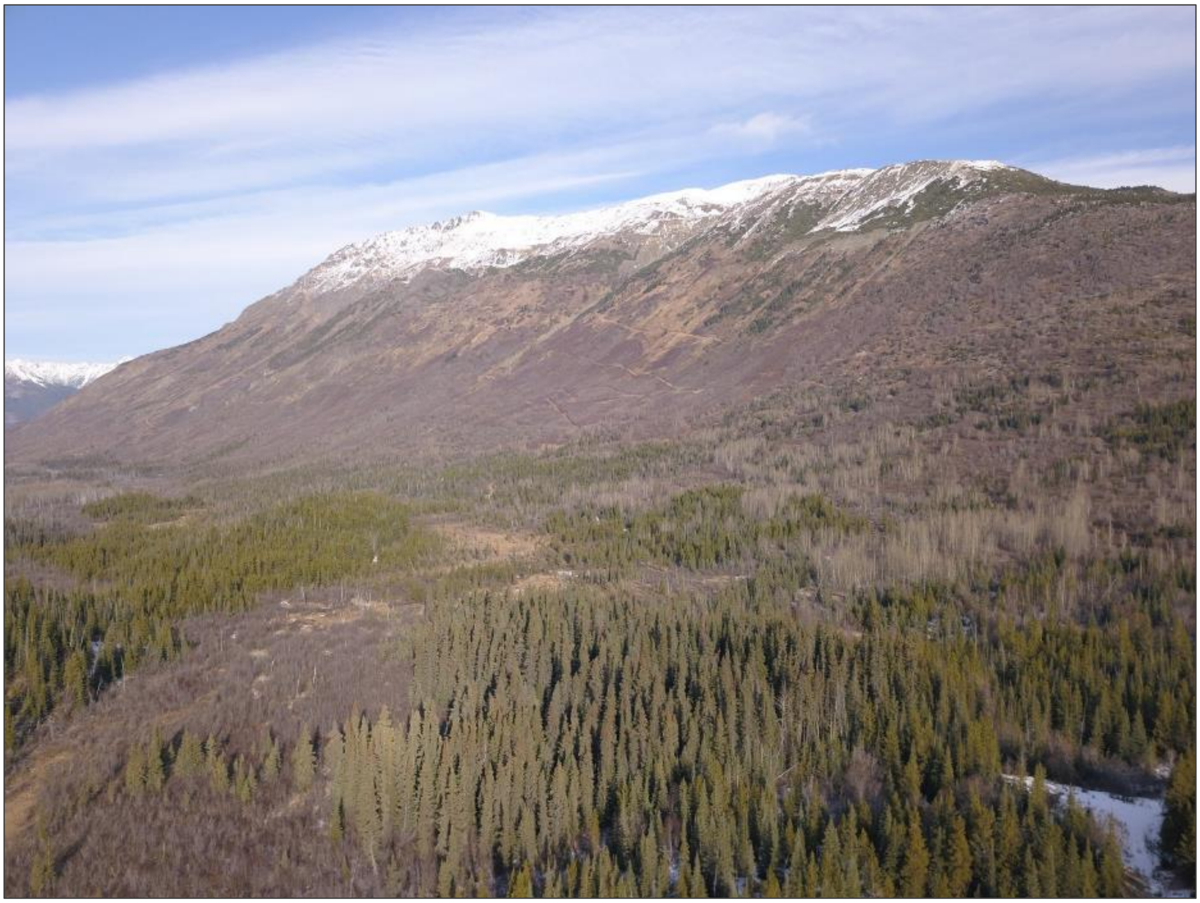
The Geology QP visited the site on Friday, October 30, 2020. The camp and core storage area appear to be in good order despite no site work for some time. A remote piloted aerial system (RPAS) was used for general photography of the site.

A general view of the core stacks and camp buildings is shown in Figure 12-3.

Figure 12-3: Schaft Creek Camp Looking Southwest Showing Camp Buildings and Core Stacks

A view of the deposit, with the network of drill roads and drill pads is shown in Figure 12-4.

Figure 12-4: Looking Northeast From Above the Camp Towards Mount LaCasse



Note: The deposit including the Paramount and Liard Zones occupies the middle ground of the image. The breccia units occur along the foot of the slope and trend north-south along the valley. A network of drill access tracks can be made out on the lower slopes of the mountain.

Several boxes of core were temporarily removed from the stacks at the camp and reviewed. The information is summarized in Table 12-1.

Table 12-1: Core Reviewed On Site

Video File	Time	Hole	From	To	Remarks	Lithology	lf_dom
DJI_0395	0	H-86-B15	297	312	Hole not found in database, old core, depths in feet?	Breccia	-

Video File	Time	Hole	From	To	Remarks	Lithology	lf_dom
DJI_0396	2:03	07CF316	41.5	43	-	Porphyritic Lava	-

table continues…

Video File	Time	Hole	From	To	Remarks	Lithology	lf_dom
DJI_0396	3:24	2010CF405B	125.58	128.94	Bornite, as it should ~1% Cu based on assays file.	Breccia	3200

table continues…

Video File	Time	Hole	From	To	Remarks	Lithology	lf_dom
-	-	2011CF410B	141.62	144.37	Acme labs ticket 586862 – correct ticket in database.	Diorite or Monzonite	3301

DJI_0397	0:26	08CF369	147.5	150.1	In database, low copper grade.	Breccia	3200

DJI_0398	0:04	SCK-13-433	106.9	109.5	Assays copper ~0.15%.	-	-

table continues…

Video File	Time	Hole	From	To	Remarks	Lithology	lf_dom
DJI_0398	2:41	SCK-15-440	180.1	182.7	Acme ticket 2307885 – ticket confirmed in database, negligible grades. Far north.	Andesite	-

The collar of inclined drill hole 07CF306 was located some 600 m north of the camp on the track to the main concentration of drill pads. Two GPS units, a Suunto Traverse Alpha and a Garmin GPSmap 66i were used to measure the collar position.

Source	Easting	Northing	Elevation	Horiz Difference	Elev Difference
GPSmap 66i	379,087.0	6,358,929.6	937.0	3.4	45.0
Suunto Traverse Alpha	379,092.2	6,358,931.6	892.0	2.1	12.2
Collar Database	379,090.1	6,358,930.9	879.8	-	-

The database plan position of the collar was in reasonable agreement with the two GPS measurements, but the elevation measurements from the two GPS units were poor. The collar elevation in the database agrees closely with both the detailed topographic surface model (Dem_SCK) and the SRTM data, so the elevation measurements provided by the GPS units are likely to be at fault.

(Reference *Global Positioning System Standard Positioning Service Performance Standard*, 5th Edition April 2020, Office of the Department of Defense Chief Information Officer Attn: Assistant for GPS, Positioning and Navigation 6000 Defense Pentagon Washington, DC 20301-6000.)

The following QPs conducted a site visit of the Property:

- Mr. Hassan Ghaffari, P.Eng. of Tetra Tech, visited the site on September 22, 2010 and conducted a general project site overview in the proposed infrastructure areas.
- Mr. John Huang, Ph.D., P.Eng. of Tetra Tech, visited the site on August 9, 2010 and conducted an overview of the proposed processing plant site.

- Mr. Michael O'Brien, P.Geo. of Red Pennant, visited the Property on October 30, 2020 and reviewed drill cores and the general layout of camp and topography.
- Mr. Daniel Friedman, P.Eng. of Knight Piésold, visited the site from July 28 to August 11, 2008 and conducted an overview of the proposed general project and TSF site.
- Mr. Brendon Masson, P.Eng. of McElhanney, visited the site on December 10, 2010 and conducted a general project site overview in the proposed access road areas.

12.4 Data Verification Conclusion

The QP is satisfied that the sampling and assay data, topographic information, and drill core management for this Project have been comprehensively verified and are suitable to be used for mineral resource estimation.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

The Schaft Creek deposit is a low-sulphidation, calc-alkalic, polymetallic (copper-molybdenum-gold-silver), porphyry deposit. Historically, the deposit was treated as three separate zones of mineralization; the Main (Liard), Paramount and West Breccia. In 2015, the mineralization was subdivided into four rock types: volcanic, intrusive, porphyry, and breccias to characterize their grindability and verify comminution circuit design (the Schaft Creek 2015 GeoMet program). The main focuses of the 2015 GeoMet program were to evaluate the primary comminution circuits proposed by the 2013 study and investigate the comminution variability for updating the mill throughput projection.

Several laboratories conducted metallurgical tests on samples from the various GeoMet units and mineralized zones of the Schaft Creek deposit to support various studies. The following laboratories undertook the major test programs:

- G&T/ALS
- PRA (Inspectorate)
- Hazen
- Polysius
- CESL

The main metallurgical test programs were conducted between 2004 and 2015, including mineralogy, flotation, grindability, and dewatering tests. The most recent metallurgical test program, focusing on comminution test work such as SMC tests and Bond BWi determination, was conducted by ALS. The primary grinding circuit was further assessed using JK simulations developed by SimSAG Pty Ltd. for blasting, crushing, and grinding processes. Table 13-1 summarizes the major test work programs for the Project.

Table 13-1: Major Metallurgical Testing Programs

Year	Program ID	Laboratory	Mineralogy	Flotation	Grindability	Others
2015	KM4658	ALS			√	
2012	KM3149	G&T	√	√	√	√
2010	KM2291	G&T	√	√		√
2009	KM2292	G&T		√		
2008	KM2136	G&T	√	√	√	
2008	2337 3326	Polysius			√	
2008	10736	Hazen			√	
2007	-	CESL				√
2007	PRA0701301	PRA		√		
2007	10515	Hazen			√	
2006	PRA0603303	PRA		√	√	
2005	PRA0502002	PRA	√	√	√	
2004	PRA0402903	PRA		√		√

Note: PRA Reports 0409111 were not available for the review.

This section summarizes the results of the various metallurgical investigations, including mineralogical characteristics, open batch test work, locked cycle tests, pilot plant test work, and metallurgical performance projections.

A comprehensive review of the historical test work is available in the Copper Fox FS on the Project, issued by Tetra Tech January 23, 2013. A summary of metallurgical and process characteristics used to inform project design are summarized below:

- Chalcopyrite is the dominant copper sulphide mineral together with lesser concentrations of bornite and chalcocite. The most significant non-copper sulphide present is pyrite.
- The other main sulphide mineral in the mineralization was pyrite. The pyrite contents in these mineral samples were relatively low. The 2010 test program showed a significantly lower pyrite content, ranging from 0.04 to 0.15% in comparison to the 0.4 to 1.0% pyrite contents reported in the 2008 test program. The 2012 test program showed 0.1 to 0.8% pyrite content, averaging at approximately 0.3%.
- Comminution characteristics indicate that the mineralized zones can be classified as hard with respect to SAG mill and ball mill grinding. The average A x b values for the breccia, intrusive and porphyry are functionally equivalent at approximately 34, while the volcanic lithology represents a distinctly harder mineralization type with an A x b value of 31. The BMWi value also varies distinctly with lithology ranging from 16.6 kWh/t to 22.4 kWh/t, indicative of a very hard mineralized material. The average Ai is 0.25 g, fluctuating from 0.17 g to 0.57 g.
- Test work to date supports a process primary grind size of 80 % passing 150 µm.

- The copper and molybdenum bulk flotation locked cycle test results showed that the mineral samples tested responded well to a simple, conventional process. Recovery is predominantly feed grade dependent, with some performance influence from copper mineralization.
- Bulk rougher regrind size requirements of 80 % passing 25 to 30 µm were determined in preparation for the subsequent three stage cleaner flotation process.
- At an average primary grind size of 80% passing 151 µm, G&T test results show that on average, 86.2% of the copper was recovered from the head sample containing approximately 0.37% copper. The other associated metal recoveries were 73.3% for gold, 55.7% for silver, and 71.9% for molybdenum. The average feed grades of the samples were approximately 0.27 g/t gold, 2.7 g/t silver, and 0.019% molybdenum. On average, the concentrate produced contained 30.9% copper. The average data from G&T and PRA show that at the primary grind size of 80% passing 146 µm, 86.7% of the copper reported to the copper concentrate at a grade of 29.9% copper. The gold, silver, and molybdenum recoveries to the concentrate were 74.7%, 56.9%, and 73.7%, respectively.
- Molybdenum separation process rougher concentrate regrind requirements were determined to be approximately 20 µm or finer. The inclusion of a leach facility for processing out of specification final molybdenum concentrate may be necessary.
- Multi-element assays on the bulk concentrates generated from the locked cycle tests showed, on average, that the impurities of the copper concentrates produced from the mineralization should be below smelting penalty thresholds set forth by most smelters.

13.2 Samples

Four distinct metallurgical test programs have been completed since 2008 (KM2136, KM2291/2292, KM3148 and KM4658).

The samples used for the 2008 to 2015 test work were collected from the following:

- Historical drilling programs
- The 2005 to 2011 drilling programs
- The 2015 drill program on the main mineralization zones

In 2015, the deposit mineralization was reclassified geometallurgically and sub-divided into four rock types: Volcanic, Intrusive, Porphyry, and Breccia. Samples of geometallurgically distinct units were then used to validate the 2013 FS comminution circuit design throughput projections and investigate the comminution throughput variability of the project. (Teck Schaft Creek 2015 GeoMet Program).

This historical variability work is deemed appropriate to inform this Preliminary Economic level assessment. The composites for the KM2291 program were created focusing on material representing the first five years of projected mine life. This material was used as source feed for testing conducted in KM2292 on a bulk sample in pilot plant testing. Samples tested in program KM3149 were comprised exclusively of material from the Paramount Zone which mineralogically and metallurgically differs from the main portion of the deposit as it is currently defined.

The 2015 test program, KM4658 was conducted to determine comminution characterization on the samples comprised of 101 discrete samples based on spatial location, lithology, alteration and geotechnical characteristics informed by the 2015 reclassification of the deposit.

The locations of the diamond drill holes used in the metallurgical test work before 2011 are shown in Figure 13-1.

13.2.1 2015 Test Samples

A total of 101 samples were collected and submitted for the comminution test work. The samples were collected based on spatial location, lithology, alteration, and geotechnical characteristics. Sampling was undertaken between June and August 2015 from the 70 drill holes:

- 47 samples from 23 drill holes of the 1969 to 1981 historical drill programs
- 48 samples from 43 drill holes of the 2005 to 2011 drill programs
- 6 samples from 4 drill holes of the 2015 drill program

13.2.2 2011/2012 Test Samples

The 2011/2012 test program used six composite samples, which were generated from drill hole numbers 398, 402, 405, 409, 399, 401, 403, 406, 407, 408, and 410, from the 2010/2011 drill program. The samples came from the Paramount Zone, including the West Breccia area. These samples covered a significant portion of the mineralized zone and included different grade classes as well as lithology and alteration. Master Composite 1 was comprised by blending an equal mass of each of the six individual composite samples. The head assays of the six samples and Master Composite 1 are shown in Table 13-2.

Figure 13-1: Schaft Creek Metallurgical Test Work Drill Hole Location Map

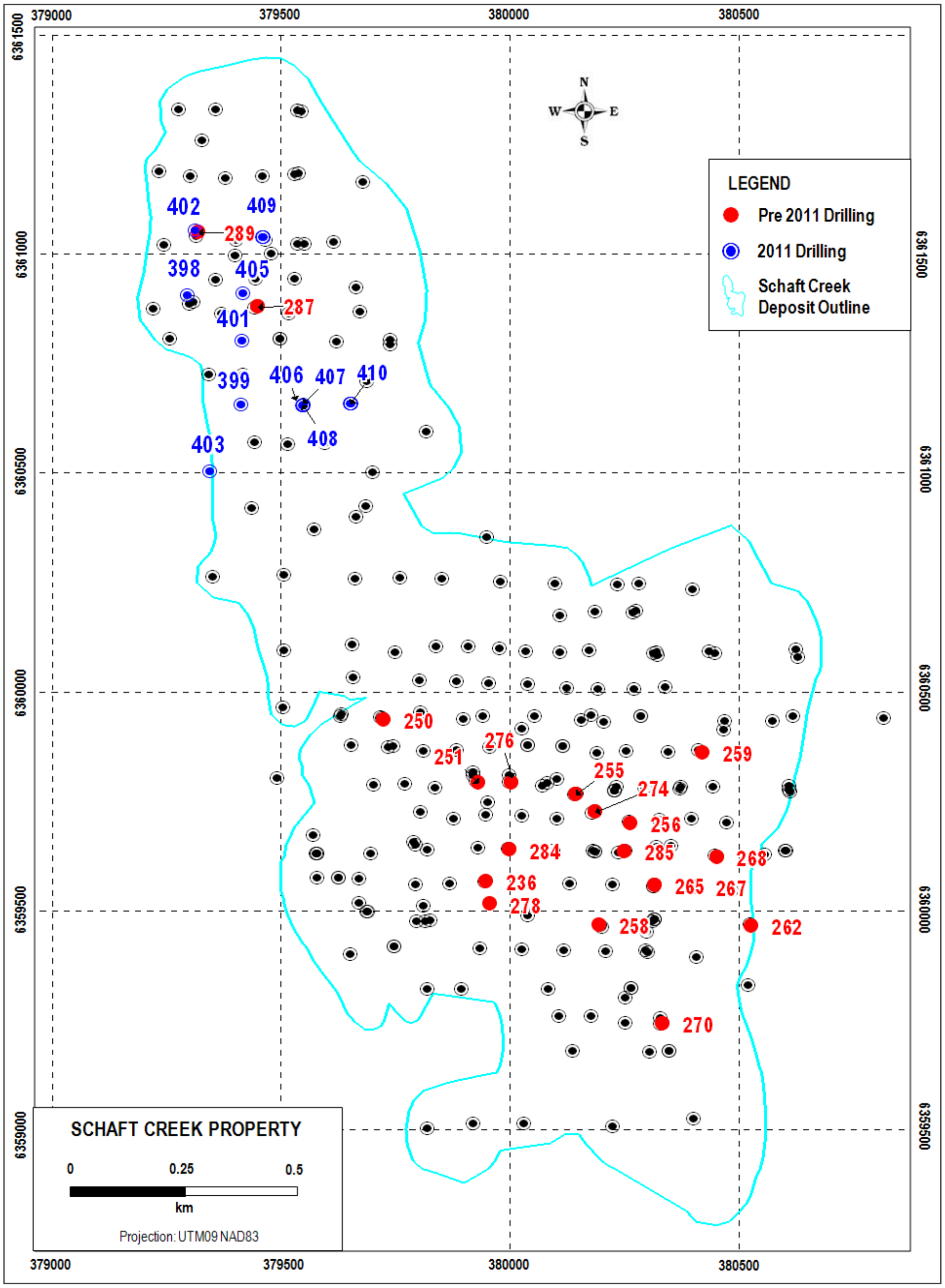


Table 13-2: Composite Samples, 2012 (G&T)

Composite ID	Au (g/t)	Ag (g/t)	Cu (%)	Mo (%)	Fe (%)
Composite 1	0.15	2	0.19	0.010	2.08
Composite 2	0.06	<1	0.17	0.009	3.30
Composite 3	0.13	2	0.30	0.017	2.55
Composite 4	0.08	2	0.37	0.027	1.62
Composite 5	0.48	4	0.79	0.040	2.64
Composite 6	0.66	4	0.71	0.054	2.29
Master Composite 1	0.23	2	0.38	0.023	2.55

13.2.3 2010 Test Samples

In 2010, a total of 275 kg drill core samples from 21 drill holes were combined into 5 composite samples, representing the initial consecutive 5-year mill feed based on the PFS mining schedule. The composites were identified as Composite 1 to Composite 5. The chemical analysis for the five samples is shown in Table 13-3.

Table 13-3: Head Assay Composite Samples, 2010 (G&T)

Composite ID	Au (g/t)	Ag (g/t)	Cu (%)	Cu Ox* (%)	Mo (%)	Fe (%)
Composite 1	0.35	2	0.40	0.03	0.015	3.1
Composite 2	0.28	3	0.36	0.03	0.019	3.5
Composite 3	0.45	3	0.48	0.03	0.034	3.8
Composite 4	0.24	2	0.33	0.03	0.023	3.6
Composite 5	0.21	3	0.28	0.03	0.018	3.0

*Weak acid-soluble copper

13.2.4 2009 Test Samples

In 2009, the testing program (KM2292) employed a total of 8,000 kg of samples from 22 drill holes. The samples representing the Liard Zone for the pilot plant tests were grouped into five pilot plant feed samples in five separate bins. The samples used are: 05CF236, 05CF239, 05CF244, 06CF250, 06CF251, 06CF254, 06CF255, 06CF256, 06CF258, 06CF259, 06CF260, 06CF261, 06CF262, 06CF265, 06CF268, 06CF269, 06CF284, 06CF285, 06CF286, 06CF287, 06CF289, and 06CF290.

The head assays on the five samples from the pilot plant feed bins are shown in Table 13-4.

Table 13-4: Head Assay Pilot Plant Test Sample, 2008 (G&T)

Composite ID	Au (g/t)	Ag (g/t)	Cu (%)	Cu Ox* (%)	Mo (%)	Fe (%)
Pilot Plant Bin 1	0.27	2	0.32	0.033	0.019	3.4
Pilot Plant Bin 2	0.27	2	0.33	0.036	0.017	3.2
Pilot Plant Bin 3	0.47	2	0.34	0.037	0.019	3.4
Pilot Plant Bin 4	0.19	2	0.29	0.038	0.013	3.4
Pilot Plant Bin 5	0.24	2	0.31	0.036	0.014	3.4
Average	0.29	2	0.32	0.036	0.016	3.4

*Weak acid-soluble copper

13.2.5 2008 Test Samples

In 2008, metallurgical variability tests were conducted using a master composite sample generated from the Liard Zone (to confirm primary grind size) along with 34 samples from Liard and Paramount Zones. The assay results of the master composite and variability test samples are shown in Table 13-5 and Table 13-6, respectively.

Table 13-5: Master Sample, 2008 (G&T)

Composite ID	Au (g/t)	Ag (g/t)	Cu (%)	Mo (%)	Fe (%)
Master Composite	0.24	2.5	0.32	0.010	3.32

Table 13-6: Head Assay Variability Test Samples, 2008 (G&T)

Composite ID	Au (g/t)	Ag (g/t)	Cu (%)	Mo (%)	Fe (%)
126636	0.72	4.5	0.60	0.004	1.95
126644	0.47	2.9	0.32	0.002	3.00
126649	0.14	1.8	0.21	<0.001	1.22
126694	0.90	4.9	0.77	0.017	2.81
126702	1.27	4.8	0.90	0.025	1.70
126705	0.50	2.8	0.36	<0.001	2.86
126710	0.66	3.8	0.48	<0.001	2.36
126712	0.28	1.8	0.16	0.001	1.82
126720	0.20	1.6	0.15	0.007	4.67

table continues…

Composite ID	Au	Ag	Cu	Mo	Fe
	(g/t)	(g/t)	(%)	(%)	(%)
126725	0.17	1.8	0.27	0.012	4.41
126737	0.51	2.3	0.63	0.010	4.32
126768	0.11	1.9	0.20	0.007	4.97
126777	0.27	1.5	0.37	0.007	3.80
126785	0.27	2.8	0.49	0.014	3.02
126787	0.42	5.8	0.76	0.006	3.67
126787QC	0.17	1.3	0.33	0.017	2.42
126799	0.51	5.2	0.90	0.014	1.29
126865	0.30	1.1	0.21	0.037	0.69
126876	0.30	3.4	0.38	0.032	3.51
126881	0.54	5.1	0.52	0.022	3.53
126882	0.73	4.6	0.51	0.064	3.62
126927	1.35	5.4	0.98	0.043	2.44
127121	0.07	1.2	0.17	0.005	2.83
127126	0.01	0.9	0.27	0.004	2.71
127145	0.06	6.5	1.17	0.016	1.68
127151	0.09	8.6	0.78	0.011	1.56
127197	0.61	7.3	1.05	0.001	2.64
127321	0.04	1.2	0.26	<0.001	4.82
127337	0.13	1.4	0.33	<0.001	3.73
127348	0.09	1.6	0.16	0.0009	3.88
127352	0.15	1.3	0.33	0.020	2.49
127352QC	0.31	6	0.74	0.008	3.87
127376	0.32	2.8	0.78	0.018	5.28
127377	0.41	2.8	0.91	0.016	5.74
127378	0.28	3.3	0.63	0.028	6.30
127573	0.25	3.0	0.38	0.092	4.14

As shown in Table 13-6, the head grades of the tested samples vary significantly from 0.15% to 1.17% for copper, from 0.01 g/t to 1.35 g/t for gold, from 0.9 g/t to 8.6 g/t for silver, from less than 0.001% to 0.092% for molybdenum, and from 0.69% to 6.30% for iron.

In early 2008, 273 individual drill core interval samples were grouped into 3 composite samples, representing the 3 main mineralization zones. The elements of interest were assayed for these composite samples and are shown in Table 13-7.

Table 13-7: Head Assay Composite Samples, 2008 (G&T)

Composite ID	Au (g/t)	Ag (g/t)	Cu (%)	Mo (%)	Fe (%)	S(t) (%)	C (%)
PZ Zone	0.19	2.0	0.28	0.017	2.7	0.56	0.79
WZ Zone	0.44	5.4	0.57	0.013	4.2	0.86	0.45
LZ Zone	0.23	2.1	0.28	0.012	3.9	0.35	1.02

Note: S(t) = total sulphur

13.3 Mineralogy

Several mineralogical examinations were conducted since 2005, including the analysis on the flotation product samples. The two main mineralogical studies were conducted by G&T on the composite samples for KM2050 (2008) and KM2291 (2010) test programs. A further mineralogical study was carried out by G&T on the composite samples KM3149 (2012). The key findings are summarized below:

- Chalcopyrite was the dominant copper sulphide mineral, together with ancillary bornite and chalcocite.
- The other non-copper sulphide mineral in the mineralization was mainly pyrite. The pyrite contents in these mineral samples were relatively low. The 2010 test program showed a significantly lower pyrite content, ranging from 0.04% to 0.15% in comparison to the 0.4% to 1.0% pyrite contents reported in the 2008 test program. The 2012 test program showed 0.1% to 0.8% pyrite contents, averaging approximately 0.3%.
- Most of the iron minerals in the samples from the 2010 test program were in oxide forms, such as magnetite, hematite, goethite, and limonite.
- Feldspar was the dominant silicate mineral, ranging between 44% and 52% of the total minerals.
- Moderate amounts of quartz, micas, and chlorite were also detected in the samples.

The estimated weight percentages of the major sulphide minerals and gangues are shown in Table 13-8.

Table 13-8: Mineral Composition, 2008/2010 (G&T)

Sample	Test Program							
	KM2291 (2010)					KM2050 (2008)		
	Composite					Composite		
	1	2	3	4	5	Paramount Zone	Liard Zone	West Breccia Zone
Chalcopyrite (%)	0.73	0.76	0.67	0.89	0.56	0.8	1.2	0.8
Bornite (%)	0.17	0.19	0.28	0.22	0.06	0.1	0.2	0.2
Chalcocite (%)	0.06	0.004	0.02	0.01	0.01	*	*	*
Molybdenite (%)	0.001	0	0.03	0.06	0.01	0.04	0.03	0.03
Pyrite (%)	0.06	0.04	0.11	0.06	0.15	0.6	1	0.4
Iron Oxides (%)	1.16	1.67	2.08	1.92	1.76	98.5†	97.6†	98.6†
Feldspar (%)	51.8	47.7	50.5	46.7	44.2	-	-	-
Quartz (%)	17.6	17.6	15.6	18.3	19	-	-	-
Micas (%)	11.4	10.5	9.91	11.3	11.1	-	-	-
Chlorite (%)	1.94	6.14	8.31	8.31	4.09	-	-	-
Garnet (%)	6.31	1.85	1.38	1.24	5.53	-	-	-
Calcite (%)	2.41	2.83	2.89	3.15	3.3	-	-	-
Others (%)	6.38	10.8	8.23	7.88	10.2	-	-	-

*No data
†Total gangue minerals

As shown in Figure 13-2 and Table 13-9, the 2008 mineralogical study on variability samples showed a significant variation in secondary copper minerals and pyrite contents. In some of the variability samples, the ratio of chalcopyrite to secondary copper minerals (bornite, chalcocite, and covellite) was low. On average, the ratio was approximately 2.7:1. For the 2012 test program samples, chalcopyrite comprised 78% of the copper minerals. These mineralogical determinations also revealed that the ratio of pyrite to copper minerals was low for all the samples.

Figure 13-2: Sulphide Mineral Ratio Variability Test Samples, 2008 (G&T)

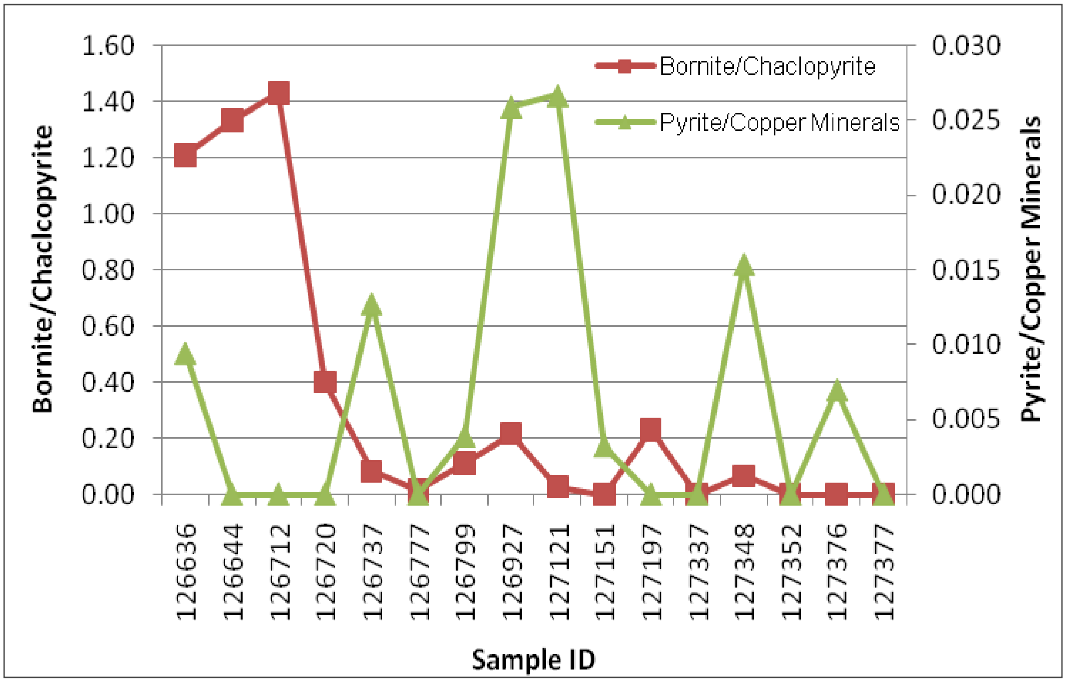


Table 13-9: Mineral Composition on Variability Samples, 2008 (G&T) Work

Sample	Estimate of Mineral Content (%)						
	Cp	Bn	Ch/Cv	Py	Ma	He	Gn
126636	0.36	0.75	0	0.02	0.45	0.16	98.6
126644	0	0.35	0.13	0	0	0.36	99.3
126649	0.50	0.05	0	0.02	0.32	0	99.3
126694	0.15	0.98	0.13	0.03	0.50	0.15	98.4
126702	0	1.24	0.15	0	0.32	0.12	98.4
126705	0.01	0.56	0	0.01	0.38	1.17	98.7
126710	0.03	0.69	0.04	0.01	0.50	0.50	98.7
126712	0	0.24	0	0.03	1.19	0.63	98.9
126720	0.10	0.18	0	0.02	6.06	0.93	96.4
126725	0.33	0.24	0	0	1.39	1.16	98.2
126737	1.55	0.14	0	0.34	1.42	0.43	97.1
126768	0.03	0.30	0	0.01	2.75	0.89	97.9
126777	0.97	0.05	0	0.37	1.18	0.65	97.7

table continues…

Sample	Estimate of Mineral Content (%)						
	Cp	Bn	Ch/Cv	Py	Ma	He	Gn
126785	0.93	0.27	0	0.01	0.31	1.13	98.1
126787	1.03	0.63	0	0	1.04	1.31	97.2
126799	1.04	0.23	0	0	0.33	1.28	98.0
126865	0.6	0	0	0.05	0.36	0	99.2
126876	0.47	0.34	0	0	0.24	0.09	99.0
126881	0.37	0.42	0.16	0	1.53	0.13	98.3
126882	0.17	0.69	0.02	0.01	1.10	0.29	98.5
126927	1.91	0.5	0	0.08	0.48	0	97.3
127121	0.43	0.03	0	0.37	0.46	0.11	98.9
127126	0.49	0	0	0	0.65	0.05	99.2
127145	3.37	0	0	0.05	0.24	0	96.5
127151	2.25	0	0	0.41	0.47	0.04	97.1
127197	1.94	0.58	0.01	0.01	0.09	0.01	97.4
127321	0.74	0	0	0.03	0.18	0.03	99.1
127337	0.95	0	0	0.53	1.91	0.16	97.5
127348	0.37	0.05	0	0.02	0.41	0.43	99.2
127352	0.95	0	0	0.57	0.18	0.14	98.3
127376	2.25	0	0	0.16	1.03	1.02	96.6
127377	2.59	0	0	0.14	1.27	0.14	96.6
127378	1.81	0	0	0.36	0.59	0.14	97.5
127573	0.24	0.43	0.03	0.01	1.61	1.18	98.0
Average	0.85	0.29	0.02	0.11	0.91	0.44	98.1

G&T also estimated the percent of mineral liberation at a grind size of 80% passing approximately 100 µm for the 2008 samples and at a grind size of 80% passing approximately 150 µm for the 2010 samples. The data are summarized in Table 13-10.

Table 13-10: Mineral Liberation Estimate (Two Dimensions), 2008/2010 (G&T)

Sample	Test Program							
	KM2291 (2010)					KM2050 (2008)		
	Composite					Paramount	Liard	West Breccia
	1	2	3	4	5	Zone	Zone	Zone
Grind Size 80% passing (µm)								
-	161	156	153	153	145	111	117	89
Liberation Rate (%)								
Chalcopyrite	-	-	-	-	-	59	70	50
Bornite	-	-	-	-	-	83	81	64
Copper Sulphides	39	45	51	47	46	61	72	53
Molybdenite	52	38	55	58	57	73	62	70
Pyrite	57	56	69	83	61	71	80	77
Gangues	96	97	95	97	96	98	97	98

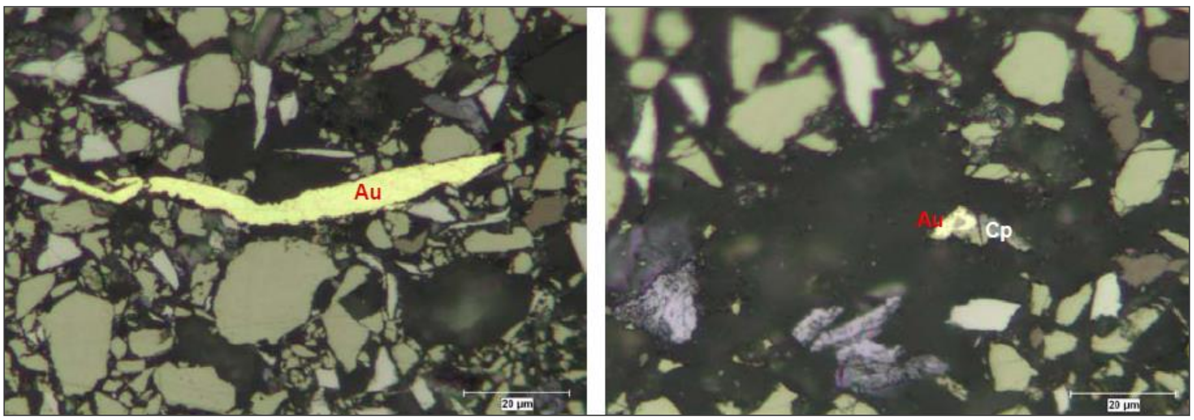
At the 80% passing target size of 100 microns used for the KM2050 test program, sulphide liberation levels ranged between 53 and 72%. For the KM2291 test program, liberation of copper minerals ranged from 39% to 51% at the grind size of 80% passing approximately 150 µm. More than 95% of the gangue minerals were in liberated form.

The KM3149 test program showed liberation patterns similar to those recorded for the mine plan composites included in the KM2291 test program. When measured in two dimensions, on average, approximately 47% of the copper sulphides were in liberation form at the nominal primary grind sizing of 80% passing 150 µm. Copper sulphide liberation for the KM3149 samples appeared to be related to the copper content. As estimated by G&T, copper sulphide liberation of approximately 50% to 55% is appropriate for good rougher performance. This primary grind size of 80% passing 150 µm should be sufficient given the low quantities of pyrite present. Most of the interlocked copper sulphides were associated with non-sulphide gangue in binary structures. These particles, on average, contained approximately 14% copper sulphides and should be recoverable by flotation processes.

The 2008 test work conducted a mineralogical study to assess gold occurrence of the copper flotation concentrate produced from the copper-molybdenum separation test work. The determination used Automated Digital Imaging System (ADIS) to scan the copper concentrate. The tests showed that approximately 80% of the gold observed was liberated. The remainder was locked with copper minerals or multi-phase particles. The average observed gold grain size was 11 µm in equivalent circle diameter. The largest gold observed occurred in multi-phases with an equivalent circle diameter of 18 µm. G&T indicated that most of the gold grains in the concentrate are too small for effective recovery by gravity concentration. The typical gold grains found by ADIS are shown in Figure 13-3.

No gold searches were conducted on the tailings samples, which include the cleaner flotation tailings and rougher flotation tailings. G&T suggested determining whether there was free gold lost into the two tailings.

Figure 13-3: Gold Occurrence in Copper Concentrate



13.4 Hardness Test Results

Each of the metallurgical testing programs included comminution characterization to determine sample hardness to crushing and grinding. The parameters determined include BWi, Rod Mill Work Index (RWi), Crushing Work Index (CWi), JK SimMet Drop Weight breakage parameters, SMC test breakage parameters, and high-pressure grinding rolls (HPGR) related hardness parameters. The 2015 test program characterized the materials more specifically by geometallurgical approach and represented the mineralization both spatially and geologically.

The historical test results are summarized in the following sections.

13.4.1 Mineral Sample Hardness Parameters – Crushing and Ball/Rod Mill Milling

Four test programs conducted crushability and grindability tests on various samples collected for different test programs. The test results are summarized in Table 13-11.

Table 13-11: Bond Crushing, Grinding, and Abrasion Indices

Test	Sample	BWi	RWi	CWi	Ai*	UCS§
Program	ID	(kWh/t)	(kWh/t)	(kWh/t)	(g)	MPa
PRA 0502002/2005	Composite #1	21.0	-	-	-	-
	Composite #2	22.1	-	-	-	-
PRA 0603303/2006	MLS	22.4	24.0	-	0.25	-
	WBZ	20.7	21.2	-	0.3	-
	WLZ	24.5	23.7	-	0.27	-
	LNZ	24.1	24.1	-	0.18	-
Hazen 10515/2007	Liard	21.9	20.1	8.93	0.1981	-
	Paramount	20.1	21.4	6.71	0.3796	-
	West Breccia	19.8	19.6	6.31	0.186	-
Hazen 10736/2008	Composite 1	22.6	21.6	11.86	0.1752	-
	Composite 2	24.0	22.5	11.31	0.1744	-
	Composite 3	20.5	19.3	8.59	0.5693	-
	Composite 4	24.4	21.8	7.96	0.3154	-
	Composite 5	23.8	22.2	10.2	0.1825	-
G&T 2012	Sample 1	18.8	17.6	-	0.1991	-
	Sample 2	20.7	18.9	-	0.18	-
	Sample 3	18.6	18.5	-	0.2201	-
	Sample 4	17.7	17.3	-	0.2137	-
G&T 2012	Sample 5	18.3	18.7	-	0.171	-
	Sample 6	18.7	18.2	-	0.1978	-

table continues…

Test Program	Sample ID	BWi (kWh/t)	RWi (kWh/t)	CWi (kWh/t)	Ai* (g)	UCS§ MPa
ALS 2015	All - Average	20.2	-	-	-	-
	All - Minimum	13.7	-	-	-	-
	All - Maximum	25.4	-	-	-	-
	Bx-L (Breccia)†	19.5‡	-	-	-	30
	Bx-N (Breccia)†	19.5‡	-	-	-	30
	Int-L (Intrusive)†	16.6‡	-	-	-	61
	Int-N (Intrusive)†	16.6‡	-	-	-	61
	Py-L (Porphyry)†	18.2‡	-	-	-	61
	Py-N (Porphyry)†	18.2‡	-	-	-	61
	Vic-L (Volcanic)†	22.4‡	-	-	-	70
	Vic-N (Volcanic)†	22.4‡	-	-	-	70
Average	-	21.2	20.8	8.65	0.25	55.5

*Ai = Bond abrasion index; averaged by test programs and excludes highest and lowest data.
†GeoMet unit; L = low RQD; N = normal RQD.
‡Average values.
§UCS = unconfined compressive strength.

The 2015 comminution test program delivered results which were lithologically distinct. The lithology types were further separated into low and normal designations based on the RQD information obtained during core logging. The low designation has an average RQD value of 29% and is defined spatially to approximately 150 m below surface where a distinct change in average RQD values was noted. The normal designation is attributed to the material below 150 m in depth and has an average RQD value of 69%.

The low energy impact work indices (CWi) are relatively low, ranging from 6.3 kWh/t to 11.9 kWh/t. The grindability test results (Bond BWi and Bond RWi) indicate that the mineral samples are high in grinding resistance to ball mill and rod mill grinding. There is a considerable variation in the grinding hardness among these samples tested. The BWi ranges from 13.7 kWh/t to 25.4 kWh/t, averaging 21.3 kWh/t. The RWi is slightly lower than the BWi, averaging 20.8 kWh/t. The average Ai is 0.25 g, fluctuating between 0.17 g and 0.57 g.

The comminution test results indicate that the mineral samples are classified as medium to very hard with respect to ball mill grinding. The BMWi value varies distinctly with lithology. The intrusive hosted mineralization returned the lowest values with an average of 16.6 kWh/t, indicating a medium to hard material in terms of the resistance to ball mill grinding while the volcanic hosted mineralization average BMWi was 22.4 kWh/t, indicating a very hard classification.

13.4.2 Mineral Sample Hardness Parameters and Simulations – SAG Mill Milling

13.4.2.1 Mineral Sample Hardness Parameters

Hazen conducted JK SimMet Drop Weight breakage tests and SMC tests on the samples collected for the 2007 and 2008 test programs. In 2012, G&T conducted further JK SimMet Drop Weight breakage tests and SMC tests on the 2012 test samples, and JKTech Pty Ltd. (JKTech) analyzed the generated data. In 2015, the mineral sample's resistance to SAG mill and ball mill grinding were investigated by using SMC test procedure. The test results are summarized in Table 13-12 to Table 13-17. The results indicate that the mineral samples are rated moderately hard to very hard for SAG mill milling. The key parameter (A x b) ranges from 27.4 to 44.7.

Figure 13-4 shows frequency distribution of the A x b values from the 2015 test program in the JKTech database.

Table 13-12: JK SimMet Drop-weight Breakage Parameters, 2007 (Hazen)

Parameter	Value		
	Liard Zone	Paramount Zone	West Breccia Zone
SG (by weighing in water and air)	2.73	2.69	2.74
JK SimMet Parameter			
A (maximum breakage)	47.6	54.3	49.6
b (relation between energy and impact breakage)	0.96	0.72	0.86
A x b (overall AG-SAG hardness)	44.7	39.1	42.7
Ta (abrasion parameter)	0.64	0.35	0.71

Note: AG = autogenous grinding

Table 13-13: SMC Breakage Parameters, 2007 (Hazen)

Parameter	Value		
	Liard Zone	Paramount Zone	West Breccia Zone
SG (by weighing in water and air)	2.73	2.69	2.74
SMCT Parameter			
A (maximum breakage)	62.9	63.5	63.7
b (relation between energy and impact breakage)	0.57	0.55	0.59
A x b (overall AG-SAG hardness)	35.8	34.9	37.6
SMC test (Drop-Weight index [DWi])	7.6	7.7	7.3

Table 13-14: JK SimMet Drop-weight Breakage Parameters, 2008 (Hazen)

Parameter	Value				
	Comp 1	Comp 2	Comp 3	Comp 4	Comp 5
SG (by weighing in water and air)	2.72	2.73	2.68	2.72	2.7
JK SimMet Parameter					
A (maximum breakage)	49.64	46.02	60.23	50.17	50.01
b (relation between energy and impact breakage)	0.81	0.86	0.58	0.73	0.82
A x b (overall AG-SAG hardness)	40	39.5	35.1	36.4	41
Ta (abrasion parameter)	0.41	0.45	0.35	0.45	0.43

Table 13-15: SMC Breakage Parameters, 2008 (Hazen)

Parameter	Value				
	Comp 1	Comp 2	Comp 3	Comp 4	Comp 5
SG (by weighing in water and air)	2.3	2.2	2.69	2.72	2.41
SMC Parameter					
A (maximum breakage)	83	80.7	76.8	77.3	67.9
b (relation between energy and impact breakage)	0.36	0.34	0.41	0.37	0.48
A x b (overall AG-SAG hardness)	29.9	27.4	31.5	28.6	32.6
SMC Test DWi	7.8	8	8.4	9.5	7.5
Mia (kWh/t)	25.7	27.5	23.5	25.5	23.8
Ta (abrasion parameter)	0.4	0.39	0.37	0.33	0.42

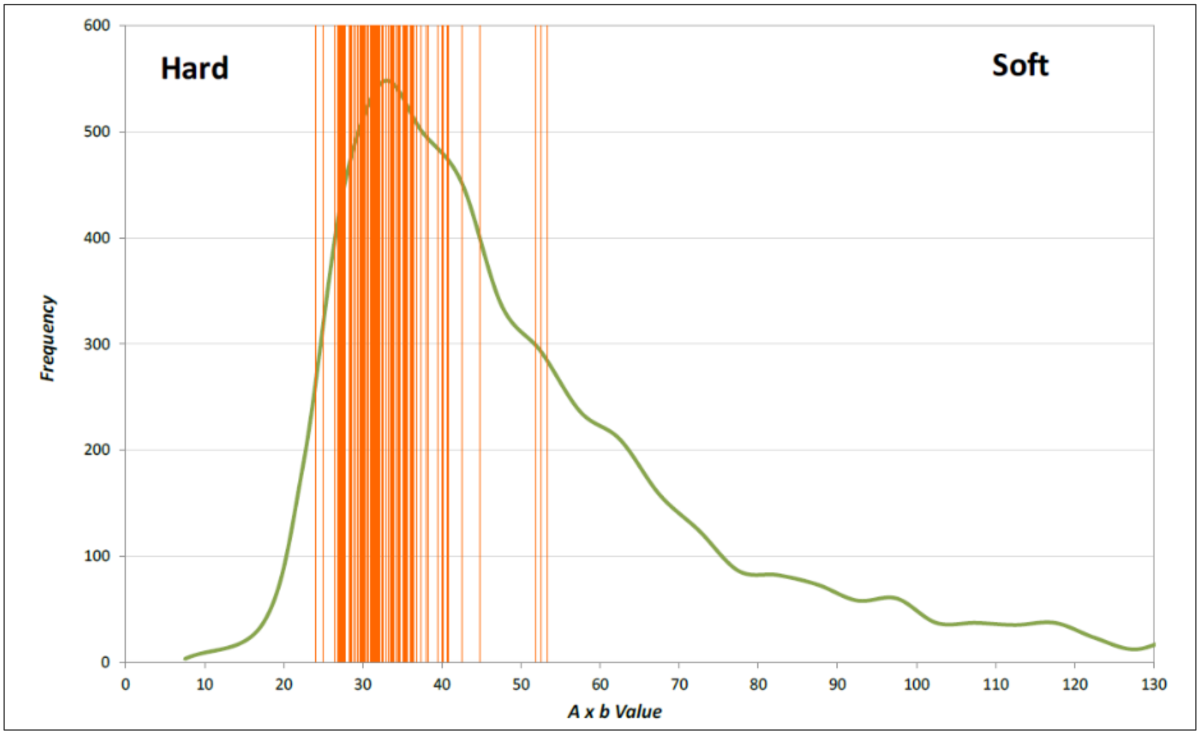
Table 13-16: SMC Breakage Parameters, 2012 (G&T/JKTech)

Parameter	Value					
	Comp 1	Comp 2	Comp 3	Comp 4	Comp 5	Comp 6
SG (by weighing in water and air)	2.68	2.71	2.76	2.68	2.71	2.68
SMC Parameter						
A (maximum breakage)	78.6	93	76.1	73.7	60.4	64.4
b (relation between energy and impact breakage)	0.46	0.3	0.42	0.51	0.68	0.56
A x b (overall AG-SAG hardness)	36.2	27.9	32	37.6	41.1	36.1
SMC test DWi (kWh/t)	7.47	9.61	8.57	7.11	6.61	7.38
Mia (kWh/t)	21.4	25.9	23.2	20.5	19.2	21.2
Ta (abrasion parameter)	0.35	0.27	0.3	0.36	0.39	0.35

Table 13-17: Average SMC Breakage Parameters, 2015 (ALS/JKTech)

Parameter	Value			
	Brc-L/Brc-N	Int-L/Int-N	Por-L/Por-N	Voc-L/Voc-N
SG	2.69	2.68	2.67	2.75
SMC Parameter				
A x b (overall AG-SAG hardness)	34.2	34.2	34.8	31.8
SMC test DWi (kWh/t)	8.09	8.05	7.87	8.77
Mia (kWh/t)	22.6	22.7	22.3	23.7
Mib (kWh/t)	22.5	18.5	20.6	26.5
Mic (kWh/t)	9.0	9.0	8.8	9.6
Ta (abrasion parameter)	0.33	0.33	0.34	0.30

Figure 13-4: Frequency Distribution of 2015 A x b Values in the JKTech Database



13.4.2.2 SABC Comminution Circuit Simulations

In 2007 and 2010, Contract Support Services conducted SAG mill / ball mill / pebble crusher circuit simulations for the Project. The 2010 simulations were based on the following conditions:

- 100% Liard sample and 85% Liard / 15% Paramount samples
- 120,000 t/d at 92% availability
- Grinding circuit product particle size of 80% passing 150 µm

The estimated power requirement for SAG mills was 18,200 kW per mill. A total of two 38 ft. SAG mills are required for the Project. The required power for the ball mills was estimated to be 58,800 kW, or 14,700 kW per mill (with a total of four ball mills). The pebble circulation load was projected to be approximately 17%.

In 2012, further SABC simulations were conducted using the data generated from the 2012 test work and the historical test programs. These simulations were conducted at 120,000 t/d at an availability of 92% and 130,000 t/d at an availability of 94%. The SAG mill feed particle sizes are 80% passing 120 mm for the 120,000 t/d throughput and 80% passing approximately 135 µm for the higher throughput. The simulation results are summarized below:

- Throughput 120,000 t/d: Two grinding circuits, each consisting of one 38 ft. by 20 ft. (effective grinding length [EGL]) SAG mill and two 26 ft. by 36 ft. (EGL) ball mills, are capable of grinding the mill feed to 80% passing 150 µm at a process rate of 120,000 t/d. The gross power draw is about 18.4 MW for each SAG mill at a feed particle size of 80% passing 120 mm and 14 MW for each ball mill. The transfer particle size is projected to be 80% passing 2,200 µm.

- Throughput 130,000 t/d: Two grinding circuits, each consisting of one 40 ft. by 21 ft. SAG mill and two 26 ft. by 44 ft. ball mills, are capable of grinding the mill feed to 80% passing 150 µm at a process rate of 130,000 t/d. The gross power draw is about 21.2 MW for each SAG mill at a feed particle size of 80% passing 130 mm and 15.5 MW for each ball mill. The transfer particle size is projected to be 80% passing 2,915 µm.

In 2015, further JK SimMet simulations were conducted based on the data produced from the GeoMet program. The primary grinding circuit arrangement used in the simulations is the same as the SABC circuit in the 2013 study, which consists of two grinding lines, each line equipped with one 40 ft. x 23 ft. (EGL) SAG mill, two 26 ft. x 44.5 ft. (EGL) ball mills, and one MP1000 equivalent pebble crusher. The feed particle sizes used for the simulations range from 80% passing 89.6 mm to 106.3 mm. The hydrocyclone overflow particle size was 80% passing 150 µm. The simulation results are summarized in Table 13-18.

Table 13-18: 2015 Primary Grinding Circuit Simulation Results

GeoMet Unit	SAG Mill Power	Ball Mill Power	Product Size	Throughput	
	kW	kW	80% Passing, µm	t/h*	kt/d
Brc-L	22.1	35.1	150	3,460	153
Brc-N	22.3	35.1	150	3,470	153
Int-L	22.0	35.1	150	3,190	141
Int-N	22.3	35.1	150	3,195	141
Py-L	21.8	35.1	150	3,005	133
Py-N	22.0	35.1	150	3,010	133
Vic-L	21.9	35.1	150	2,670	118
Vic-N	22.2	35.1	150	2,680	118

*At a mill availability of 92%

The simulation results show that the maximum dual-line circuit throughput is approximately 118 kt/d when processing 100% volcanic-type mineralization and approximately 153 kt/d when processing 100% breccia-type mineralization. The results indicate that when volcanic-type material is the dominant mill feed, the primary grinding circuit proposed for the previous study may not be able to achieve the designed throughput of 133 kt/d at a mill availability of 92%. It appears that based on the proposed grinding circuit, the ball mill capacity is the circuit limit.

Subsequently design work based on the SimSAG simulations was completed in 2019 which resulted in slight modifications to the ball mill circuit. In order to meet the increased design throughput target for the volcanic material type, an additional 2 MW of grinding power was added to each ball mill bringing installed power to 20 MW per mill.

Additionally, the ball mill sizes were increased slightly from 26 ft. x 44.5 ft. (EGL) to 26 ft. x 45 ft. (EGL). Based on the identified hardness of volcanic lithology material and the additional 11% power added to the ball mill motors, the SABC circuit is expected to be capable of processing the volcanic lithology at the target throughput of 133 ktpd.

Further circuit arrangement assessments should be conducted in conjunction with the potential mine plan, including further investigations into the effect of the feed/product particle size distributions on the grinding circuit capacity.

13.4.3 Mineral Sample Hardness Parameters – HPGR Crushing

Starting in June 2008, Polysius carried out HPGR tests on a sample consisting of large pieces of split cores with a net weight of 700 kg. The tests included bench scale LABWAL HPGR tests and REGRO semi-industrial HPGR tests. The material was crushed to two different sizes, less than 31 mm for the REGRO test, and less than 12.5 mm for the LABWAL test.

The tests found that the mineral sample was amenable to HPGR process and the results indicated the following:

- A pressure of at least 4 N/mm^2 was needed to reduce the feed particle size to, on average, 80% passing 8 mm (50% passing 2 mm and 20% passing 0.25 mm).
- The optimum press force was found to be a little higher than 4.2 N/mm^2; increasing pressure beyond this limit had minimal impact on size reduction.
- The moisture was not detrimental to the performance of the rolls.
- The specific throughputs were reasonably high, in a range of 210 ts/hm^3 to 250 ts/hm^3, under all the conditions tested.
- The average specific energy consumption was less than 2 kWh/t at 4 N/mm^2.
- The material after pressing did not form competent flakes, which implies that the HPGR product could be screened with a relatively high efficiency.
- The material was of low to medium abrasiveness, with an ATWAL wear index of 9.2 g/t at a moisture of 3%; wear life for HPGR was estimated at 8,000 h.

Results obtained from the LABWAL unit were reproduced on the semi-industrial scale REGRO unit.

More ATWAL tests are recommended if the HPGR technology will be used for the Project.

13.5 Metallurgical Test Results

13.5.1 Copper/Molybdenum Bulk Flotation

13.5.1.1 Process Condition Development Tests

Primary Grinding Size

Optimum primary grind size was tested through the test programs since 2004. Two different primary grind sizes were recommended.

The 2005 test program by PRA suggested a primary grind size of 80% passing approximately 140 µm for the samples generated from the historical dill core sample. The test results are summarized in Figure 13-5 and Figure 13-6.

Figure 13-5: Primary Grind Size Test Results – Copper, 2005 (PRA)

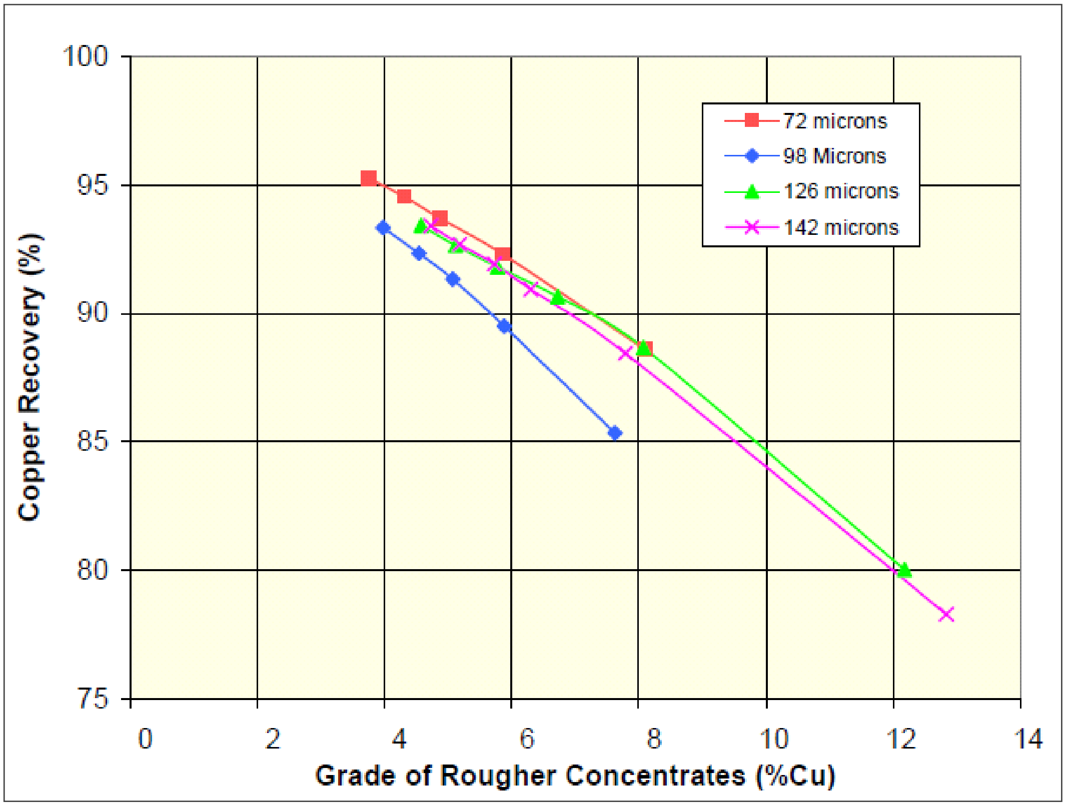
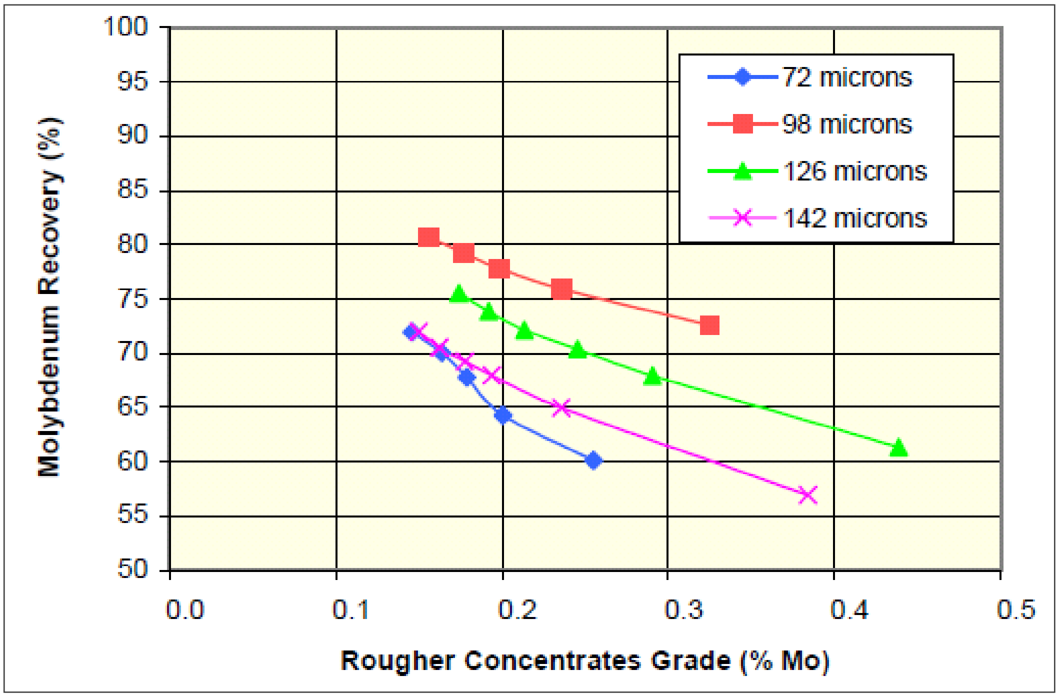


Figure 13-6: Primary Grind Size Test Results – Molybdenum, 2005 (PRA)



The 2006 test work on four composite samples showed that the effect of primary grind size on copper recovery was insignificant. PRA suggested the optimum primary grind size be 80% passing 130 µm. Test programs conducted in 2007 recommended a finer primary grind size at 80% passing 100 µm.

In the KM2050 test program (2008), G&T tested two different primary grind sizes on the three different zone composites. The test results, as shown in Table 13-19 and Table 13-20, indicate that the samples from the different mineralized zones responded similarly when tested. In the tested particle size range, it appears that primary grind size had insignificant impacts on the copper and molybdenum metallurgical performance at the primary grind sizes of 80% passing approximately 100 µm and 160 µm (190 µm for the Liard Zone sample). G&T indicated that the primary grind size could probably be designed at 80% passing 200 µm.

Table 13-19: Effect of Primary Grind Size on Metal Recovery, 2008 (G&T)

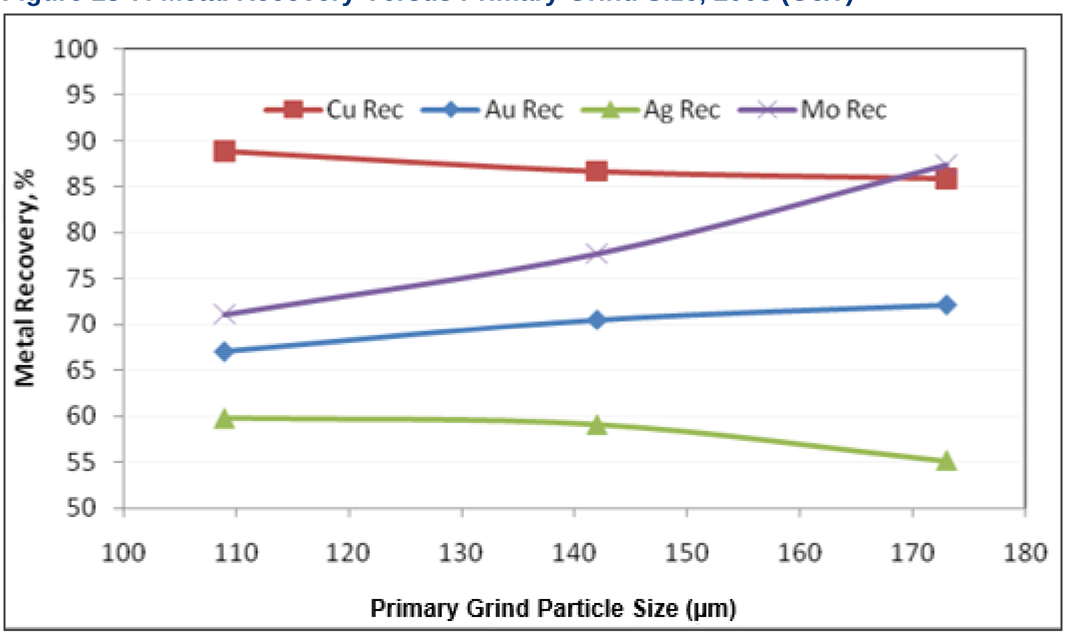
Sample ID	Test ID	Product	Mass	Distribution		Grinding Size 80% Passing
			(%)	Cu (%)	Mo (%)	(µm)
Paramount Zone Composite	KM2050-01	Rougher Concentrate	12.3	90.5	92.5	158
		Rougher Tailings	87.7	9.5	7.5	-
	KM2050-04	Rougher Concentrate	12.6	89.1	86.0	109
		Rougher Tailings	87.4	10.9	14.0	-
Liard Zone Composite	KM2050-02	Rougher Concentrate	10.2	92.2	95.2	190
		Rougher Tailings	89.8	7.8	4.8	-
	KM2050-05	Rougher Concentrate	15.9	94.6	95.0	102
		Rougher Tailings	84.1	5.4	5.0	-
West Breccia Zone Composite	KM2050-03	Rougher Concentrate	13.0	97.0	90.5	158
		Rougher Tailings	87.0	3.0	9.5	-
	KM2050-06	Rougher Concentrate	12.2	93.6	87.5	96
		Rougher Tailings	87.8	6.4	12.5	-

The later test work in 2008 further investigated the effect of the primary size on metallurgical performance of the Liard master composite using the locked cycle test procedure. The test results are summarized in Table 13-20 and illustrated in Figure 13-7.

Table 13-20: Effect of Primary Grind Size on Metal Recovery, 2008 (G&T)

Primary Grind Size 80% Passing (µm)	Product	Grade Cu (%)	Grade Mo (%)	Grade Au (g/t)	Recovery Mass (%)	Recovery Cu (%)	Recovery Mo (%)	Recovery Au (%)
109	Feed	0.32	0.013	2.1	100	100	100	100
	Bulk Concentrate	32.0	1.04	142	0.9	89	71	67
	Cleaner Tailings	0.14	0.020	3	9.3	4	14	9
	Rougher Tailings	0.03	0.002	0.7	89.8	7	15	24
142	Feed	0.33	0.014	2.1	100	100	100	100
	Bulk Concentrate	32.3	1.21	135	0.9	87	78	71
	Cleaner Tailings	0.15	0.018	2.0	9.9	4	11	9
	Rougher Tailings	0.03	0.002	0.7	89.2	9	11	20
173	Feed	0.33	0.012	2.4	100	100	100	100
	Bulk Concentrate	30.9	1.12	143	0.9	86	87	72
	Cleaner Tailings	0.15	0.009	3.0	9.1	4	7	9
	Rougher Tailings	0.04	0.001	0.9	90	10	6	19

Figure 13-7: Metal Recovery Versus Primary Grind Size, 2008 (G&T)



The locked cycle test results showed that copper recovery dropped from 89% to 86% and silver recovery from 60% to 55% when the primary grind size was increased from 80% passing 109 µm to 173 µm. However, molybdenum recovery increased significantly from 71% at the finest grind size to 87% at the coarsest grind size, and the gold recovery increased from 67% to 72%.

In the study completed in 2008, the designed primary grind size was 80% passing 150 µm.

Further tests were conducted in 2010 to verify the previous findings. The test conditions were similar to those used in the 2008 test program. Lime was used to adjust pH to approximately nine at rougher flotation and sodium ethyl xanthate (SEX) and fuel oil (FO) were used as collectors. The tested primary grind size ranged from 80% passing 137 µm to 180 µm. The confirmation tests were conducted on the Composite 1 sample. The open circuit test results are shown in Table 13-21.

Table 13-21: Effect of Primary Grind Size on Metal Recovery, 2010 (G&T)

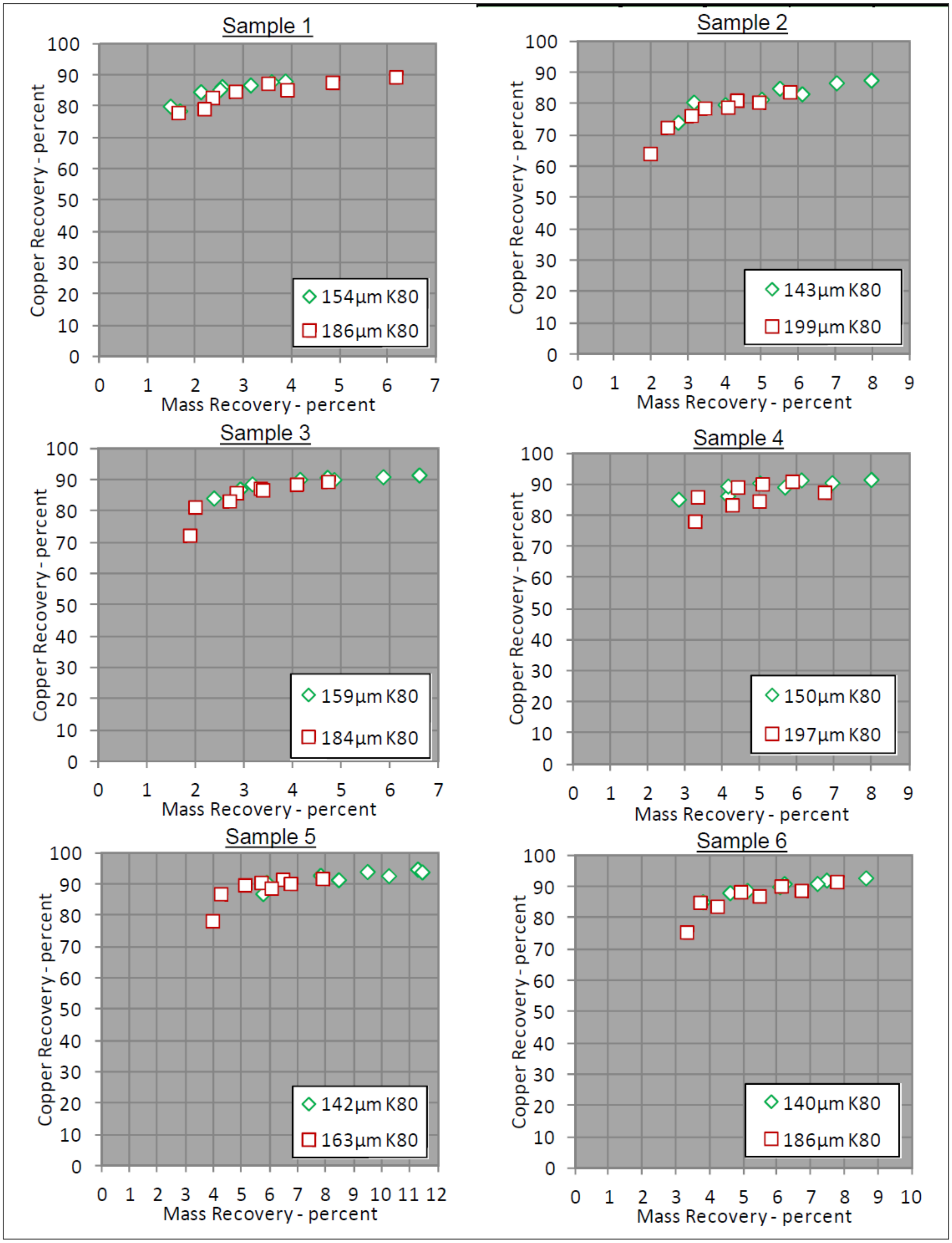
Primary Grind Size 80% Passing		Grade			Recovery			
		Cu	Mo	Au	Mass	Cu	Mo	Au
(µm)	Product	(%)	(%)	(g/)t	(%)	(%)	(%)	(%)
137†	Feed	0.39	0.014	0.36	100	100	100	100
	Rougher Concentrate	4.06	0.133	3.42	8.8	90.7	86.6	84.6
157*	Feed	0.39	0.012	0.29	100	100	100	100
	Rougher Concentrate	4.26	0.088	3.38	8.3	90.8	62.5	95.4
157†	Feed	0.38	0.012	0.31	100	100	100	100
	Rougher Concentrate	3.77	0.121	3.03	9.2	91.9	92.5	91.1
157†	Feed	0.38	0.013	0.27	100	100	100	100
	Rougher Concentrate	3.87	0.128	2.90	9.1	92.1	86.1	96.7
157‡	Feed	0.37	0.014	0.35	100	100	100	100
	Rougher Concentrate	6.90	0.228	6.03	4.7	87.1	76.1	80.1
180†	Feed	0.39	0.014	0.33	100	100	100	100
	Rougher Concentrate	3.92	0.13	3.03	9.1	90.7	86.6	83.4

*Collector dosage: 22 g/t FO, 24 g/t SEX
†Collector dosage: 60 g/t FO, 40 g/t SEX
‡Collector dosage: 30 g/t FO, 50 g/t SEX

The test results revealed that the effect of the primary grind size on the copper metallurgical performance was not significant. The test results also indicated that there were testing variations in metal recoveries, especially for gold and molybdenum. G&T projected that the tested samples could likely be processed at a coarser grind size of 80% passing up to 200 µm.

The 2012 tests used test conditions similar to those in the 2010 and 2008 test programs with a conventional reagent scheme. The alkalinity was elevated to about pH 9 using lime. FO was added in the grinding mill to collect molybdenum, while SEX was added as the copper collector. Four batch rougher flotation tests were completed on each of the six samples. On each sample, two tests were completed at a grind size of 80% passing 150 µm, and two were completed at a coarser sizing of approximately 80% passing 190 µm. The 2012 tests appear to show that the coarser primary grind sizing had only insignificant effects on copper, gold, and silver metallurgical performance; however, molybdenum performance appeared to deteriorate at the coarser sizing on all six samples. On average, molybdenum recovery dropped at the coarser primary grind sizing by approximately 7% at the same mass recovery. The test results are summarized in Figure 13-8.

Figure 13-8: Effect of Primary Grind Size on Metal Recovery, 2012 (G&T)



Regrind Size

The effect of regrind size on metallurgical performance has been investigated throughout the project history. The results indicate that to obtain the project target metallurgical performance, the rougher concentrate should be reground prior to cleaner flotation. Regrind targets of between 16 and 35 µm were tested using open-circuit flotation procedure. The further finer regrind size did not materially improved copper metallurgical performances although molybdenum performance was enhanced at the finer regrind size. Regrind size between 25 to 30 µm was considered be appropriate.

The 2005 PRA test program investigated the effect of regrind size on metallurgical performance. The regrind size ranged from 80% passing 80 µm (without regrind) to 100% passing 37 µm. The results are shown in Figure 13-9 for copper performance and in Figure 13-10 for molybdenum performance. The results appear to show that to obtain a high concentrate grade, the rougher concentrate should be reground prior to cleaner flotation. A finer regrinding appeared to benefit the molybdenum metallurgical performance.

Figure 13-9: Effect of Regrind Size on Copper Recovery, 2005 (PRA)

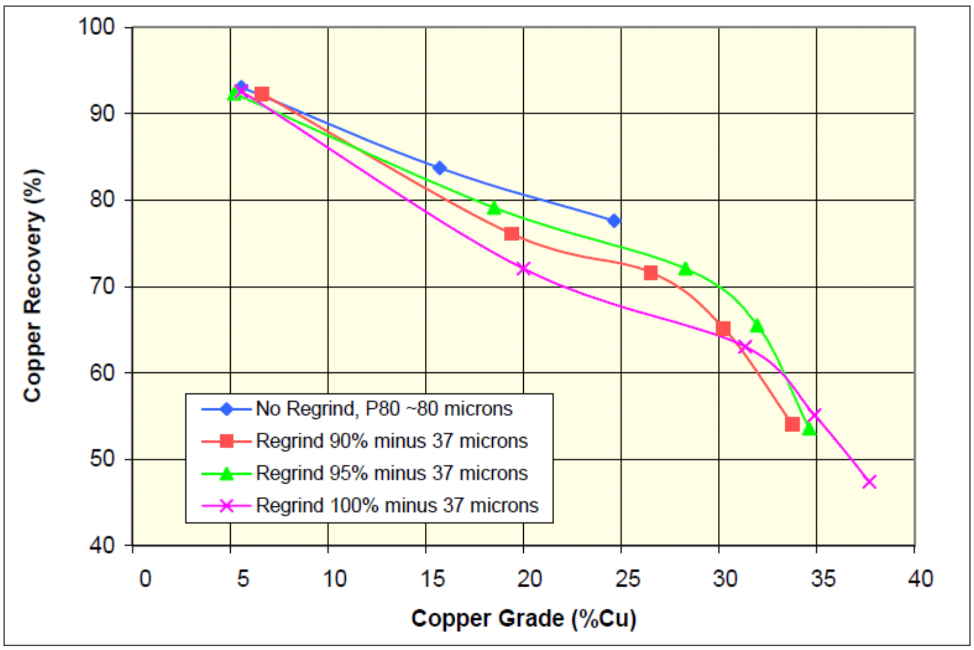
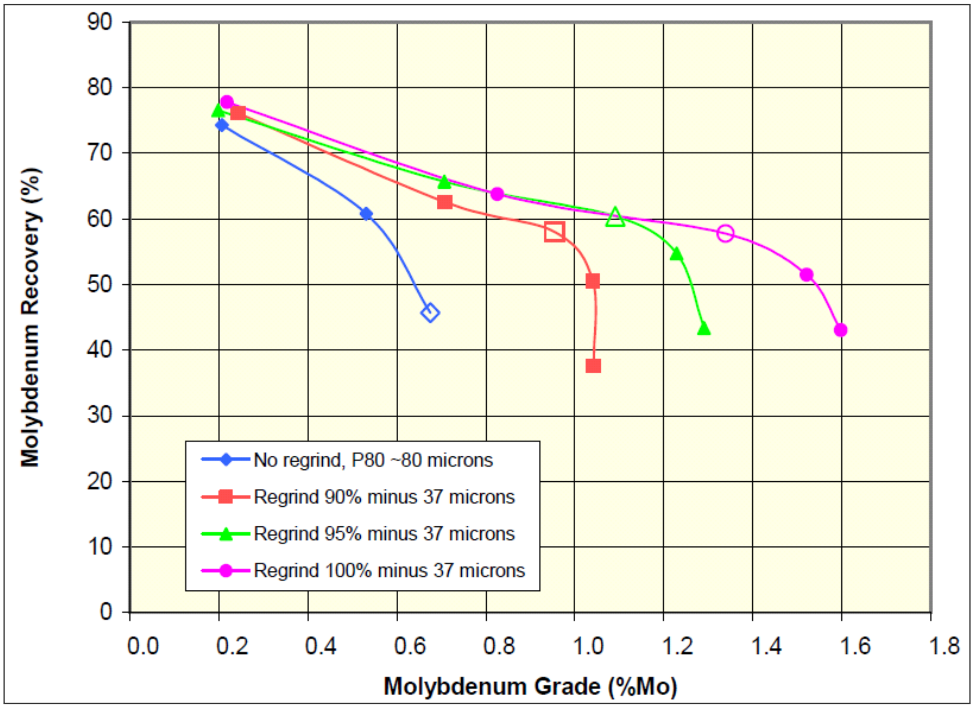


Figure 13-10: Effect of Regrind Size on Molybdenum Recovery, 2005 (PRA)



The 2008 test work by G&T further tested the effect of regrind size on the copper and molybdenum metallurgical performances. The test results are illustrated in Figure 13-11 and Figure 13-12.

The test results indicated that at the tested regrind sizes, which changed insignificantly in the tests, the copper metallurgical response was not affected by finer regrinding. However, it appears that the regrinding improved molybdenum recovery.

In 2010, further tests were conducted to verify the previous test results. The test results in Table 13-22 show that lower concentrate grades were produced at slightly coarser regrind sizes. The effects of regrind size on the copper recoveries were not significant.

Figure 13-11: Effect of Regrind Size on Copper Metallurgical Performance, 2008 (G&T)

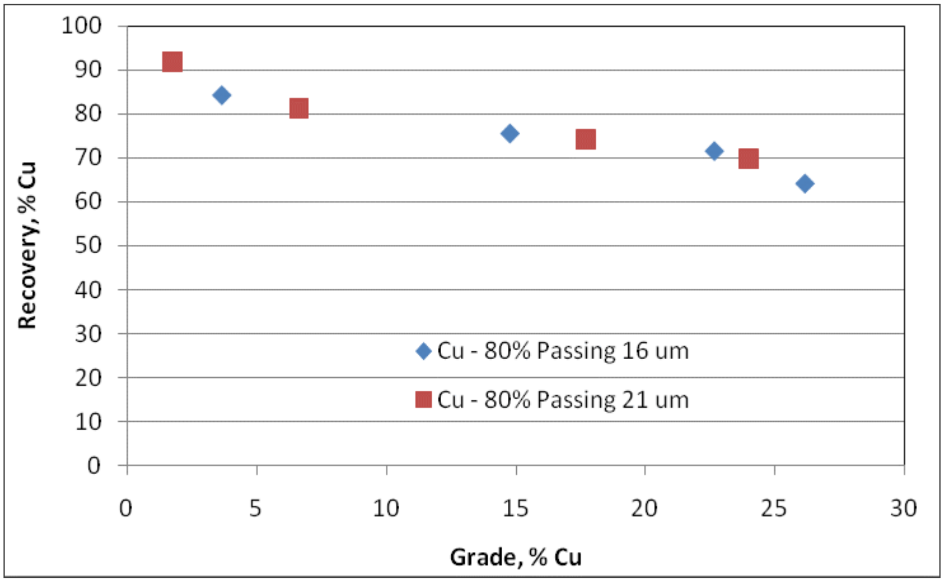


Figure 13-12: Effect of Regrind Size on Molybdenum Metallurgical Performance, 2008 (G&T)

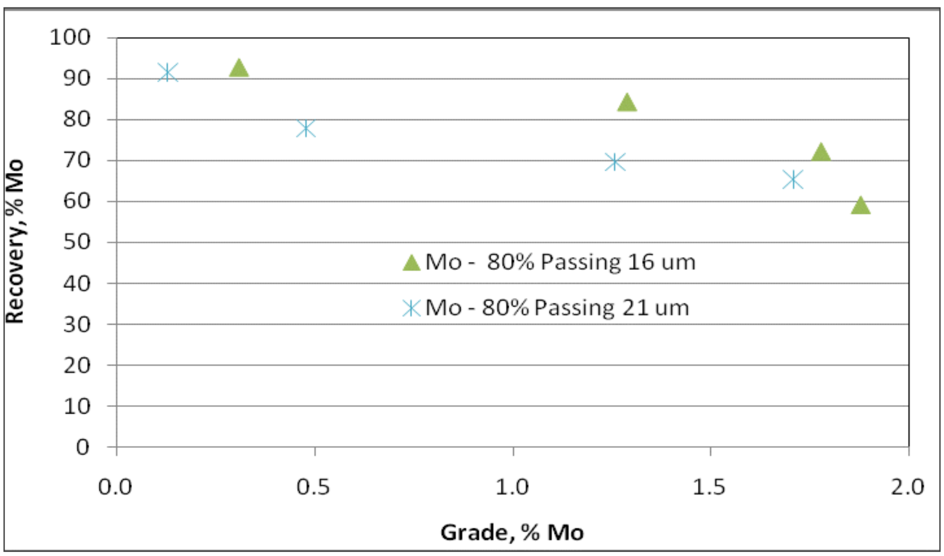


Table 13-22: Effect of Regrind Size on Metal Recovery, 2010 (G&T)

Sample	Grind		Assay			Recovery			
			Cu	Mo	Au	Mass	Cu	Mo	Au
(Test ID)	Size*	Product	(%)	(%)	(g/t)	(%)	(%)	(%)	(%)
Composite 1									
KM2291-23	157/25	Feed	0.38	0.012	0.31	100	100	100	100
		Third Cleaner Concentrate	29.8	0.92	22.0	1.1	83.5	80.7	76.1
		Rougher Concentrate	3.77	0.121	3.03	9.2	91.9	92.5	91.1
KM2291-29	157/31	Feed	0.38	0.013	0.27	100	100	100	100
		Third Cleaner Concentrate	27.4	0.866	18.3	1.2	83.8	75.5	78.6
		Rougher Concentrate	3.87	0.128	2.9	9.1	92.1	86.1	96.7
Composite 2									
KM2291-25	154/20	Feed	0.34	0.019	0.25	100	100	100	100
		Third Cleaner Concentrate	32.7	1.58	21.6	0.9	84.1	73.4	74.4
		Rougher Concentrate	3.55	0.194	2.58	8.8	91.7	90.3	89.2
KM2291-30	154/24	Feed	0.34	0.019	0.25	100	100	100	100
		Third Cleaner Concentrate	30.0	1.43	23.5	1	85.1	73.5	90.0
		Rougher Concentrate	3.21	0.174	2.49	9.8	92.1	90.5	96.5
Composite 3									
KM2291-26	153/22	Feed	0.47	0.034	0.49	100	100	100	100
		Third Cleaner Concentrate	33.1	2.31	28.7	1.2	87.0	82.4	72.5
		Rougher Concentrate	3.76	0.266	3.65	11.6	93.6	89.7	87.3
KM2291-31	153/28	Feed	0.45	0.03	0.49	100	100	100	100
		Third Cleaner Concentrate	30.2	1.97	23.1	1.3.0	85.3	85.3	60.2
		Rougher Concentrate	3.48	0.227	3.42	12.1	92.5	92.6	84.0

table continues…

Sample (Test ID)	Grind Size*	Product	Assay Cu (%)	Assay Mo (%)	Assay Au (g/t)	Recovery Mass (%)	Recovery Cu (%)	Recovery Mo (%)	Recovery Au (%)
Composite 4									
KM2291-27	153/21	Feed	0.33	0.029	0.23	100	100	100	100
		Third Cleaner Concentrate	29	2.0	16.8	0.9	82.4	65.9	69.0
		Rougher Concentrate	2.63	0.216	1.73	11.6	91.7	87.6	87.3
KM2291-32	153/24	Feed	0.34	0.026	0.24	100	100	100	100
		Third Cleaner Concentrate	25.1	1.75	13.4	1.1	82.3	75.0	60.9
		Rougher Concentrate	2.35	0.166	1.65	13.2	92.0	84.8	89.3
Composite 5									
KM2291-28	144/24	Feed	0.28	0.017	0.21	100	100	100	100
		Third Cleaner Concentrate	31.1	1.72	19.1	0.7	82.6	74.7	66.9
		Rougher Concentrate	2.19	0.125	1.53	11.7	90.9	84.6	83.5
KM2291-33	144/30	Feed	0.29	0.018	0.21	100	100	100	100
		Third Cleaner Concentrate	26.0	1.53	16.8	0.9	82.2	77.9	72.1
		Rougher Concentrate	2.11	0.127	1.44	12.4	90.6	87.8	83.6

*Primary grind size / regrind size: 80% passing, microns

Reagent Regime

In general, the samples tested responded well to a conventional and simple reagent regime, according to the test results with various copper and molybdenum mineral collectors. The collectors used in bulk copper-molybdenum flotation included potassium ethyl xanthate (PEX), SEX, and potassium amyl xanthate (PAX). The molybdenum collectors tested were FO and Cytec A3302. The frothers tested included pine oil, DF250, methyl isobutyl carbinol (MIBC), and F549. The optimized reagents used for the 2010 locked cycle tests were SEX and FO as collectors and MIBC as a frother.

PRA test work also used sodium sulphide and sodium phosphate as regulators in an effort to improve metal recovery or concentrate grade. However, the effect of these reagents on the metallurgical performances was not noteworthy.

Bulk Flotation pH

PRA investigated the effect of pulp pH on the copper and molybdenum bulk rougher flotation. The test results are shown in Figure 13-13 and Figure 13-14.

Figure 13-13: Effect of pH on Bulk Rougher Flotation – Copper, 2005 (PRA)

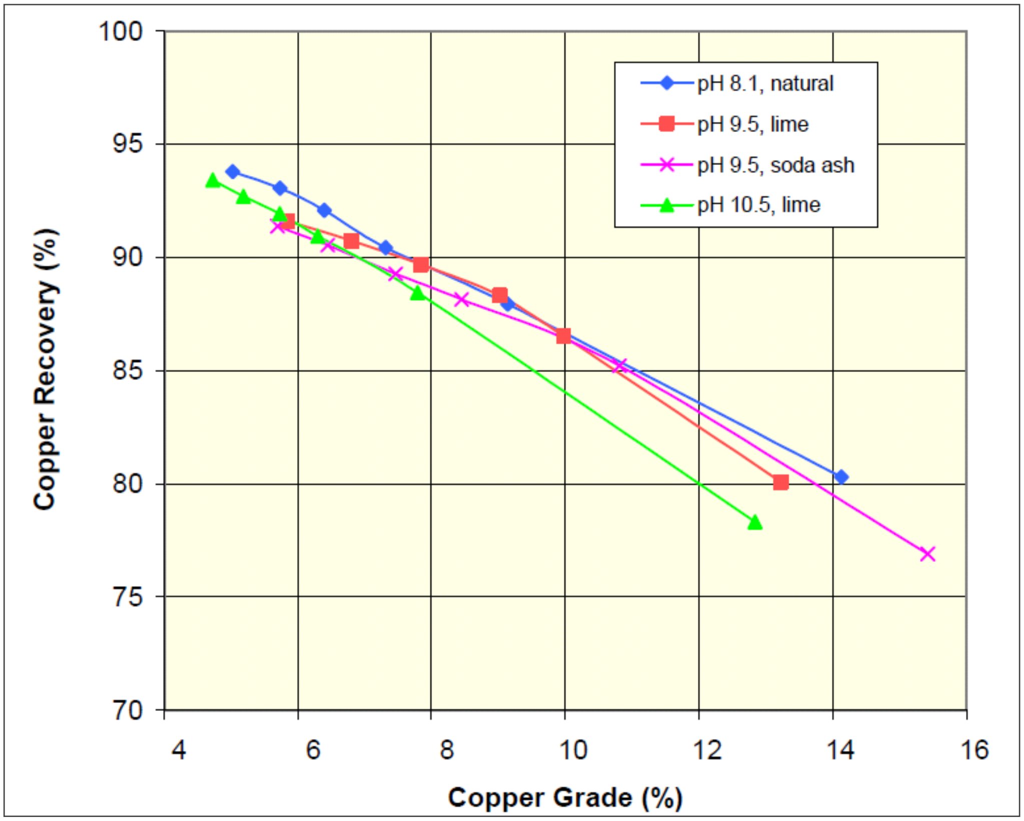
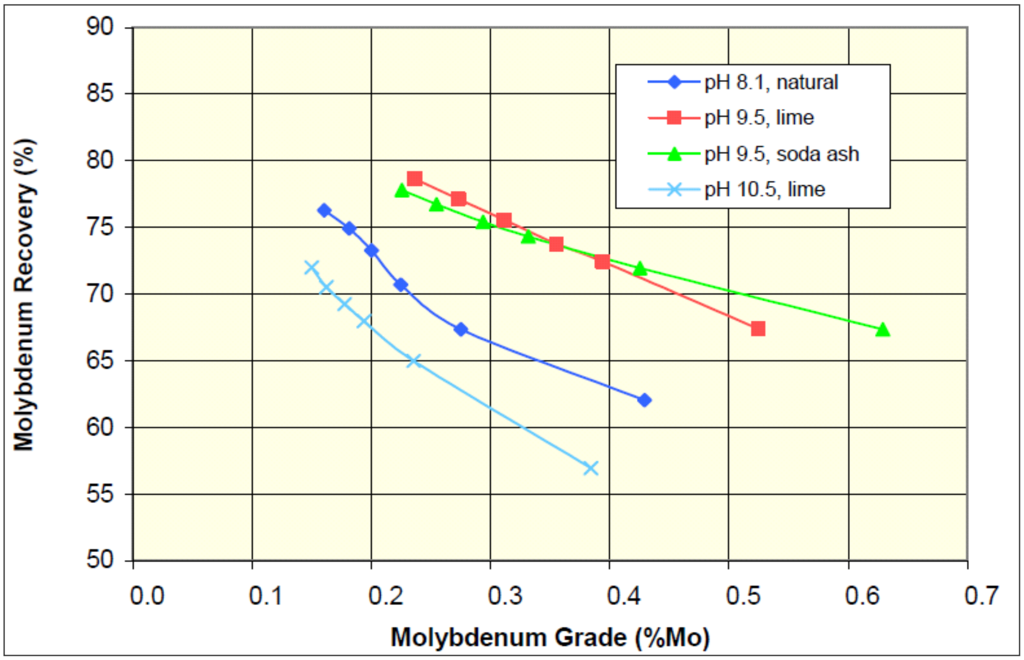


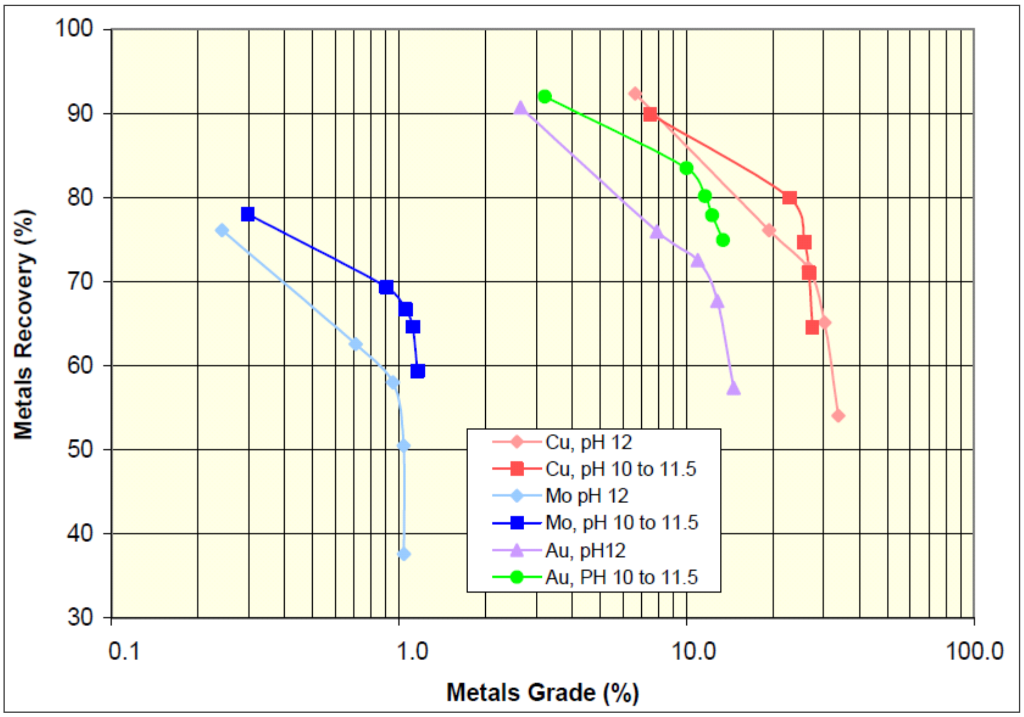
Figure 13-14: Effect of pH on Bulk Rougher Flotation – Molybdenum, 2005 (PRA)



The test results appear to indicate that pH did not significantly influence the copper rougher flotation at the pH range of between 8.1 and 9.5. However, copper flotation was slightly depressed at pH 10.5. Molybdenum minerals exhibited slightly different metallurgical responses, with the best metallurgical performance recorded at pH 9.5. Both a lower pH (8.1) and a higher pH (10.5) generated poorer metallurgical performances. Further bulk rougher flotation was performed at pH 9.0 or 9.5, which was considered the optimum pH.

The test program also investigated the effect of pulp pH on the copper-molybdenum bulk cleaner flotation. The test results in Figure 13-15 indicate that metal recoveries declined when the cleaner flotation was conducted at pH 12.

Figure 13-15: Effect of pH on Bulk Cleaner Flotation – Molybdenum, 2005 (PRA)



Variability Test Results

Two variability testing programs were conducted by PRA and G&T. The variability tests used the optimum test conditions developed.

The variability test results show a significant variation in the metallurgical performances between the individual drill core interval samples. The variation can be traced to significant differences in mineralogy between the samples. The variations in the results reflect a very poor correlation in the metallurgical performance among the samples tested. However, the master composites show much less variation in the metallurgical performance. Figure 13-16 to Figure 13-20 are focused on a review of the test results obtained by the more recent testing programs by G&T.

Figure 13-16: Copper Recovery vs. Copper Head Grade

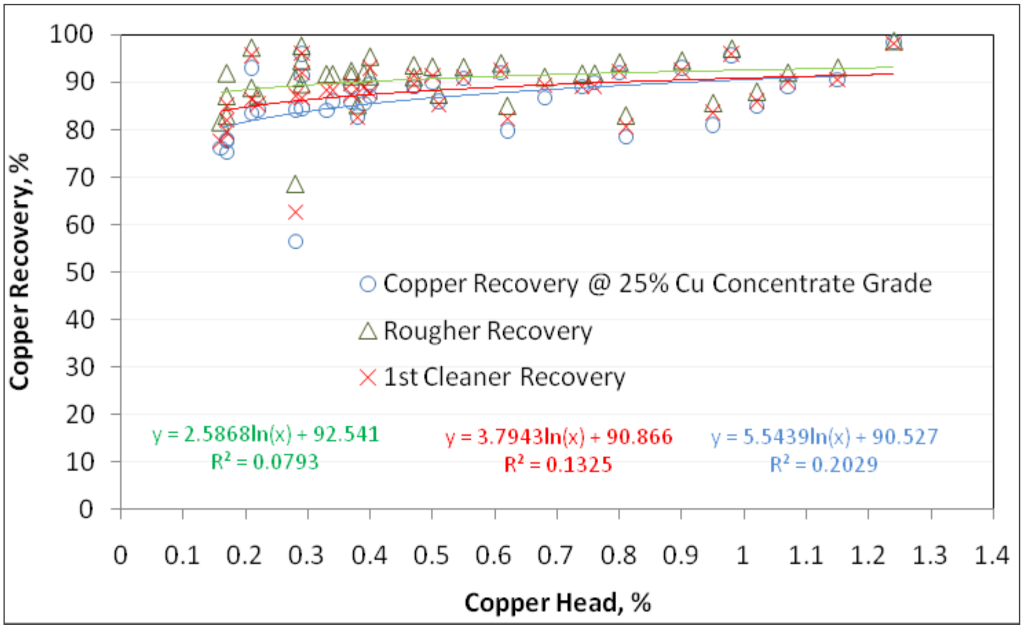


Figure 13-17: Gold Recovery vs. Gold Head Grade

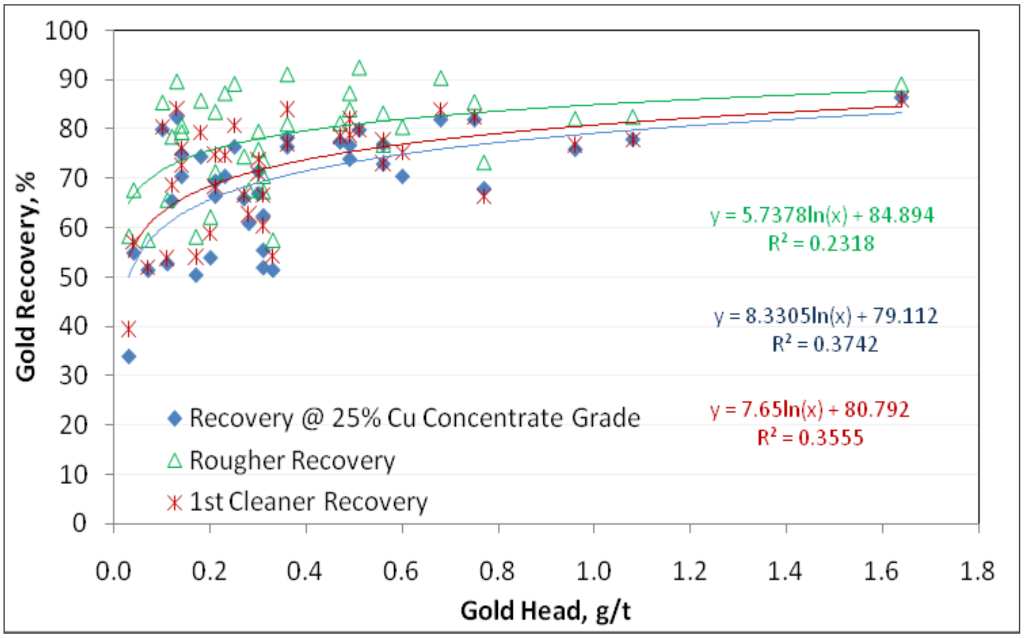


Figure 13-18: Silver Recovery vs. Silver Head Grade

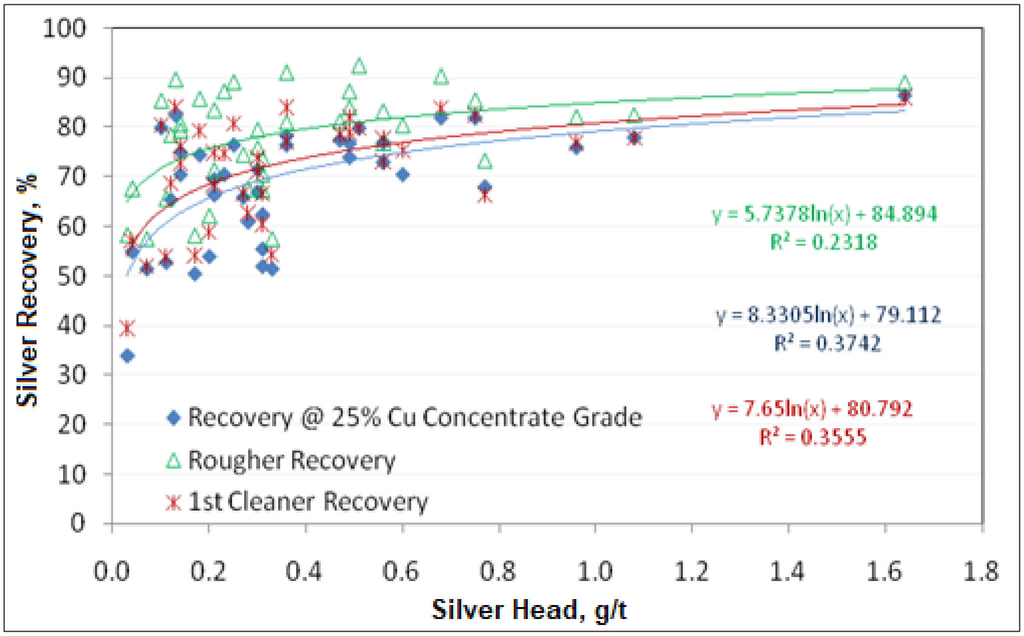


Figure 13-19: Molybdenum Recovery to First Cleaner Concentrate vs. Molybdenum Head Grade

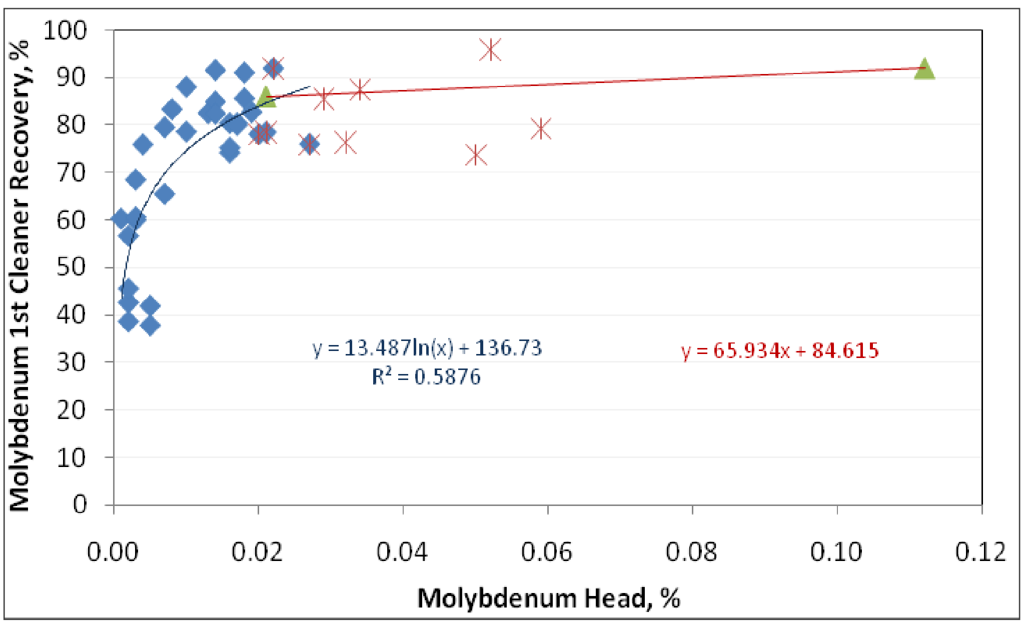
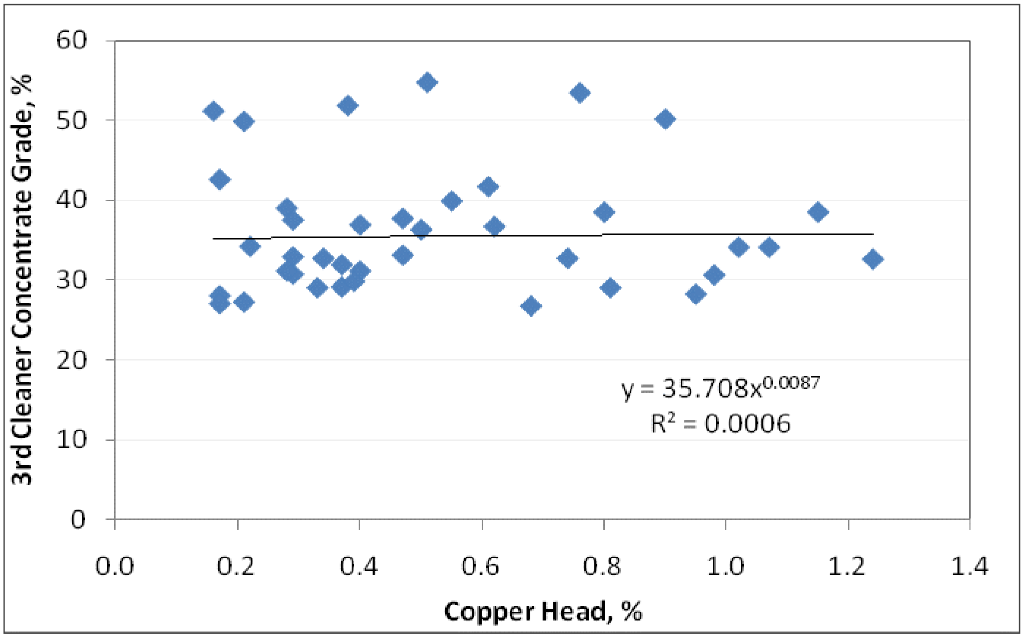


Figure 13-20: Copper Concentrate Grade vs. Copper Head Grade



The variability test results show a significant range in the metallurgical performances between the individual drill core interval samples. This observation is based on the concentrate grades and recoveries which were achieved in open circuit testing under similar flotation conditions. Locked cycle testing was not completed on the variability samples although batch cleaner stage test work was conducted to inform overall metallurgical performances. The copper grades of the third cleaner concentrates (open bench tests) varied from 26% to 55%, averaging at 36% which are indications that secondary copper minerals such as bornite may be present in high levels.

The test results indicate that the metal recoveries increase with an increase in head grades. There are more significant fluctuations in gold and silver recoveries in comparison with copper and molybdenum recoveries. It appears that the correlation of gold, silver, and molybdenum recoveries with copper recovery is poorer than the correlation of gold, silver, and molybdenum recoveries with its own head grade. In general, the average metallurgical performance is expected to be similar to the master composite samples.

Locked Cycle Tests – Bulk Flotation

Six test programs conducted locked cycle tests to evaluate the flotation metallurgical performance of various composite samples since 2005. A summary of the bulk flotation locked cycle test results is presented in Table 13-23. This summary excludes the 2005 test results because the flowsheet used was significantly different from the optimum flowsheet that was used in more recent test programs. Duplicate locked cycle tests were done on the 2008 Master Composite sample (KM2136) at three different primary grind sizes, namely 80% passing approximately 109 µm, 142 µm, and 173 µm.

Table 13-23: Bulk Flotation Locked Cycle Test Results

Samples	Primary Grind (P80, µm)	Regrind (P80, µm)	Head				Concentrate Grade Copper	Recovery – To Bulk Concentrate				
			Cu	Mo	Ag	Au	Cu	Cu	Mo	Ag	Au	Mass
			(%)	(%)	(g/t)	(g/t)	(%)	(%)	(%)	(%)	(%)	(%)*
Master Composite 1 (2012)†	155	20	0.42	0.026	3.5	0.27	31.7	89.0	63.9	50.5	72.6	5.4
Sample 1 (2012)	154	21	0.18	0.01	3.0	0.18	27.6	82.6	60.2	23.9	56.1	6.0
Sample 2 (2012)	143	23	0.17	0.009	1.0	0.09	26.2	78.5	33.8	18.9	31.9	6.7
Sample 3 (2012)	159	28	0.32	0.0014	2.0	0.2	30.1	90.2	68.2	48.3	58.8	7.5
Sample 4 (2012)	150	26	0.38	0.024	2.0.	0.11	28.8	86.3	42.7	64.5	76.9	7.8
Sample 5 (2012)	142	23	0.79	0.035	5.0	0.5	32.4	92.0	54.4	61.8	76.0	12.7
Sample 6 (2012)	140	24	0.7	0.053	4.0	0.59	34.8	90.2	65.7	71.7	74.1	9.4
Composite 1 (2010)	180	23	0.37	0.012	3.0	0.33	30.1	86.5	75.6	57.0	79.0	10.5
Composite 2 (2010)	154	24	0.33	0.018	2.0	0.32	31.0	83.2	67.2	50.8	63.9	12.3
Composite 3 (2010)	153	23	0.45	0.03	3.0	0.51	34.1	85.6	83.0	60.7	77.6	10.6
Composite 4 (2010)	153	25	0.33	0.021	2.0	0.2	27.2	84.3	86.2	63.4	83.8	12.3
Composite 5 (2010)	144	27	0.27	0.018	2.0	0.17	28.3	81.2	74.8	62.8	82.2	10.7
Paramount Zone Composite (2008)	109	14	0.27	0.016	3.2	0.22	26.2	77.8	75.2	48.2	70.3	-
Liard Zone Composite (2008)	102	12	0.30	0.015	2.3	0.23	29.8	84.7	86.5	73.6	76.2	-
West Breccia Zone Composite (2008)	96	17	0.69	0.026	6.2	0.38	29.9	86.9	61.4	81.1	85.1	-
Master Composite 1 (2008)‡	141	20	0.33	0.013	2.3	0.25	31.7	87.4	76.2	56.5	69.8	10.3
Master Composite 1 (2008)	109	21	0.32	0.014	2.0	0.26	32.7	88.3	66.6	59.9	58.9	7.9

table continues…

Samples	Primary Grind (P80, µm)	Regrind (P80, µm)	Head				Concentrate Grade Copper	Recovery – To Bulk Concentrate				
			Cu	Mo	Ag	Au	Cu	Cu	Mo	Ag	Au	Mass
			(%)	(%)	(g/t)	(g/t)	(%)	(%)	(%)	(%)	(%)	(%)*
Master Composite 1 (2008)	109	19	0.33	0.012	2.0	0.25	31.4	89.5	75.7	59.7	75.2	12.2
Master Composite 1 (2008)	142	19	0.33	0.014	2.0	0.25	33.0	86.4	65.3	48.6	68.7	8.4
Master Composite 1 (2008)	142	21	0.33	0.014	2.0	0.29	31.6	87.0	90.2	69.6	72.4	13.2
Master Composite 1 (2008)	173	18	0.33	0.011	3.0	0.23	31.2	86.4	84.3	51.2	71.1	7.9
Master Composite 1 (2008)	173	21	0.34	0.013	2.0	0.25	30.6	85.4	90.5	59.1	73.3	12.1
Master (2007)	101	20	0.39	0.017	2.1	0.29	25.4	85.4	76.2	68.5	81.6	25.1
Liard Zone (2007)	109	20	0.39	0.009	2.44	0.37	25.4	87.9	87.9	67.0	84.2	12.4
Paramount Zone (2007)	101	25	0.45	0.033	2.6	0.28	26.5	93.7	88.4	52.6	74.5	25.9
West Breccia Zone (2007)	107	20	0.41	0.02	2.34	0.28	27	93.2	83.4	42.8	86.0	27.6
MLS (2006)	139	19	0.45	0.018	2.0	0.28	25.1	83.9	81.4	61.2	84.3	14.1
WBZ (2006)	142	Unknown	0.41	0.027	2.3	0.2	22.1	82.9	80.0	70.2	72.6	14.7
WLZ (2006)	168	15	0.36	0.017	1.7	0.31	34.2	70.0	59.9	63.0	71.9	5.4
LNZ (2006)	157	14	0.34	0.024	1.96	0.31	29.8	80.6	80.0	66.6	76.6	9.7
PIT Composite (2005)	133	16	0.39	0.022	2.2	0.25	26.4	86.7	81.3	60.2	80	18.6

*Rougher flotation
†Two locked cycle test results average
‡Six locked cycle test results average

The bulk flotation locked cycle test results showed that the mineral samples tested responded well to a simple, conventional process: bulk sulphide flotation followed by fine regrinding on the bulk concentrate and three stages of cleaner flotation.

In general, the master composite samples from the Main (Liard) zone showed similar metallurgical responses between the various testing programs. The samples from the Paramount Zone and the West Breccia Zone yielded different metallurgical performances.

However, the overall average test results for the Paramount Zone and West Breccia Zone samples are similar to the data obtained from the Liard mineralization. The 2012 samples that covered a significant portion of the Paramount zone (including the West Breccia area) and different grade classes, as well as lithology and alteration, showed similar metallurgical performances as the Liard mineralization. Further test work is suggested for a better understanding of the metallurgical performances of the Paramount mineralization and the West Breccia mineralization.

At an average primary grind size of 80% passing 151 µm, G&T test work data shows that on average, 86.2% of the copper was recovered from the head sample containing approximately 0.37% Cu. The other associated metal recoveries were 73.3% for gold, 55.7% for silver, and 71.9% for molybdenum. The average sample feed grades were approximately 0.27 g/t Au, 2.7 g/t Ag, and 0.019% Mo. The concentrate produced contained 30.9% Cu. The average data from G&T and PRA show that at the primary grind size of 80% passing 146 µm, 86.7% of the copper reported to the copper concentrate at a grade of 29.9% Cu. The gold, silver, and molybdenum recoveries to the concentrate were 74.7%, 56.9%, and 73.7%, respectively. Apart from lower-grade concentrates produced by PRA, the test results from the two laboratories were very comparable.

The test results from the KM2136 testing program on the master composite sample presented in Figure 13-7 show that a finer primary grind size may be beneficial for copper and silver recovery, but detrimental for gold and molybdenum. It is not clear why the gold recovery decreased with a finer primary grind. Due to the very high grinding resistance, it was decided that a primary grind size of 80% passing 150 µm would be used for this study, as it was in the 2008 study.

The 2012 test program also conducted locked cycle tests to investigate the metallurgical responses of the composite sample and various individual samples mainly collected from the lower portion of the Paramount Zone. In general, the samples produced the results similar to those from the previous test programs. However, Sample 1 and Sample 2 had poorer performances than the other four samples, likely due to their lower copper contents in the head samples.

In general, the average metallurgical performances from the variability tests agree with the locked cycle test results.

Pilot Plant Tests

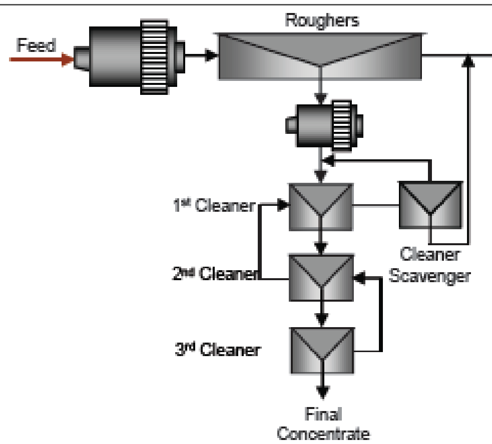
During October and November 2008, G&T conducted a pilot plant test campaign using 8,000 drill core interval samples from the Liard Zone. As indicated by G&T, the main objective of the pilot plant test program was to produce a bulk copper-molybdenum concentrate sample for copper-molybdenum separation tests and smelter evaluation. .

The head samples were crushed to 100% passing ⅜ in and stored in five bins. The head assays on the cut samples from the five bins are shown in Table 13-24.

On average, the copper content of the sample was 0.32% and molybdenum grade was 0.016%. Approximately 11% of the total copper was present in non-sulphide form. It appears that the oxidation degree of the pilot plant test sample was higher than that of the sample used in the 2008 test program (KM2136) that contained approximately 7% of non-sulphide copper.

Twelve pilot plant campaigns were conducted, including two grind size calibration trials. The flowsheet used was based on the previous locked cycle tests and is shown in Figure 13-21. The reagents used were lime as pH regulator and PEX and FO as collectors. The pH ranged from 7.8 to 9.6 for rougher flotation and from 10.5 to 11.1 for cleaner flotation.

Figure 13-21: Pilot Plant Test Flowsheet, 2008 (G&T)



Reagent Summary

Composite	Stage	K80 µm	pH	Reagents - g/t Lime	Fuel Oil	PEX
Pilot Plant Composite	Grind	63-176	7.8-9.6	90-190	24-38	-
	Roughers	-	-	-	-	26-52
	Regrind	15-51	-	-	0-1	-
	Cleaners	-	10.5-11.1	-	-	0-4
Pilot Plant Composite in Laboratory	Grind	132	8.3	300	20	-
	Roughers	-	9.0	150	-	65
	Regrind	21	110.0	500	10	-
	Cleaners	-	110.0	√	14	8

The test results are summarized in Table 13-24.

Table 13-24: Pilot Plant Test Results – Copper-Molybdenum Bulk Concentrate, 2008 (G&T)

Test	Grind Size*	Grade				Recovery				
		Cu	Mo	Ag	Au	Mass	Cu	Mo	Ag	Au
	(µm)	(%)	(%)	(g/t)	(g/t)	(%)	(%)	(%)	(%)	(%)
P3	174	22.9	1.06	105	13.4	0.7	50.0	43.5	25.6	58.0
P4	176	23.3	1.44	102	15.1	1.1	70.3	64.6	20.9	76.3
P5	165	21.1	1.15	99	12.9	1.2	74.0	66.1	21.2	99.8
P6	137	28.8	1.71	125	18.9	0.7	63.5	63.1	21.8	55.8
P7	139	27.0	1.54	115	19.2	1.2	81.8	77.9	39.7	76.1
P8	70	27.6	1.19	121	20.2	0.9	78.2	54.5	22.2	70.9
P9	146	28.1	1.58	128	19.5	0.9	73.9	60.1	53.6	83.0

table continues…

Test	Grind Size*	Grade				Recovery				
		Cu	Mo	Ag	Au	Mass	Cu	Mo	Ag	Au
	(µm)	(%)	(%)	(g/t)	(g/t)	(%)	(%)	(%)	(%)	(%)
P10	186	28.5	0.90	129	19.7	0.9	76.9	63.6	42.5	92.2
P11	175	28.0	1.40	122	16.6	1.0	77.6	58.4	54.8	75.7
P12	173	28.2	1.52	116	18.5	1.0	77.9	70.3	41.4	88.4
Average (P7-P12)	148	27.9	1.36	122	18.9	1.0	77.7	64.1	42.4	81.0

*Primary grind size: 80% passing, microns

The test results showed that at a nominal primary grind size of 80% passing 160 µm, on average 78% of the copper was recovered into the bulk concentrate containing 27% Cu. 64% of the molybdenum was recovered to the concentrate. The metallurgical performance of the sample in the pilot plant tests was inferior to the results obtained from the locked cycle tests. G&T indicated that the inferior metallurgical performance was due to the elevated copper-oxide content in the sample.

A separate open batch test for the 2008 pilot plant tests produced a 30.7% Cu copper-molybdenum bulk concentrate. The copper and molybdenum reporting to the bulk concentrate were 81% and 55%, respectively. The two metals reported to the bulk rougher concentrate were 88.2% for copper and 80.0% for molybdenum.

In 2007, G&T conducted pilot plant campaigns on three composite samples generated from the Paramount, Liard, and West (Breccia) Zones. The composite samples were identified as PZ Composite, LZ Composite, and WZ Composite. The objective of the pilot plant tests was to generate sufficient bulk concentrates for copper-molybdenum separation tests. The flowsheet used was similar to Figure 13-21, but using A3302 as the molybdenum collector instead of FO. The primary grind size varied between 80% passing 82 µm and 119 µm, excluding the first run.

As shown in Table 13-25, the copper recoveries ranged from 62% for the PZ composite to 80% for the WZ composite. Molybdenum recoveries were higher than copper recoveries, ranging from 75% for the PZ composite to 85% for the WZ composite. Possibly due to difficulty maintaining control in small-scale tests, the results produced were not as promising as those obtained from the locked cycle tests.

Table 13-25: Pilot Plant Test Results – Copper-Molybdenum Bulk Concentrate, 2008 (G&T)

Sample	Grade (%)		Recovery (%)		
	Cu	Mo	Mass	Cu	Mo
LZ Composite*	25.2	1.84	0.87	73	77
PZ Composite†	26.5	2.22	0.78	62	75
WZ Composite‡	28	1.06	1.8	80	85

*The data for LZ composite is average data from Run 4.
†The data for PZ composite is average data from Runs 5 and 7.
‡The data for WZ composite is average data from Run 8 (8:30 to 10:30) and Run 11 (8:30 to 11:00).

The lower recoveries produced from the pilot plant tests may be due to:

- The pilot plant's objective was to create a product for further processing, not circuit optimization.
- The pilot plant was run in dayshift batch cycles rather than continuous round the clock campaigns, resulting in shorter periods of stable, optimized performance.

13.5.2 Copper-Molybdenum Separation

Both PRA and G&T conducted copper and molybdenum separation tests using the samples generated from pilot plant campaigns.

G&T used the bulk concentrates produced by the pilot plant campaigns to investigate the molybdenum-copper separation flowsheet. The mineralogical composition and mineral liberation degree are shown in Table 13-26 and Table 13-27.

Table 13-26: Liberation and Composition – Bulk Concentrate, 2008 (G&T)

Item	Mineral Liberation or Composition (%)					
	Chalcopyrite	Bornite	Chalcocite	Molybdenite	Pyrite	Gangues
Liberated	94	91	88	86	90	74
Composition	48.8	12.5	0.7	2	17.3	18.7

Table 13-27: Liberation and Composition – Bulk Concentrate, 2010 (G&T)

Item	Mineral Liberation or Composition (%)			
	Copper Sulphides	Molybdenite	Pyrite	Gangues
Liberated	84.5	75.4	71.8	47.2
Binary with				
Copper Sulphides	-	8.1	9.2	48.6
Molybdenite	0.1	-	0	1
Pyrite	0.2	0	-	1.9
Gangues	14.9	11.3	15.1	-
Multiphases	0.3	5.2	3.9	1.3

The 2008 mineralogical examination indicates that at the regrind size of 80% passing 20 µm, the sulphide minerals were very well liberated. However, the liberation degree of molybdenite was lower than the copper minerals. Also, approximately 26% of the gangue minerals were associated with sulphide minerals, mainly with chalcopyrite and pyrite.

The bulk concentrate used for the 2010 molybdenum-copper separation was coarse in particle size. The averaged particle size was 80% passing 54 µm. Compared to the 2008 test sample, the 2010 sample had a lower liberation degree.

The separation used sodium hydrosulphide to depress copper minerals under nitrogen atmosphere. Although other reagent schemes, such as Nokes (D-910) and sodium cyanide were used as ancillary reagents to selectively suppress copper minerals, they did not improve the separation.

The test results showed that the regrinding of molybdenum rougher flotation concentrate improved the separation efficiency. As reported by the 2010 test work, the molybdenum concentrate grade improved from 38% to 47% molybdenum when the molybdenum rougher concentrate was reground from 80% passing 43 µm to approximately 30 µm.

Five copper-molybdenum separation locked cycle tests were performed on the bulk concentrates generated from the pilot plant tests. The separation tests results are presented in Table 13-28.

Table 13-28: Copper-Molybdenum Separation Test Results

Test ID	Product	Grade Cu (%)	Grade Mo (%)	Recovery Cu (%)	Recovery Mo (%)	Recovery Wt (%)
KM2050/Test 31	Bulk Concentrate	26.6	1.45	100	100	100
	Molybdenum Concentrate	1.63	44.6	0.2	87.6	2.8
	Molybdenum Rougher Tailings	27.4	0.18	99.8	12.4	97.2
KM2050/Test 32	Bulk Concentrate	26.7	1.44	100	100	100
	Molybdenum Concentrate	1.16	50	0.1	67.9	2
	Molybdenum First Cleaner Tailings	22.7	2.62	10	21.3	11.7
	Molybdenum Rougher Tailings	27.8	0.18	90	10.8	86.3
KM2050/Test 33	Bulk Concentrate	26.6	1.5	100	100	100
	Molybdenum Concentrate	1.19	46.4	0.1	74.5	2.4
	Molybdenum First Cleaner Tailings	20.3	2.47	8	17.2	10.5
	Molybdenum Rougher Tailings	28.1	0.14	91.9	8.3	87.1
KM2291/Test 59	Bulk Concentrate	24.5	1.55	100	100	100
	Molybdenum Concentrate	3.48	44.7	0.3	69.8	2.4
	Molybdenum First Cleaner Tailings	25.8	3.52	9.7	20.9	9.2
	Molybdenum Rougher Tailings	24.9	0.16	90	9.2	88.4
KM2291/Test 61	Bulk Concentrate	24.9	1.36	100	100	100
	Molybdenum Concentrate	3.63	44.2	0.3	69.3	2.1
	Molybdenum First Cleaner Tailings	26.5	3.09	9.9	21.1	9.3
	Molybdenum Rougher Tailings	25.3	0.15	89.8	9.5	88.4

The molybdenum recovery to molybdenum concentrate ranged from 67% to 88%. On average, 73.1% of the molybdenum was recovered to the molybdenum concentrates. The molybdenum concentrate grades fluctuated from 44% to 50% molybdenum. Approximately 0.2% of the copper in the bulk concentrate was lost in the molybdenum concentrate.

The locked cycle testing was not conducted at the optimum process conditions. Also as shown in the KM2291 test work, approximately 0.3% of the copper in the copper-molybdenum bulk concentrate was lost in the molybdenum concentrate, on the high end of expectation for this separation process. Further locked cycle testing is recommended to refine the molybdenum separation process conditions to improve molybdenum concentrate grade and molybdenum recovery.

13.5.3 Other Tests

13.5.3.1 Thickening Test

In 2007 G&T conducted three thickening tests on copper concentrate (molybdenum flotation tailings) using a standard cylinder settling method. The tests used A130 as flocculant. The test results and the unit settling rates estimated by G&T are summarized in Table 13-29.

Table 13-29: Settling Test Results, 2007 (G&T)

	Flocculant Dosage	Unit Area	Underflow Solids
Test	(g/t)	(m^2/d/t)	(%)
1	25.2	0.24	60
2	26.5	0.26	60
3	28.0	0.28	60

13.5.3.2 Filtration Test

The 2007 test program also conducted a filtration test on the copper flotation concentrate using a laboratory scale vacuum filter leaf. The test results are shown in Table 13-30.

Table 13-30: Filtration Test Results, 2007 (G&T)

Parameter	Unit	Value
Solid SG	-	3.94
Particle Size, 80% passing	µ	33
Filtration Rate	ml/sec	11
Vacuum	in Hg	60
Filtrate Clarity	-	Good
Final Cake Moisture	%	18.2

13.5.3.3 Copper and Gold Hydrometallurgical Extraction from Concentrates

In 2007, CESL conducted exploratory tests to investigate copper and gold hydrometallurgical extractions. The tests used the CESL proprietary leaching technology, including pressure oxidation of the copper concentrate followed by pressure cyanidation of the copper leach residue. Two-thirds cleaner bulk copper-molybdenum concentrates produced from the PRA test work, assaying 26.3% and 24.9% Cu, were used for the testing. The preliminary tests produced high copper extractions in the range of 96% to 98%, indicating that Schaft Creek copper concentrates were amenable to the CESL process. These copper extractions were achieved at 15 to 30 minutes of retention time as compared to a typical requirement of 60 minutes normally required for a chalcopyrite concentrate.

Pressure cyanidation tests extracted between 89% and 92% of the gold and between 81% and 88% of the silver from the copper leach residue. Sodium cyanide consumption was approximately 3 kg/t due to the high thiocyanate and other metal cyanate compounds formed.

13.5.3.4 Acid Base Accounting Tests

In 2004, PRA conducted ABA tests on 16 selected drill core interval samples. The data are summarized as Table 13-31.

Table 13-31: ABA Test Results, 2004 (PRA)

Item	S(-2) (%)	Paste (pH)	AP	NP	NP/AP	(NP-AP)
Average	0.43	8.8	13.4	75.5	7.36	62.2
Range	0.1 to 0.9	7.5 to 9.3	3.4 to 28.6	53 to 114	3.0 to 16.9	45 to 91

Note: AP = acid generation potential, expressed as kilograms of its CaCO3 equivalent per tonne of sample;
NP = neutralization potential;
S(-2) = sulphide sulphur

13.5.4 Concentrate Multi-Element Assay

The multi-element assay data on the concentrates generated from the locked cycle tests are provided in Table 13-32.

The average copper grade of the concentrate produced from the Schaft Creek mineralization is expected to be higher than that expected by most smelters. On average, the impurities in the copper concentrates produced from the mineralization should not attract smelting penalties as set out by most smelters. Fluorine levels in some of the concentrates may be higher than the penalty thresholds.

Based on the separation testing completed to date, the copper content of the molybdenum concentrate could be higher than the penalty thresholds outlined by most molybdenum smelters. To mitigate the potential penalty, the molybdenum concentrate should be leached by ferric chloride to reduce the copper level, which would increase molybdenum grade as well.

In the 2006 testing program, an 18.44% Mo concentrate generated from Test F75 by the 2006 PRA test program contained 181 ppm rhenium. Mineralogical examination of the concentrate indicated that rhenium occurred as discrete particles intimately associated with the molybdenite. Because the concentrate was contaminated, the rhenium concentration in a 48% to 50% Mo concentrate is expected to be higher than 400 ppm.

In the 2012 testing program, G&T analyzed palladium and platinum for the bulk concentrate generated from Test 34. The concentrate contained 0.17 ppm palladium and 0.07 ppm platinum, respectively.

Table 13-32: Concentrate Multi-element Assay

Element	Symbol	Units	KM2050		KM2291 – Bulk Cu-Mo Concentrate					KM3149 – Bulk Cu-Mo Concentrate
			Mo	Cu	Composite					Conc. Master
			Conc.	Conc.	1	2	3	4	5	Composite 1 [IV-Vi]
Aluminum	Al	%	0.50	1.25	1.24	1.70	0.93	1.27	1.63	0.88
Antimony	Sb	g/t	234	228	24	15	52	32	25	100
Arsenic	As	g/t	3.17	30.5	124	125	143	181	182	85
Barium	Ba	g/t	76	71	-	-	-	-	-	-
Bismuth	Bi	g/t	212	238	53	57	107	57	55	170
Cadmium	Cd	g/t	10	12	8	8	24	8	8	16
Calcium	Ca	%	0.30	0.77	0.56	0.73	0.33	0.46	0.67	0.34
Carbon	C	%	2.12	0.18	-	-	-	-	-	0.15
Chromium	Cr	g/t	226	73	-	-	-	-	-	-
Cobalt	Co	g/t	110	140	56	72	88	96	88	98
Copper	Cu	%	1.27	28.2	32.1	28.6	34.8	30.5	30.3	31.3
Fluorine	F	g/t	-	-	102	145	96	116	130	110
Iron	Fe	%	4.02	28.7	23.2	18.5	24.5	23.7	23.0	25.6
Lead	Pb	%	0.06	0.04	0.02	0.02	0.1	0.01	0.01	0.03
Magnesium	Mg	%	0.15	0.26	0.51	0.89	0.36	0.47	0.59	0.46
Manganese	Mn	%	0.01	<0.01	0.01	0.02	0.01	0.01	0.01	0.004
Mercury	Hg	g/t	<1	<1	<1	<1	<1	<1	<1	<1
Molybdenum	Mo	%	49.6	0.7	0.99	1.37	2.37	2.2	1.91	1.29
Nickel	Ni	%	0.007	0.007	28	56	24	40	56	96
Phosphorus	P	g/t	<1	<1	90	126	70	103	106	73

table continues…

Element	Symbol	Units	KM2050		KM2291 – Bulk Cu-Mo Concentrate					KM3149 – Bulk Cu-Mo Concentrate
			Mo	Cu	Composite					Conc. Master
			Conc.	Conc.	1	2	3	4	5	Composite 1 [IV-Vi]
Potassium	K	g/t	<1	<1	-	-	-	-	-	-
Selenium	Se	g/t	317	150	110	99	103	84	103	134
Silica	SiO2	%	2.76	7.42	3	5	2	3	4	2.12
Silver	Ag	g/t	181	147	131	137	199	131	176	131
Sulphur	S	%	36.7	29.4	26.6	24.7	27.2	26.2	25	32.8
Zinc	Zn	%	0.01	0.03	0.05	0.02	0.16	0.11	0.03	0.08

*Copper concentrate is the molybdenum rougher tailings produced during the copper-molybdenum separation process.

13.6 Projected Metallurgical Performance

Using the results of the locked cycle and variability tests as a basis, the metallurgical performances of the mineralization from the Project have been projected. The metal recovery projections use the average locked cycle test results achieved by G&T at the primary grind size of 80% passing approximately 150 µm and the regression equations that are derived from the plots of the variability test results.

The regression equations for projecting the metal recoveries are adjusted according to the following:

- Metal recoveries produced from the locked cycle tests
- Potential metal losses during the copper-molybdenum separation
- The difference between the proposed flowsheet and the bench test flowsheet

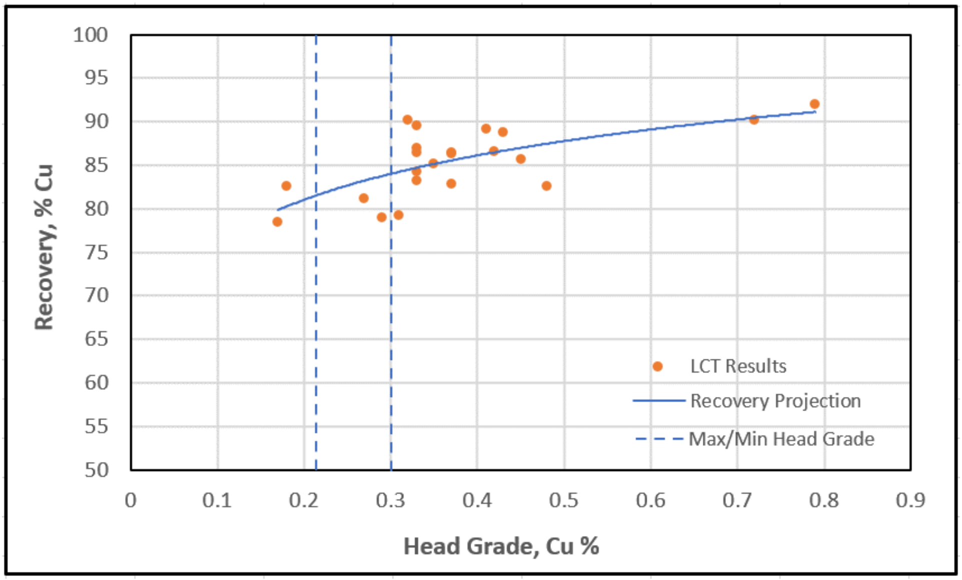
The metal recovery and concentrate grade estimates for the materials, having head grades beyond the range of the tested heads, are assumed based on the test results and experience. The projections are detailed in Table 13-33.

Table 13-33: Copper and Molybdenum Concentrates Projections

Copper Concentrate		
Concentrate Grade	Copper Grade = 28% Cu	
Metal Recovery		
Copper Head	> 1.25% Cu	Copper Recovery = 96.5%
	0.15% to 1.25% Cu	Copper Recovery = 7.3552 x LN (Copper Head, %) + 92.872
	0.10% to 0.15% Cu	Copper Recovery = 70%
	0.05% to 0.10% Cu	Copper Recovery = 45%
	0.02% to 0.05% Cu	Copper Recovery = 10%
	< 0.02% Cu	Copper Recovery = 0%
Gold Head	> 5.0 g/t Au	Gold Recovery = 95%
	1.5 g/t to 5.0 g/t Au	Gold Recovery = 93%
	0.1 g/t to 1.5 g/t Au	Gold Recovery = 9.812 x LN (Gold Head, g/t) + 88.904
	0.05 g/t to 0.1 g/t Au	Gold Recovery = 50%
	0.02 g/t to 0.05 g/t Au	Gold Recovery = 10%
	< 0.02 g/t Au	Gold Recovery = 0%
Silver Head	> 8.0 g/t Ag	Silver Recovery = 92%
	1.0 g/t to 8.0 g/t Ag	Silver Recovery = 23.839 x LN (Silver Head, g/t) + 35.330
	0.5 g/t to 1.0 g/t Ag	Silver Recovery = 20%
	< 0.5 g/t Ag	Silver Recovery = 0%
Molybdenum Concentrate		
Concentrate Grade	Molybdenum Grade = 50% Mo	
Molybdenum Recovery		
Molybdenum Head	> 0.10% Mo	Recovery = 85%
	0.01% to 0.10% Mo	Recovery = (4.195 x LN (Molybdenum Head, %) + 85.424) x 88%
	0.005% to 0.01% Mo	Recovery = 45%
	0.003% to 0.005% Mo	Recovery = 20%
	< 0.003% Mo	Recovery = 0%

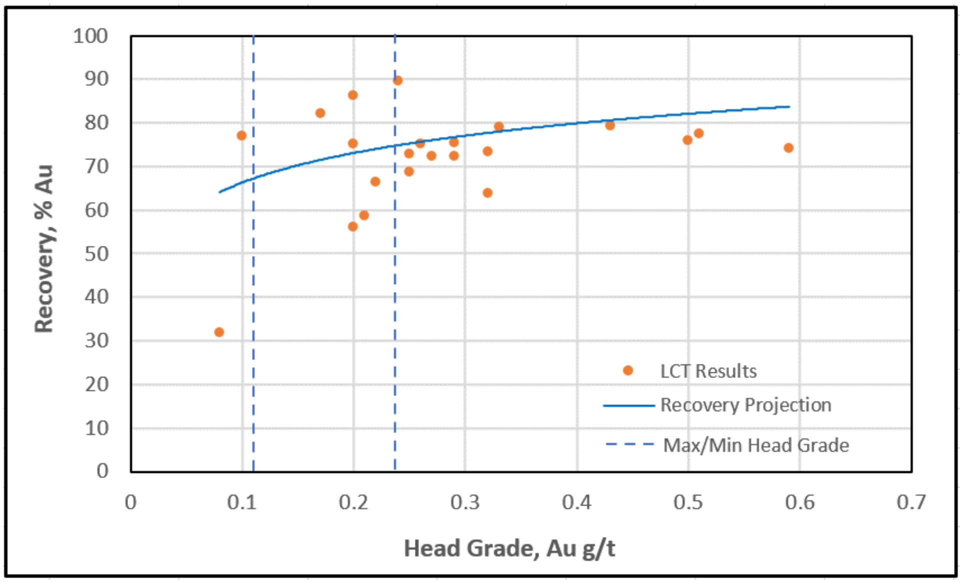
The comparisons between the projected metal recoveries and the test results produced from the locked cycle tests after 2008 are shown in Figure 13-22 to Figure 13-25.

Figure 13-22: Copper Locked Cycle Test Results with Projected Copper Recovery



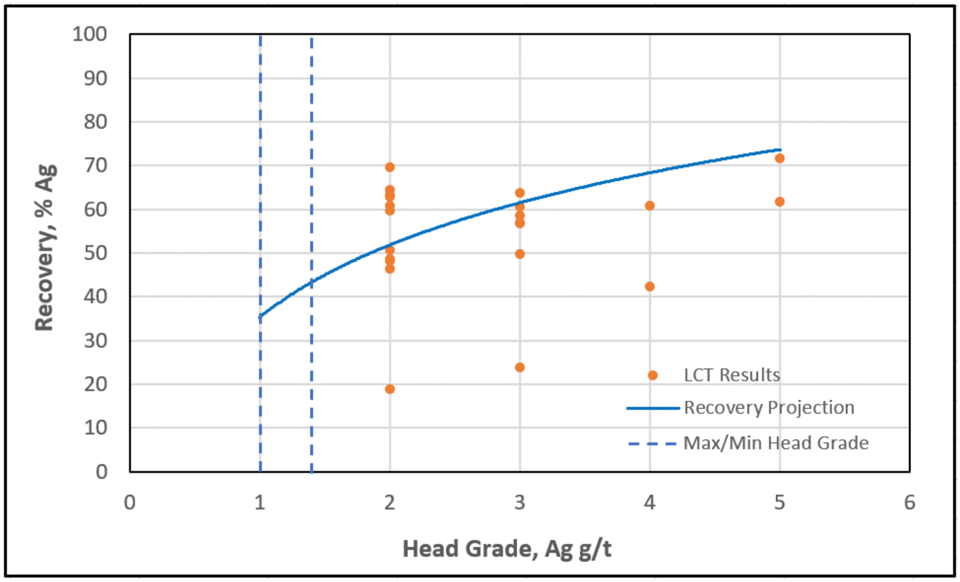
Notes: LCT = locked cycle test
Dashed vertical lines show annual average mill feed head grade range based on 2020 mine plan.

Figure 13-23: Gold Locked Cycle Test Results via Projected Gold Recovery



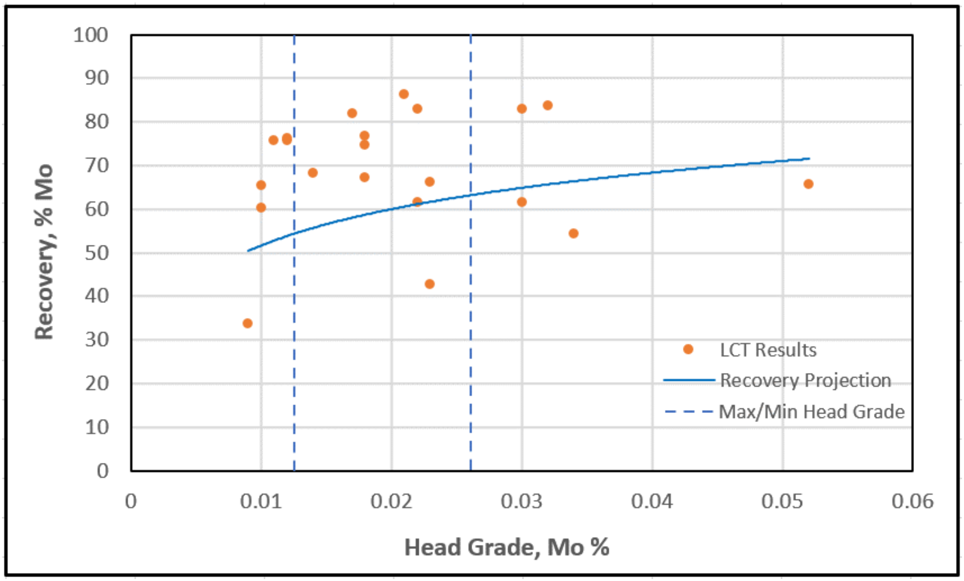
Note: Dashed vertical lines show annual average mill feed head grade range based on 2020 mine plan.

Figure 13-24: Silver Locked Cycle Test Results with Projected Silver Recovery



Note: Dashed vertical lines show annual average mill feed head grade range based on 2020 mine plan.

Figure 13-25: Molybdenum Locked Cycle Test Results with Projected Molybdenum Recovery



Note: Dashed vertical lines showing annual average mill feed head grade range based on 2020 mine plan. The test results are for bulk copper-molybdenum concentrate; the projection is for molybdenum concentrate after copper and molybdenum separation with a separation efficiency of 88%.

Copper concentrate grade is assumed to be 28% and molybdenum concentrate grade is assumed to be 50%, based on the concentrate grades obtained from the locked cycle tests and similar industrial operations. It should be noted that the average copper grade of the bulk copper-molybdenum concentrate from the test works is approximately 30%, which is slightly higher than the projected concentrate grade.

The LOM mill metallurgical performance projections are shown at the end of Section 17.0.

Further test work is recommended to understand metallurgical performances from various mineralization samples. Recommendations are detailed in Section 26.

14.0 MINERAL RESOURCE ESTIMATES

14.1 Introduction

In 2018, the Schaft Creek JV geologists developed the 3D electronic geological model for the Schaft Creek deposit using Leapfrog Geo software. In 2020, the QP imported the triangulated surfaces and wireframes to be used as mineral resource estimation domains into Leapfrog Geo 6.0.3 software. The domains and drillhole assay data were verified in Leapfrog Geo by the QP and independent estimates were made using Leapfrog Edge by the QP.

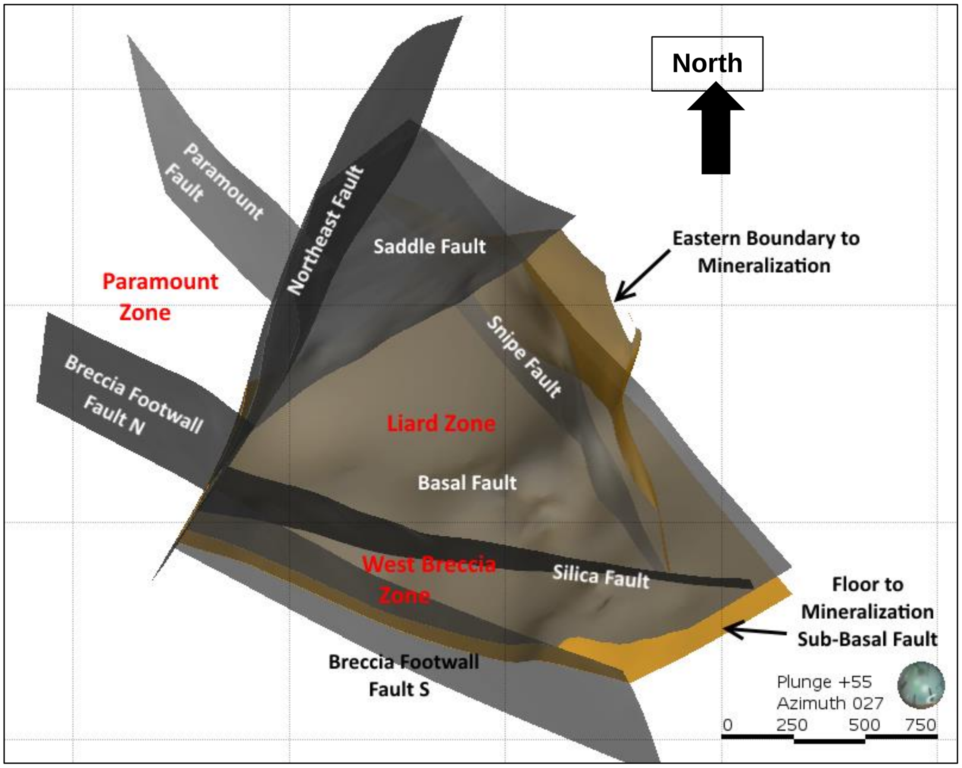
14.2 Geological Models

During 2017 and 2018, original paper cross sections were scanned and imported into Leapfrog by the Schaft Creek JV personnel and projected in 3D space to guide the initial placement of lithological and structural domains in the 3D geological model. Once sections were imported into Leapfrog, volumes from interval data and interpreted linework were interpolated by Leapfrog software and additional structural guides were added by Schaft Creek JV geologists to ensure volumes were geologically reasonable in orientation and magnitude.

Grade shell interpolants were created to domain mineralization within the host Stuhini Group volcanic and sedimentary rocks (grouped for simplicity as “Andesites” [AN]).

A major fault unit, the Basal Fault, was modelled as a mineralization hard boundary. A practical bottom limit to mineralization in the footwall of the Basal Fault was generated by creating a surface conforming to the general shape of the Basal Fault plane which was interpreted to lie 25 m below the end of drill hole traces. Other modelled faults are shown in Figure 14-1.

Figure 14-1: Schematic View Showing Modelled Structural Zones

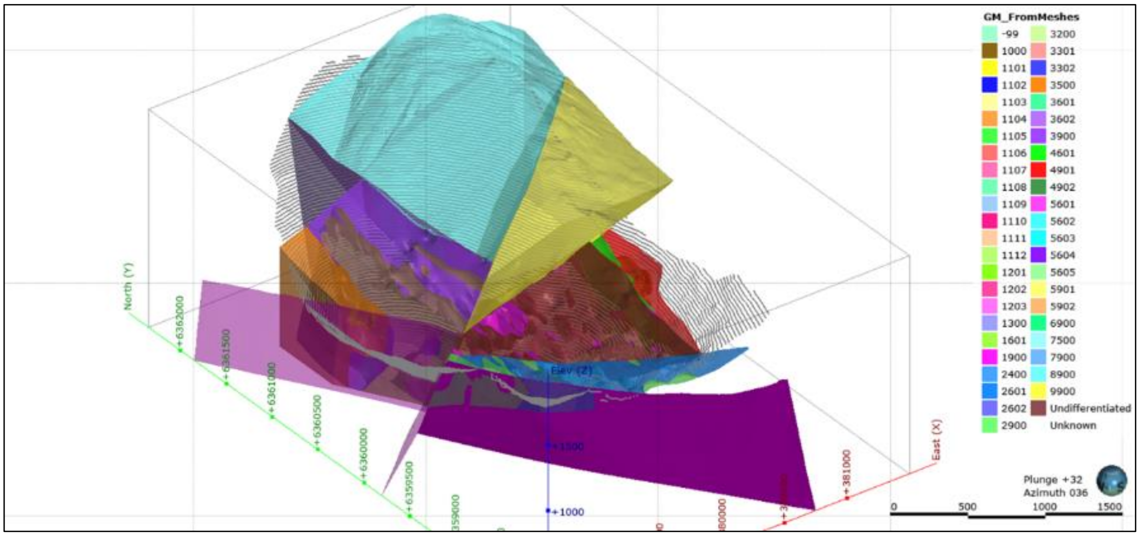


Source: Schaft Creek JV (2019)

In late 2020 the QP imported and validated the Leapfrog wireframes for lithological boundaries and fault blocks using Leapfrog Geo v6.0.3 software. The geological model meshes are the basis for the resource grade estimation domains.

The resource estimation domains are shown in Figure 14-2.

Figure 14-2: Perspective Diagram of Estimation Domains (Conceptual Pit Shown as 15 m contours)



Source: Red Pennant (2021)

A block model was created by the QP and the wireframe surfaces were used to define estimation domains. The model has a parent block size of 20 x 20 x 15 m with 5 x 5 x 5 m sub-blocks.

Grade domains were defined in Leapfrog using the resource estimation domains. Copper, molybdenum, silver, and gold were all estimated using the same domains.

The domains are identified by a four-digit code. The first digit identifies the structural block and the second digit represents lithology type. The third and fourth digits are used to identify individual blocks of similar lithology within the same structural block.

The domains are coded as shown in Table 14-1.

Table 14-1: Domain Coding

	Lithology	Syn Mineral porph	Paramount Breccia	Quartz Monzonite	West Breccia	Granodiorite	Mineralized Andesite	Andesite	Overburden
Structural Block		100	200	300	400	500	600	900	1000
Liard	1000	**1100**	**1200**	**1300**	1400	1500	**1600**	**1900**	**1000**
West Breccia	2000	2100	2200	2300	**2400**	2500	**2600**	**2900**	
Paramount	3000	3100	**3200**	**3300**	3400	**3500**	**3600**	**3900**	
Snipe Wedge	4000	4100	4200	4300	4400	4500	**4600**	**4900**	
Basal Fault Footwall	5000	5100	5200	5300	5400	5500	**5600**	**5900**	
South West of Breccia Footwall	6000	6100	6200	6300	6400	6500	6600	**6900**	
West Wedge Paramount	7000	7100	7200	7300	7400	**7500**	7600	**7900**	
East Wedge Paramount	8000	8100	8200	8300	8400	8500	8600	**8900**	
Saddle Wedge	9000	9100	9200	9300	9400	9500	9600	**9900**	
Notes:									
estimation domains									
	combination								
	combination								
	combination								
	split								

In some instances individual volumes have been combined to simplify estimation based on geological and statistical similarities.

Resource estimation domains are listed in Table 14-2.

Table 14-2: Resource Estimation Domains

1000	11_12_13_16	32_33_36	1900
2400	2601	2602	2900
3500	3900	4601	4901
5601	5602	5603	5604
5605	5901_5902	6900_7500	7900
8900	9900	1000	

In Table 14-1, the bolded codes indicate combinations of structural block and lithology that are valid domains for estimation. Three groups were estimated as combination (yellow) 1100, 1200, 1300, and 1600; (orange) 3200, 3300, and (blue) 3600; 6900 and 7500. Units 2660, 5600, and 5900 (green) are split into contiguous subunits for estimation.

14.3 Exploratory Data Analysis

Exploratory data analysis included review of histograms, scatter plots, sample statistics, and mean versus standard deviation plots. Results (see Table 14-3) included:

- Breccia domains have the highest average grades for all four metals of interest (Cu, Au, Mo, and Ag) and the highest variances along with the syn-mineralisation porphyry; mineralized porphyry domains have moderate average grades and moderate variances; granodiorite domains generally have low average grades and variances.
- Generally, the correlation between copper and molybdenum is poor. The correlation with copper and gold is reasonable in all domains. The correlations between copper and silver vary by domain; good for the breccia, monzodiorite, and the waste andesite domains but poor for the mineralized andesite.

Table 14-3: Length-weighted Raw Sample Grade Statistics

Domain	Metal	Count	Length	Mean Grade	SD	CV	Variance	Minimum	Q1	Q2	Q3	Maximum
1000	Ag (g/t)	349	544.766	0.769	1.157	1.504	1.338	0.020	0.250	0.250	1.115	12.000
11_12_13_16	Ag (g/t)	15278	42951.59	1.274	2.105	1.653	4.430	0.000	0.600	1.086	1.512	208.100
32_33_36	Ag (g/t)	8515	19404.07	1.557	4.241	2.723	17.989	0.050	0.483	1.100	1.800	200.000
1900	Ag (g/t)	1122	2668.089	0.439	0.444	1.012	0.197	0.000	0.250	0.370	0.532	9.600
2400	Ag (g/t)	711	1996.902	2.146	2.593	1.209	6.725	0.050	0.900	1.371	2.632	40.000
2601	Ag (g/t)	721	1817.01	0.892	0.691	0.774	0.477	0.020	0.400	0.847	1.138	7.543
2602	Ag (g/t)	67	167.891	2.006	2.680	1.336	7.184	0.100	0.807	1.056	2.000	14.400
2900	Ag (g/t)	737	2054.875	0.411	0.250	0.607	0.062	0.040	0.250	0.302	0.552	2.329
3500	Ag (g/t)	443	1311.077	0.766	0.513	0.670	0.263	0.098	0.399	0.700	1.015	5.880
3900	Ag (g/t)	1095	2532.15	0.692	8.422	12.167	70.926	0.030	0.100	0.250	0.435	241.800
4601	Ag (g/t)	663	1873.085	0.835	0.423	0.507	0.179	0.100	0.546	0.831	1.095	3.000
4901	Ag (g/t)	444	1167.962	0.422	0.238	0.563	0.057	0.040	0.250	0.380	0.552	2.300
5601	Ag (g/t)	133	344.675	1.117	1.624	1.454	2.637	0.171	0.667	0.886	1.172	20.000
5602	Ag (g/t)	60	173.351	0.708	0.310	0.437	0.096	0.171	0.552	0.763	0.847	1.416
5603	Ag (g/t)	27	58.641	0.677	0.245	0.362	0.060	0.200	0.526	0.683	0.859	1.065
5604	Ag (g/t)	5	10.67	1.615	1.409	0.872	1.984	0.717	0.717	1.115	1.371	4.114
5605	Ag (g/t)	24	74.077	1.160	0.271	0.234	0.074	0.717	0.992	1.144	1.251	1.942
5901_5902	Ag (g/t)	1194	3054.849	0.354	0.416	1.175	0.173	0.000	0.200	0.250	0.452	5.143
6900_7500	Ag (g/t)	931	2598.507	0.448	1.466	3.275	2.150	0.010	0.250	0.250	0.435	40.900

table continues…

Domain	Metal	Count	Length	Mean Grade	SD	CV	Variance	Minimum	Q1	Q2	Q3	Maximum
7900	Ag (g/t)	468	1237.834	0.568	0.440	0.775	0.193	0.050	0.300	0.500	0.736	4.000
8900	Ag (g/t)	585	1270.012	0.568	5.268	9.270	27.755	0.030	0.050	0.100	0.250	100.000
9900	Ag (g/t)	651	1732.11	0.423	1.243	2.938	1.546	0.010	0.209	0.250	0.250	21.100
1000	Au (g/t)	349	544.766	0.094	0.150	1.592	0.023	0.003	0.017	0.040	0.113	1.025
11_12_13_16	Au (g/t)	15278	42951.59	0.202	0.273	1.355	0.075	0.002	0.079	0.147	0.249	19.304
32_33_36	Au (g/t)	8515	19404.07	0.170	0.325	1.918	0.106	0.002	0.028	0.079	0.211	10.834
1900	Au (g/t)	1122	2668.089	0.033	0.061	1.862	0.004	0.002	0.010	0.022	0.037	1.157
2400	Au (g/t)	711	1996.902	0.124	0.264	2.138	0.070	0.003	0.049	0.086	0.143	6.179
2601	Au (g/t)	721	1817.01	0.086	0.081	0.949	0.007	0.003	0.032	0.067	0.111	0.781
2602	Au (g/t)	67	167.891	0.138	0.149	1.083	0.022	0.005	0.069	0.105	0.172	1.020
2900	Au (g/t)	737	2054.875	0.027	0.027	1.029	0.001	0.003	0.008	0.017	0.038	0.417
3500	Au (g/t)	443	1311.077	0.069	0.096	1.395	0.009	0.003	0.017	0.042	0.075	1.354
3900	Au (g/t)	1095	2532.15	0.026	0.046	1.788	0.002	0.003	0.003	0.014	0.032	1.035
4601	Au (g/t)	663	1873.085	0.074	0.062	0.845	0.004	0.003	0.034	0.058	0.093	0.588
4901	Au (g/t)	444	1167.962	0.030	0.058	1.916	0.003	0.003	0.011	0.024	0.039	1.073
5601	Au (g/t)	133	344.675	0.102	0.083	0.816	0.007	0.003	0.055	0.088	0.127	0.794
5602	Au (g/t)	60	173.351	0.082	0.062	0.756	0.004	0.010	0.048	0.069	0.099	0.422
5603	Au (g/t)	27	58.641	0.072	0.088	1.214	0.008	0.009	0.019	0.048	0.065	0.394
5604	Au (g/t)	5	10.67	0.100	0.060	0.597	0.004	0.032	0.062	0.111	0.127	0.177
5605	Au (g/t)	24	74.077	0.139	0.055	0.397	0.003	0.062	0.105	0.132	0.153	0.311

table continues…

Domain	Metal	Count	Length	Mean Grade	SD	CV	Variance	Minimum	Q1	Q2	Q3	Maximum
5901_5902	Au (g/t)	1194	3054.849	0.024	0.045	1.893	0.002	0.002	0.006	0.012	0.028	0.583
6900_7500	Au (g/t)	931	2598.507	0.028	0.097	3.504	0.009	0.001	0.003	0.008	0.024	1.829
7900	Au (g/t)	468	1237.834	0.036	0.075	2.105	0.006	0.003	0.015	0.028	0.045	3.177
8900	Au (g/t)	585	1270.012	0.008	0.028	3.453	0.001	0.003	0.003	0.003	0.007	0.656
9900	Au (g/t)	651	1732.11	0.013	0.023	1.807	0.001	0.002	0.003	0.007	0.013	0.214
1000	Cu (%)	349	544.766	0.139	0.158	1.135	0.025	0.001	0.024	0.082	0.214	0.945
11_12_13_16	Cu (%)	15278	42951.59	0.290	0.202	0.695	0.041	0.001	0.153	0.250	0.376	3.472
32_33_36	Cu (%)	8515	19404.07	0.284	0.245	0.863	0.060	0.001	0.118	0.233	0.388	3.628
1900	Cu (%)	1122	2668.089	0.039	0.067	1.710	0.004	0.001	0.010	0.023	0.043	2.828
2400	Cu (%)	711	1996.902	0.365	0.457	1.252	0.209	0.007	0.140	0.230	0.390	4.503
2601	Cu (%)	721	1817.01	0.211	0.164	0.776	0.027	0.001	0.104	0.170	0.280	2.091
2602	Cu (%)	67	167.891	0.321	0.279	0.870	0.078	0.050	0.120	0.184	0.456	1.200
2900	Cu (%)	737	2054.875	0.040	0.049	1.226	0.002	0.001	0.010	0.029	0.050	0.870
3500	Cu (%)	443	1311.077	0.106	0.112	1.059	0.013	0.001	0.022	0.072	0.160	0.820
3900	Cu (%)	1095	2532.15	0.040	0.066	1.661	0.004	0.001	0.006	0.019	0.049	0.870
4601	Cu (%)	663	1873.085	0.165	0.121	0.734	0.015	0.004	0.080	0.139	0.216	1.260
4901	Cu (%)	444	1167.962	0.035	0.039	1.119	0.002	0.001	0.010	0.026	0.046	0.410
5601	Cu (%)	133	344.675	0.168	0.212	1.262	0.045	0.005	0.060	0.110	0.201	2.158
5602	Cu (%)	60	173.351	0.136	0.117	0.860	0.014	0.026	0.060	0.090	0.170	0.590
5603	Cu (%)	27	58.641	0.093	0.039	0.424	0.002	0.036	0.061	0.087	0.116	0.168

table continues…

Domain	Metal	Count	Length	Mean Grade	SD	CV	Variance	Minimum	Q1	Q2	Q3	Maximum
5604	Cu (%)	5	10.67	0.145	0.096	0.662	0.009	0.060	0.070	0.150	0.180	0.280
5605	Cu (%)	24	74.077	0.208	0.115	0.550	0.013	0.070	0.140	0.190	0.230	0.590
5901_5902	Cu (%)	1194	3054.849	0.031	0.061	1.983	0.004	0.001	0.005	0.010	0.031	0.933
6900_7500	Cu (%)	931	2598.507	0.050	0.117	2.350	0.014	0.000	0.003	0.011	0.046	2.117
7900	Cu (%)	468	1237.834	0.069	0.089	1.287	0.008	0.001	0.019	0.039	0.082	0.601
8900	Cu (%)	585	1270.012	0.017	0.069	4.007	0.005	0.001	0.002	0.005	0.010	1.909
9900	Cu (%)	651	1732.11	0.018	0.048	2.650	0.002	0.001	0.003	0.006	0.013	0.643
1000	Mo (%)	349	544.766	0.0055	0.0128	2.3388	0.0002	0.0001	0.0005	0.0015	0.0072	0.1653
11_12_13_16	Mo (%)	15264	42908.3	0.0159	0.0265	1.6674	0.0007	0.0002	0.0040	0.0090	0.0186	2.8720
32_33_36	Mo (%)	8515	19404.07	0.0219	0.0308	1.4050	0.0009	0.0003	0.0040	0.0122	0.0290	0.5875
1900	Mo (%)	1122	2668.089	0.0032	0.0097	3.0553	0.0001	0.0003	0.0003	0.0006	0.0020	0.1679
2400	Mo (%)	709	1992.642	0.0191	0.0316	1.6487	0.0010	0.0003	0.0042	0.0102	0.0228	0.3477
2601	Mo (%)	713	1796.848	0.0092	0.0170	1.8391	0.0003	0.0003	0.0012	0.0040	0.0114	0.3110
2602	Mo (%)	67	167.891	0.0075	0.0085	1.1282	0.0001	0.0003	0.0018	0.0048	0.0114	0.0468
2900	Mo (%)	731	2042.947	0.0019	0.0037	1.9580	0.0000	0.0001	0.0003	0.0005	0.0012	0.0288
3500	Mo (%)	443	1311.077	0.0070	0.0134	1.9020	0.0002	0.0003	0.0012	0.0024	0.0072	0.1150
3900	Mo (%)	1095	2532.15	0.0018	0.0061	3.3730	0.0000	0.0003	0.0005	0.0005	0.0010	0.1260
4601	Mo (%)	663	1873.085	0.0050	0.0111	2.2092	0.0001	0.0003	0.0006	0.0018	0.0050	0.1481
4901	Mo (%)	441	1160.073	0.0015	0.0028	1.8620	0.0000	0.0001	0.0003	0.0003	0.0012	0.0120
5601	Mo (%)	133	344.675	0.0070	0.0155	2.2263	0.0002	0.0003	0.0003	0.0012	0.0048	0.0923

table continues…

Domain	Metal	Count	Length	Mean Grade	SD	CV	Variance	Minimum	Q1	Q2	Q3	Maximum
5602	Mo (%)	60	173.351	0.0098	0.0057	0.5843	0.0000	0.0003	0.0056	0.0120	0.0120	0.0233
5603	Mo (%)	27	58.641	0.0050	0.0055	1.1031	0.0000	0.0003	0.0012	0.0040	0.0070	0.0240
5604	Mo (%)	5	10.67	0.0294	0.0254	0.8668	0.0006	0.0018	0.0018	0.0288	0.0378	0.0659
5605	Mo (%)	24	74.077	0.0023	0.0033	1.4142	0.0000	0.0003	0.0003	0.0012	0.0024	0.0144
5901_5902	Mo (%)	1157	2944.868	0.0023	0.0074	3.2198	0.0001	0.0001	0.0003	0.0005	0.0012	0.1570
6900_7500	Mo (%)	925	2582.193	0.0017	0.0031	1.8370	0.0000	0.0002	0.0005	0.0005	0.0012	0.0240
7900	Mo (%)	468	1237.834	0.0022	0.0040	1.8075	0.0000	0.0003	0.0003	0.0005	0.0018	0.0324
8900	Mo (%)	585	1270.012	0.0009	0.0033	3.6713	0.0000	0.0005	0.0005	0.0005	0.0005	0.0650
9900	Mo (%)	651	1732.11	0.0011	0.0033	2.9448	0.0000	0.0003	0.0005	0.0005	0.0008	0.0570

The composited grades for the 6 m length composites are summarized by estimation domain in Table 14-4.

Table 14-4: Composite Statistics (6 m lengths)

Domain	Metal	Composites	Length	Mean Grade	SD	CV	Variance	Minimum	Q1	Q2	Q3	Maximum
1000	Ag (g/t)	86	523.4	0.79	0.95	1.19	0.90	0.050	0.250	0.414	1.139	6.273
11_12_13_16	Ag (g/t)	7150	42952.6	1.27	1.44	1.13	2.07	0.050	0.691	1.116	1.520	82.494
1900	Ag (g/t)	429	2542.3	0.44	0.30	0.69	0.09	0.050	0.250	0.388	0.524	3.266
2400	Ag (g/t)	335	1996.5	2.14	2.07	0.97	4.30	0.250	0.989	1.481	2.603	18.158
2601	Ag (g/t)	300	1805.6	0.89	0.56	0.62	0.31	0.070	0.522	0.895	1.110	5.143
2602	Ag (g/t)	128	383.3	2.24	1.94	0.87	3.76	0.250	1.007	1.872	2.529	12.393
2900	Ag (g/t)	343	2035.1	0.41	0.22	0.53	0.05	0.100	0.250	0.341	0.568	1.432
32_33_36	Ag (g/t)	3237	19435.8	1.56	2.62	1.68	6.87	0.050	0.589	1.179	1.827	68.956
3500	Ag (g/t)	218	1311.4	0.77	0.43	0.56	0.18	0.098	0.438	0.707	1.012	3.451
3900	Ag (g/t)	424	2521.9	0.69	6.02	8.68	36.23	0.050	0.133	0.250	0.484	123.097
4601	Ag (g/t)	313	1875.6	0.84	0.38	0.46	0.14	0.171	0.621	0.857	1.077	2.064
4901	Ag (g/t)	197	1169.7	0.42	0.20	0.47	0.04	0.113	0.250	0.417	0.549	1.136
5601	Ag (g/t)	105	621.7	0.86	0.94	1.09	0.88	0.050	0.487	0.714	0.959	8.675
5602	Ag (g/t)	45	268.7	0.65	0.27	0.41	0.07	0.171	0.457	0.702	0.864	1.255
5603	Ag (g/t)	21	125.3	0.74	0.28	0.37	0.08	0.238	0.547	0.720	0.942	1.236
5604	Ag (g/t)	12	71.5	1.74	1.51	0.87	2.28	0.402	0.871	1.085	3.750	4.866
5605	Ag (g/t)	16	96.3	1.00	0.37	0.37	0.14	0.250	0.722	1.014	1.226	1.774
5901_5902	Ag (g/t)	504	3042.4	0.35	0.37	1.03	0.13	0.000	0.209	0.285	0.437	4.904

table continues…

Domain	Metal	Composites	Length	Mean Grade	SD	CV	Variance	Minimum	Q1	Q2	Q3	Maximum
6900_7500	Ag (g/t)	427	2569.2	0.45	1.00	2.22	1.01	0.050	0.235	0.250	0.495	18.543
7900	Ag (g/t)	207	1241.0	0.57	0.37	0.65	0.14	0.050	0.327	0.526	0.710	2.663
8900	Ag (g/t)	310	1863.0	0.43	2.20	5.15	4.85	0.050	0.051	0.095	0.190	28.604
9900	Ag (g/t)	328	1972.7	0.41	0.75	1.83	0.56	0.050	0.206	0.250	0.324	9.240
1000	Au (g/t)	86	523.4	0.09	0.14	1.49	0.02	0.003	0.017	0.047	0.102	0.947
11_12_13_16	Au (g/t)	7150	42952.6	0.20	0.22	1.07	0.05	0.002	0.090	0.155	0.253	9.897
1900	Au (g/t)	429	2542.3	0.03	0.04	1.25	0.00	0.002	0.012	0.024	0.038	0.375
2400	Au (g/t)	335	1996.5	0.12	0.16	1.27	0.02	0.008	0.057	0.092	0.145	2.018
2601	Au (g/t)	300	1805.6	0.09	0.06	0.74	0.00	0.005	0.039	0.073	0.117	0.396
2602	Au (g/t)	128	383.3	0.16	0.16	1.01	0.03	0.012	0.072	0.107	0.167	0.946
2900	Au (g/t)	343	2035.1	0.03	0.02	0.82	0.00	0.002	0.010	0.021	0.040	0.160
32_33_36	Au (g/t)	3237	19435.8	0.17	0.25	1.46	0.06	0.003	0.036	0.091	0.222	5.584
3500	Au (g/t)	218	1311.4	0.07	0.08	1.11	0.01	0.003	0.023	0.044	0.083	0.740
3900	Au (g/t)	424	2521.9	0.03	0.04	1.37	0.00	0.002	0.006	0.015	0.033	0.327
4601	Au (g/t)	313	1875.6	0.07	0.05	0.70	0.00	0.007	0.039	0.058	0.093	0.364
4901	Au (g/t)	197	1169.7	0.03	0.04	1.24	0.00	0.003	0.013	0.026	0.040	0.470
5601	Au (g/t)	105	621.7	0.08	0.06	0.76	0.00	0.008	0.037	0.062	0.100	0.351
5602	Au (g/t)	45	268.7	0.07	0.05	0.71	0.00	0.010	0.034	0.061	0.091	0.228
5603	Au (g/t)	21	125.3	0.07	0.05	0.73	0.00	0.012	0.034	0.056	0.112	0.213
5604	Au (g/t)	12	71.5	0.08	0.04	0.56	0.00	0.016	0.025	0.086	0.122	0.144

table continues…

Domain	Metal	Composites	Length	Mean Grade	SD	CV	Variance	Minimum	Q1	Q2	Q3	Maximum
5605	Au (g/t)	16	96.3	0.12	0.06	0.54	0.00	0.011	0.065	0.109	0.148	0.271
5901_5902	Au (g/t)	504	3042.4	0.02	0.04	1.54	0.00	0.002	0.007	0.014	0.029	0.434
6900_7500	Au (g/t)	427	2569.2	0.03	0.08	2.77	0.01	0.001	0.003	0.009	0.028	1.181
7900	Au (g/t)	207	1241.0	0.04	0.04	1.23	0.00	0.003	0.019	0.031	0.045	0.694
8900	Au (g/t)	310	1863.0	0.01	0.02	2.06	0.00	0.003	0.003	0.003	0.007	0.220
9900	Au (g/t)	328	1972.7	0.01	0.02	1.66	0.00	0.002	0.003	0.007	0.014	0.201
1000	Cu (%)	86	523.4	0.14	0.15	1.11	0.02	0.000	0.030	0.092	0.216	0.722
11_12_13_16	Cu (%)	7150	42952.6	0.29	0.18	0.61	0.03	0.001	0.167	0.260	0.374	1.560
1900	Cu (%)	429	2542.3	0.04	0.05	1.36	0.00	0.001	0.014	0.026	0.042	0.538
2400	Cu (%)	335	1996.5	0.37	0.40	1.11	0.16	0.018	0.155	0.239	0.404	3.824
2601	Cu (%)	300	1805.6	0.21	0.13	0.61	0.02	0.040	0.116	0.180	0.272	0.836
2602	Cu (%)	128	383.3	0.48	0.43	0.90	0.19	0.037	0.190	0.370	0.632	2.912
2900	Cu (%)	343	2035.1	0.04	0.04	0.98	0.00	0.001	0.014	0.032	0.054	0.310
32_33_36	Cu (%)	3237	19435.8	0.28	0.21	0.73	0.04	0.001	0.139	0.245	0.381	2.538
3500	Cu (%)	218	1311.4	0.11	0.09	0.90	0.01	0.001	0.028	0.080	0.158	0.432
3900	Cu (%)	424	2521.9	0.04	0.05	1.35	0.00	0.001	0.007	0.023	0.052	0.378
4601	Cu (%)	313	1875.6	0.17	0.10	0.60	0.01	0.011	0.098	0.142	0.215	0.771
4901	Cu (%)	197	1169.7	0.04	0.03	0.85	0.00	0.001	0.013	0.029	0.047	0.184
5601	Cu (%)	105	621.7	0.13	0.14	1.14	0.02	0.005	0.043	0.093	0.150	1.090
5602	Cu (%)	45	268.7	0.13	0.13	1.00	0.02	0.015	0.054	0.090	0.140	0.658

table continues…

Domain	Metal	Composites	Length	Mean Grade	SD	CV	Variance	Minimum	Q1	Q2	Q3	Maximum
5603	Cu (%)	21	125.3	0.12	0.07	0.56	0.00	0.025	0.077	0.110	0.152	0.246
5604	Cu (%)	12	71.5	0.11	0.07	0.59	0.00	0.022	0.048	0.108	0.170	0.216
5605	Cu (%)	16	96.3	0.17	0.12	0.69	0.01	0.008	0.076	0.147	0.221	0.495
5901_5902	Cu (%)	504	3042.4	0.03	0.05	1.54	0.00	0.000	0.007	0.018	0.036	0.503
6900_7500	Cu (%)	427	2569.2	0.05	0.09	1.78	0.01	0.000	0.005	0.017	0.058	0.838
7900	Cu (%)	207	1241.0	0.07	0.07	1.03	0.00	0.001	0.026	0.045	0.084	0.376
8900	Cu (%)	310	1863.0	0.01	0.04	2.59	0.00	0.000	0.004	0.007	0.012	0.378
9900	Cu (%)	328	1972.7	0.02	0.05	2.19	0.00	0.001	0.004	0.007	0.015	0.334
1000	Mo (%)	86	523.4	0.01	0.01	2.02	0.00	0.0001	0.0005	0.0015	0.0071	0.0897
11_12_13_16	Mo (%)	7142	42902.2	0.0159	0.0199	1.2513	0.0004	0.0003	0.0051	0.0105	0.0196	0.3186
1900	Mo (%)	429	2542.3	0.0032	0.0075	2.3401	0.0001	0.0003	0.0005	0.0009	0.0028	0.0801
2400	Mo (%)	334	1991.5	0.0191	0.0282	1.4800	0.0008	0.0004	0.0056	0.0111	0.0230	0.3106
2601	Mo (%)	297	1785.4	0.0093	0.0133	1.4371	0.0002	0.0003	0.0013	0.0044	0.0119	0.0997
2602	Mo (%)	128	383.3	0.0135	0.0190	1.4012	0.0004	0.0005	0.0029	0.0061	0.0180	0.1133
2900	Mo (%)	342	2028.8	0.0019	0.0035	1.8894	0.0000	0.0002	0.0003	0.0005	0.0012	0.0233
32_33_36	Mo (%)	3237	19435.8	0.0219	0.0236	1.0797	0.0006	0.0003	0.0057	0.0146	0.0302	0.2460
3500	Mo (%)	218	1311.4	0.0070	0.0105	1.5029	0.0001	0.0003	0.0012	0.0029	0.0089	0.0697
3900	Mo (%)	424	2521.9	0.0018	0.0041	2.2742	0.0000	0.0003	0.0005	0.0005	0.0012	0.0525
4601	Mo (%)	313	1875.6	0.0050	0.0084	1.6814	0.0001	0.0003	0.0011	0.0023	0.0053	0.0782
4901	Mo (%)	196	1163.4	0.0015	0.0028	1.8476	0.0000	0.0003	0.0003	0.0005	0.0009	0.0120

table continues…

Domain	Metal	Composites	Length	Mean Grade	SD	CV	Variance	Minimum	Q1	Q2	Q3	Maximum
5601	Mo (%)	105	621.7	0.0047	0.0096	2.0626	0.0001	0.0003	0.0005	0.0009	0.0030	0.0530
5602	Mo (%)	45	268.7	0.0080	0.0055	0.6852	0.0000	0.0003	0.0008	0.0103	0.0120	0.0177
5603	Mo (%)	21	125.3	0.0064	0.0039	0.6136	0.0000	0.0003	0.0040	0.0060	0.0089	0.0165
5604	Mo (%)	12	71.5	0.0202	0.0160	0.7953	0.0003	0.0035	0.0068	0.0087	0.0367	0.0476
5605	Mo (%)	16	96.3	0.0020	0.0022	1.1174	0.0000	0.0003	0.0007	0.0008	0.0026	0.0077
5901_5902	Mo (%)	486	2932.8	0.0023	0.0056	2.4233	0.0000	0.0001	0.0003	0.0005	0.0011	0.0549
6900_7500	Mo (%)	424	2549.0	0.0017	0.0029	1.7061	0.0000	0.0003	0.0005	0.0005	0.0013	0.0198
7900	Mo (%)	207	1241.0	0.0022	0.0033	1.4687	0.0000	0.0003	0.0005	0.0009	0.0024	0.0238
8900	Mo (%)	310	1863.0	0.0008	0.0020	2.5394	0.0000	0.0005	0.0005	0.0005	0.0005	0.0222
9900	Mo (%)	328	1972.7	0.0014	0.0034	2.5328	0.0000	0.0003	0.0005	0.0005	0.0009	0.0384

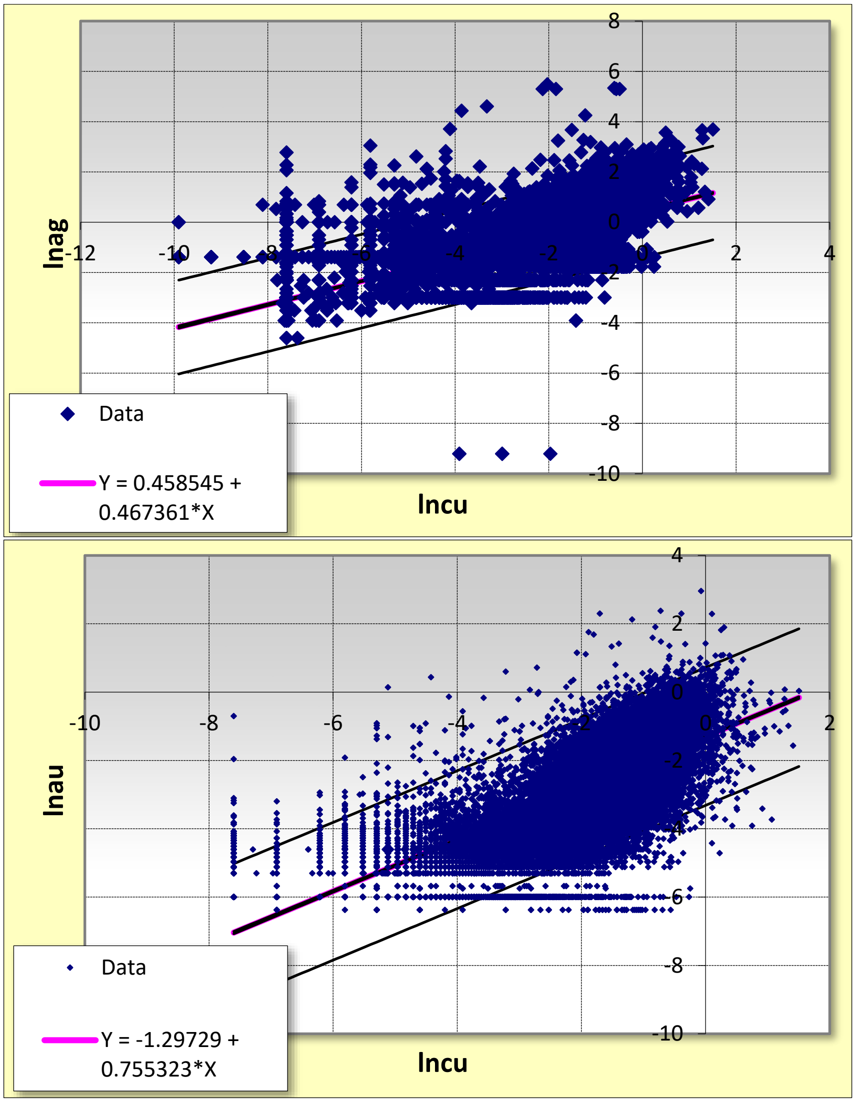
14.3.1 Gold and Silver Values Calculated by Regression

The dataset (a total of 37,616 samples) contains samples with missing values for silver and gold. This is due to some samples not being assayed for these elements (silver and gold) and also because some poor-quality assay values were removed from the dataset. In 2018, 13,009 silver and 8,180 gold samples were replaced with a value calculated from copper-related regressions based on the more recent and reliable silver and gold assays.

All three variables (silver, gold, and copper) display a lognormal distribution. Natural logs were calculated for each variable and used in the estimation. The correlation coefficient shows the relationship between variables, where 0 is no relationship and 1 is a direct relationship. The correlation coefficients for the 6 m composite log data for all domains combined is displayed in Table 14-5, and the relationship is shown with scatter plots in Figure 14-3.

The natural log of the grades for the variables were plotted and a regression line was fitted to the data. The R-squared values gives an indication of the variance of the data from the regression line (the success of the fit).

Figure 14-3: Scatter Plots for Log Transformed Gold and Silver by Copper



Source: Schaft Creek JV (2018)

The equation of the line gives a prediction of the LnAg or LnAu in relation to LnCu in the format of Y = A + B*X. A value was assigned for each missing LnAg and LnAu value using the following formulas:

$$\mathrm{LnAg} = 0.458545031 + 0.647360968 * \mathrm{LnCu}$$

$$\mathrm{LnAu} = -1.297289347 + 0.755323121 * \mathrm{LnCu}$$

In order to calculate the silver and gold values from the natural log prediction, the predicted LnAg and LnAu values are back-transformed by taking the antilog with a minor correction to account for the difference between the geometric and arithmetic mean. The mean of the back-transformed data is calculated from the variance of the LnAg and LnAu and the R-squared.

To calculate any value of silver or gold is the same as calculating the mean value using:

$$m = e^{\alpha + \beta 2/2}$$

Where

$$\beta^2 = \sigma^2.(1 - R^2)$$

The back calculated predicted silver and gold values are summarised in Table 14-5.

Table 14-5: Summary of Statistics for the Prediction of Gold and Silver from Regression with Copper

	Ag	Au
Correlation with LnCu	0.714	0.815
R-squared	0.509769	0.664
Variance (σ^2)	1.844	3.145
$\beta^2 = (\sigma^2.(1 - R^2))$	0.903985	1.05672
m = ($e^{\alpha + \beta 2/2}$)	Exp (LnCu+0.45199)	Exp (LnCu+0.52836)

The resulting data has the following median and mean values (Table 14-6).

Table 14-6: Summary of Statistics of Gold and Silver Calculated by Regression

	Original Data			Grade Calculated from Regression		
	Number	**Median**	**Mean**	**Number**	**Median**	**Mean**
Cu%	38494	0.150	0.205	0	-	-
Mo %	38368	0.005	0.012	0	-	-
Ag g/t	24607	0.040	1.119	13009	0.965	0.977
Au g/t	29436	0.064	0.143	8180	0.052	0.079

The QP considers that this is a reasonable approach to reduce the risk posed by potentially inaccurate historic assay data

14.4 Domain Estimation Boundaries

Grade estimates in domains were either estimated as discrete ("hard" boundary estimation) units with no data used outside the domains, or as gradational ("soft" boundary) units with limited data used adjacent to and outside the domain. Contact analysis and structural context was used to determine which domains could be combined for resource estimation and where hard or soft boundaries should be used.

14.5 Density Assignment

Bulk density was applied to the model by assigning the average value of 2.69 g/cm^3 for all domains except overburden (domain 1000), which has an assigned grade of 2.0 g/cm^3.

14.6 Grade Capping/Outlier Restrictions

Top cuts were applied where necessary. The application of a grade cap and threshold value were determined by analysis of histograms, log histograms, and mean variance plots for each domain. Capping values are shown in the statistical summary tables for composites.

The data are mostly sampled at 3 m or 2 m intervals, with smaller intervals being used around geological boundaries and some larger samples being taken. The data were composited to 6 m intervals, broken at geological boundaries.

Samples remaining at the base of the domain of <6 m were treated as residuals. Residuals <3 m were merged with the composite sample above. The QP regards the compositing strategy as reasonable to provide equivalent volumetric support for estimation.

14.7 Variography

Variogram modelling was undertaken using Leapfrog EDGE. Variograms were modelled for copper, molybdenum, silver, and gold using normal scores transformed data to reduce the masking effects of extreme values.

Traditional variograms with spherical models were used to model the data in all cases. In all cases, nugget values were estimated using the downhole variogram, with 6 m lag spacing to match the composite length. Lag spacing for the directional variograms was generally set to multiples of the 6 m composite length and the angular tolerance was adjusted where necessary to develop experimental variograms.

The direction of continuity varies by domain. Variograms were modelled with a nugget and either one or two spherical structures. The variograms are robust for the major domains.

A local orientation model (see Figure 14-4) for variography and search was developed using Leapfrog Geo and Leapfrog EDGE from the orientations of the copper variogram models. The orientations of the variogram models were used to build smoothed and nested form interpolants using leapfrog Geo. The orientations of form interpolants were projected to each block in the block model and used to orientate the search and variogram ellipsoids for weighting composite grades during inverse distance and kriging estimation.

The variogram model parameters used for Kriging estimates are summarized in Table 14-7.

Table 14-7: Variogram Parameters

General	Direction							Structure 1							Structure 2						
Variogram Name	Dip	Dip Azimuth	Pitch	Model Space	Variance	Nugget	Normalised Nugget	Sill	Normalised Sill	Structure	Alpha	Major	Semi-major	Minor	Sill	Normalised Sill	Structure	Alpha	Major	Semi-major	Minor
ag2_gt 1000: Model	27.7	249.8	65.2	data	0.878	0.689	0.784	0.189	0.215	Spherical		82.7	49.2	25.5							
ag2_gt 1000: Model	27.7	249.8	65.2	normal score	0.997	0.690		0.308		Spherical		82.7	49.2	25.5							
ag2_gt 11_12_13_16: Model	26.0	262.0	107.8	data	2.069	0.443	0.214	0.997	0.482	Spherical		25.7	32.8	39.4	0.625	0.302	Spherical		139.3	160.5	162.9
ag2_gt 11_12_13_16: Model	26.0	262.0	107.8	normal score	1.000	0.146		0.419		Spherical		25.7	32.8	39.4	0.431		Spherical		139.3	160.5	162.9
ag2_gt 1900: Model	17.0	288.9	138.0	data	0.082	0.042	0.508	0.017	0.205	Spherical		56.2	39.2	24.6	0.023	0.285	Spherical		222.4	242.4	63.2
ag2_gt 1900: Model	17.0	288.9	138.0	normal score	0.999	0.392		0.188		Spherical		56.2	39.2	24.6	0.417		Spherical		222.4	242.4	63.2
ag2_gt 2400: Model	17.0	288.9	147.5	data	4.293	1.397	0.325	2.069	0.482	Spherical		22.6	11.9	14.2	0.828	0.193	Spherical		128.9	57.8	77.0
ag2_gt 2400: Model	17.0	288.9	147.5	normal score	0.999	0.233		0.478		Spherical		22.6	11.9	14.2	0.290		Spherical		128.9	57.8	77.0
ag2_gt 2601: Model	17.0	288.9	154.2	data	0.311	0.153	0.491	0.092	0.297	Spherical		69.4	67.7	28.3	0.066	0.211	Spherical		186.3	152.0	63.0
ag2_gt 2601: Model	17.0	288.9	154.2	normal score	0.999	0.376		0.296		Spherical		69.4	67.7	28.3	0.325		Spherical		186.3	152.0	63.0
ag2_gt 2602: Model	83.8	79.8	4.6	data	3.754	1.626	0.433	0.672	0.179	Spherical		62.4	29.2	14.3	1.441	0.384	Spherical		206.4	166.9	139.0
ag2_gt 2602: Model	83.8	79.8	4.6	normal score	0.998	0.324		0.122		Spherical		62.4	29.2	14.3	0.548		Spherical		206.4	166.9	139.0
ag2_gt 2900: Model	11.5	344.8	67.1	data	0.048	0.014	0.286	0.019	0.392	Spherical		54.2	29.2	22.6	0.015	0.322	Spherical		247.3	238.6	123.7
ag2_gt 2900: Model	11.5	344.8	67.1	normal score	0.999	0.201		0.337		Spherical		54.2	29.2	22.6	0.462		Spherical		247.3	238.6	123.7
ag2_gt 32_33_36: Model	44.9	73.7	69.1	normal score	1.000	0.157		0.355		Spherical		21.3	29.2	32.9	0.485		Spherical		161.2	123.4	111.8
ag2_gt 32_33_36: Model	44.9	73.7	69.1	data	6.873	1.573	0.229	2.865	0.417	Spherical		21.3	29.2	32.9	2.424	0.353	Spherical		161.2	123.4	111.8
ag2_gt 3500: Model	21.4	231.4	24.9	normal score	0.999	0.160		0.239		Spherical		30.0	92.5	15.7	0.599		Spherical		204.1	199.5	114.8
ag2_gt 3500: Model	21.4	231.4	24.9	data	0.183	0.042	0.232	0.058	0.315	Spherical		30.0	92.5	15.7	0.083	0.452	Spherical		204.1	199.5	114.8
ag2_gt 3900: Model	44.3	85.5	170.1	data	35.921	11.455	0.319	10.489	0.292	Spherical		30.2	15.0	27.1	13.916	0.387	Spherical		138.1	142.0	139.9
ag2_gt 3900: Model	44.3	85.5	170.1	normal score	0.999	0.227		0.230		Spherical		30.2	15.0	27.1	0.540		Spherical		138.1	142.0	139.9
ag2_gt 4601: Model	87.9	281.2	14.7	data	0.145	0.041	0.283	0.041	0.284	Spherical		62.4	12.0	30.4	0.062	0.430	Spherical		161.0	91.8	60.7

table continues…

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General	Direction							Structure 1							Structure 2						
Variogram Name	Dip	Dip Azimuth	Pitch	Model Space	Variance	Nugget	Normalised Nugget	Sill	Normalised Sill	Structure	Alpha	Major	Semi-major	Minor	Sill	Normalised Sill	Structure	Alpha	Major	Semi-major	Minor
ag2_gt 4601: Model	87.9	281.2	14.7	normal score	0.999	0.199		0.173		Spherical		62.4	12.0	30.4	0.624		Spherical		161.0	91.8	60.7
ag2_gt 4901: Model	78.8	95.6	111.5	data	0.038	0.011	0.296	0.013	0.341	Spherical		17.5	32.4	25.5	0.014	0.362	Spherical		116.5	78.2	73.4
ag2_gt 4901: Model	78.8	95.6	111.5	normal score	0.999	0.209		0.291		Spherical		17.5	32.4	25.5	0.500		Spherical		116.5	78.2	73.4
ag2_gt 5601: Model	17.5	355.3	109.8	data	0.878	0.194	0.221	0.682	0.776	Spherical		151.3	121.6	54.0							
ag2_gt 5601: Model	17.5	355.3	109.8	normal score	0.997	0.152		0.844		Spherical		151.3	121.6	54.0							
ag2_gt 5602: Model	17.5	355.3	108.2	normal score	0.994	0.162		0.831		Spherical		141.5	121.6	54.0							
ag2_gt 5602: Model	17.5	355.3	108.2	data	0.000	0.000	0.235	0.000	0.761	Spherical		141.5	121.6	54.0							
ag2_gt 5603: Model	17.5	355.3	77.8	data	0.076	0.010	0.136	0.065	0.856	Spherical		141.5	121.6	54.0							
ag2_gt 5603: Model	17.5	355.3	77.8	normal score	0.988	0.090		0.897		Spherical		141.5	121.6	54.0							
ag2_gt 5604: Model	17.5	355.3	85.0	data	2.262	0.838	0.370	1.396	0.617	Spherical		141.5	121.6	54.0							
ag2_gt 5604: Model	17.5	355.3	85.0	normal score	0.981	0.270		0.710		Spherical		141.5	121.6	54.0							
ag2_gt 5605: Model	17.5	355.3	92.3	normal score	0.985	0.294		0.689		Spherical		141.5	121.6	54.0							
ag2_gt 5605: Model	17.5	355.3	92.3	data	0.000	0.000	0.400	0.000	0.590	Spherical		141.5	121.6	54.0							
ag2_gt 5901_5902: Model	19.6	324.9	110.1	data	0.134	0.047	0.348	0.025	0.188	Spherical		62.4	29.2	14.6	0.062	0.463	Spherical		143.3	203.3	119.7
ag2_gt 5901_5902: Model	19.6	324.9	110.1	normal score	0.999	0.251		0.069		Spherical		62.4	29.2	14.6	0.677		Spherical		143.3	203.3	119.7
ag2_gt 6900_7500: Model	6.1	31.7	23.4	data	1.010	0.345	0.341	0.253	0.250	Spherical		62.4	29.2	15.9	0.413	0.409	Spherical		211.1	203.3	99.5
ag2_gt 6900_7500: Model	6.1	31.7	23.4	normal score	0.999	0.246		0.174		Spherical		62.4	29.2	15.9	0.581		Spherical		211.1	203.3	99.5
ag2_gt 7900: Model	74.2	77.7	104.7	data	0.140	0.031	0.221	0.055	0.391	Spherical		30.0	20.8	40.0	0.055	0.390	Spherical		260.5	240.0	80.5
ag2_gt 7900: Model	74.2	77.7	104.7	normal score	0.999	0.152		0.324		Spherical		30.0	20.8	40.0	0.527		Spherical		260.5	240.0	80.5
ag2_gt 8900: Model	15.5	205.0	63.8	normal score	0.999	0.447		0.198		Spherical		30.0	25.0	19.6	0.355		Spherical		278.9	227.2	63.1
ag2_gt 8900: Model	15.5	205.0	63.8	data	7.323	4.138	0.565	1.386	0.189	Spherical		30.0	25.0	19.6	1.801	0.246	Spherical		278.9	227.2	63.1
ag2_gt 9900: Model	15.5	205.0	63.8	normal score	0.999	0.310		0.107		Spherical		30.0	25.0	13.6	0.578		Spherical		184.2	120.3	47.2

table continues…

General	Direction							Structure 1							Structure 2						
Variogram Name	Dip	Dip Azimuth	Pitch	Model Space	Variance	Nugget	Normalised Nugget	Sill	Normalised Sill	Structure	Alpha	Major	Semi-major	Minor	Sill	Normalised Sill	Structure	Alpha	Major	Semi-major	Minor
ag2_gt 9900: Model	15.5	205.0	63.8	data	0.625	0.261	0.418	0.093	0.148	Spherical		30.0	25.0	13.6	0.269	0.431	Spherical		184.2	120.3	47.2
ag2_gt: Model	5.3	225.6	35.8	normal score	1.000	0.079		0.202		Spherical		34.0	26.0	19.7	0.520		Spherical		275.0	108.1	149.1
ag2_gt: Model	5.3	225.6	35.8	data	4.066	0.374	0.092	0.931	0.229	Spherical		34.0	26.0	19.7	2.043	0.502	Spherical		275.0	108.1	149.1
au2_gt 1000: Model	27.7	249.8	65.2	data	0.021	0.016	0.784	0.005	0.215	Spherical		82.7	49.2	25.5							
au2_gt 1000: Model	27.7	249.8	65.2	normal score	0.997	0.690		0.308		Spherical		82.7	49.2	25.5							
au2_gt 11_12_13_16: Model	15.4	342.5	0.0	data	0.047	0.010	0.214	0.013	0.273	Spherical		29.4	13.9	19.7	0.020	0.417	Spherical		112.5	170.5	129.0
au2_gt 11_12_13_16: Model	15.4	342.5	0.0	normal score	1.000	0.146		0.186		Spherical		29.4	13.9	19.7	0.518		Spherical		112.5	170.5	129.0
au2_gt 1900: Model	17.0	288.9	138.0	normal score	0.999	0.278		0.201		Spherical		54.2	29.2	14.2	0.523		Spherical		266.4	182.3	57.6
au2_gt 1900: Model	17.0	288.9	138.0	data	0.002	0.001	0.381	0.000	0.247	Spherical		54.2	29.2	14.2	0.001	0.374	Spherical		266.4	182.3	57.6
au2_gt 2400: Model	17.0	288.9	138.0	data	0.024	0.010	0.400	0.007	0.304	Spherical		53.1	29.2	26.6	0.007	0.294	Spherical		134.7	72.9	121.7
au2_gt 2400: Model	17.0	288.9	138.0	normal score	0.999	0.295		0.257		Spherical		53.1	29.2	26.6	0.445		Spherical		134.7	72.9	121.7
au2_gt 2601: Model	17.0	288.9	138.0	data	0.004	0.002	0.465	0.001	0.148	Spherical		62.4	29.2	14.3	0.002	0.384	Spherical		206.4	166.9	139.0
au2_gt 2601: Model	17.0	288.9	138.0	normal score	0.999	0.352		0.094		Spherical		62.4	29.2	14.3	0.549		Spherical		206.4	166.9	139.0
au2_gt 2602: Model	83.8	79.8	4.6	data	0.026	0.013	0.519	0.002	0.093	Spherical		62.4	29.2	14.3	0.010	0.384	Spherical		206.4	166.9	139.0
au2_gt 2602: Model	83.8	79.8	4.6	normal score	0.998	0.402		0.043		Spherical		62.4	29.2	14.3	0.548		Spherical		206.4	166.9	139.0
au2_gt 2900: Model	11.5	344.8	67.1	data	0.000	0.000	0.380	0.000	0.247	Spherical		54.2	29.2	14.2	0.000	0.368	Spherical		256.1	106.1	139.3
au2_gt 2900: Model	11.5	344.8	67.1	normal score	0.999	0.278		0.201		Spherical		54.2	29.2	14.2	0.514		Spherical		256.1	106.1	139.3
au2_gt 32_33_36: Model	44.9	73.7	174.3	normal score	1.000	0.138		0.259		Spherical		19.6	29.2	40.3	0.604		Spherical		235.6	179.7	122.4
au2_gt 32_33_36: Model	44.9	73.7	174.3	data	0.061	0.012	0.203	0.020	0.329	Spherical		19.6	29.2	40.3	0.028	0.469	Spherical		235.6	179.7	122.4
au2_gt 3500: Model	21.4	231.4	32.6	normal score	0.999	0.201		0.146		Spherical		30.0	29.2	19.9	0.648		Spherical		213.9	173.5	129.2
au2_gt 3500: Model	21.4	231.4	32.6	data	0.006	0.002	0.286	0.001	0.210	Spherical		30.0	29.2	19.9	0.003	0.502	Spherical		213.9	173.5	129.2
au2_gt 3900: Model	44.3	85.5	170.1	normal score	0.999	0.138		0.119		Spherical		30.2	42.8	12.1	0.739		Spherical		300.2	300.2	79.3

table continues…

General	Direction							Structure 1							Structure 2						
Variogram Name	Dip	Dip Azimuth	Pitch	Model Space	Variance	Nugget	Normalised Nugget	Sill	Normalised Sill	Structure	Alpha	Major	Semi-major	Minor	Sill	Normalised Sill	Structure	Alpha	Major	Semi-major	Minor
au2_gt 3900: Model	44.3	85.5	170.1	data	0.001	0.000	0.203	0.000	0.188	Spherical		30.2	42.8	12.1	0.001	0.606	Spherical		300.2	300.2	79.3
au2_gt 4601: Model	87.9	281.2	0.8	normal score	0.999	0.295		0.259		Spherical		35.7	15.1	25.5	0.449		Spherical		160.7	257.9	60.5
au2_gt 4601: Model	87.9	281.2	0.8	data	0.003	0.001	0.400	0.001	0.290	Spherical		35.7	15.1	25.5	0.001	0.311	Spherical		160.7	257.9	60.5
au2_gt 4901: Model	78.8	95.6	176.7	data	0.001	0.000	0.296	0.001	0.443	Spherical		62.4	31.3	25.5	0.000	0.258	Spherical		162.1	150.8	131.7
au2_gt 4901: Model	78.8	95.6	176.7	normal score	0.999	0.209		0.393		Spherical		62.4	31.3	25.5	0.393		Spherical		162.1	150.8	131.7
au2_gt 5601: Model	17.5	355.3	99.1	normal score	0.997	0.138		0.858		Spherical		151.3	121.6	54.0							
au2_gt 5601: Model	17.5	355.3	99.1	data	0.020	0.004	0.202	0.016	0.795	Spherical		151.3	121.6	54.0							
au2_gt 5602: Model	17.5	355.3	87.4	normal score	0.994	0.521		0.472		Spherical		141.5	121.6	54.0							
au2_gt 5602: Model	17.5	355.3	87.4	data	0.002	0.002	0.637	0.001	0.359	Spherical		141.5	121.6	54.0							
au2_gt 5603: Model	17.5	355.3	77.8	normal score	0.988	0.266		0.721		Spherical		141.5	121.6	54.0							
au2_gt 5603: Model	17.5	355.3	77.8	data	0.003	0.001	0.366	0.002	0.626	Spherical		141.5	121.6	54.0							
au2_gt 5604: Model	17.5	355.3	85.0	normal score	0.981	0.336		0.644		Spherical		141.5	121.6	54.0							
au2_gt 5604: Model	17.5	355.3	85.0	data	0.002	0.001	0.447	0.001	0.541	Spherical		141.5	121.6	54.0							
au2_gt 5605: Model	17.5	355.3	92.3	normal score	0.985	0.136		0.847		Spherical		141.5	121.6	54.0							
au2_gt 5605: Model	17.5	355.3	92.3	data	0.004	0.001	0.200	0.003	0.790	Spherical		141.5	121.6	54.0							
au2_gt 5901_5902: Model	19.6	324.9	65.3	data	0.001	0.000	0.256	0.000	0.215	Spherical		62.4	29.2	10.1	0.001	0.525	Spherical		220.5	165.5	126.5
au2_gt 5901_5902: Model	19.6	324.9	65.3	normal score	0.999	0.178		0.105		Spherical		62.4	29.2	10.1	0.711		Spherical		220.5	165.5	126.5
au2_gt 6900_7500: Model	6.1	31.7	23.4	normal score	0.999	0.138		0.148		Spherical		62.4	29.2	15.3	0.711		Spherical		252.3	203.3	136.2
au2_gt 6900_7500: Model	6.1	31.7	23.4	data	0.006	0.001	0.203	0.002	0.263	Spherical		62.4	29.2	15.3	0.003	0.533	Spherical		252.3	203.3	136.2
au2_gt 7900: Model	74.2	77.7	6.7	normal score	0.999	0.321		0.069		Spherical		30.0	24.4	40.0	0.608		Spherical		278.9	401.9	122.7
au2_gt 7900: Model	74.2	77.7	6.7	data	0.003	0.001	0.430	0.000	0.100	Spherical		30.0	24.4	40.0	0.001	0.469	Spherical		278.9	401.9	122.7
au2_gt 8900: Model	15.5	205.0	63.8	data	0.000	0.000	0.354	0.000	0.380	Spherical		30.0	25.0	14.2	0.000	0.266	Spherical		278.9	227.2	40.6

table continues…

General	Direction							Structure 1							Structure 2						
Variogram Name	Dip	Dip Azimuth	Pitch	Model Space	Variance	Nugget	Normalised Nugget	Sill	Normalised Sill	Structure	Alpha	Major	Semi-major	Minor	Sill	Normalised Sill	Structure	Alpha	Major	Semi-major	Minor
au2_gt 8900: Model	15.5	205.0	63.8	normal score	0.999	0.256		0.364		Spherical		30.0	25.0	14.2	0.379		Spherical		278.9	227.2	40.6
au2_gt 9900: Model	15.5	205.0	63.8	normal score	0.999	0.256		0.262		Spherical		30.0	25.0	18.0	0.478		Spherical		278.9	227.2	112.8
au2_gt 9900: Model	15.5	205.0	63.8	data	0.000	0.000	0.354	0.000	0.294	Spherical		30.0	25.0	18.0	0.000	0.350	Spherical		278.9	227.2	112.8
cu_pct 1000: Model	27.7	249.8	65.2	normal score	0.997	0.690		0.308		Spherical		82.7	49.2	25.5							
cu_pct 1000: Model	27.7	249.8	65.2	data	0.023	0.018	0.784	0.005	0.215	Spherical		82.7	49.2	25.5							
cu_pct 11_12_13_16: Model	34.6	250.2	12.8	normal score	1.000	0.138		0.299		Spherical		94.7	41.4	22.4	0.563		Spherical		306.9	341.7	177.5
cu_pct 11_12_13_16: Model	17.0	288.9	138.0	data	0.032	0.006	0.203	0.011	0.344	Spherical		62.4	29.2	25.5	0.012	0.376	Spherical		269.2	203.3	131.7
cu_pct 11_12_13_16: Model	17.0	288.9	138.0	normal score	1.000	0.138		0.260		Spherical		62.4	29.2	25.5	0.481		Spherical		269.2	203.3	131.7
cu_pct 11_12_13_16: Model	34.6	250.2	12.8	data	0.032	0.006	0.203	0.013	0.405	Spherical		94.7	41.4	22.4	0.012	0.392	Spherical		306.9	341.7	177.5
cu_pct 1900: Model	17.0	288.9	138.0	normal score	0.999	0.278		0.201		Spherical		54.2	29.2	14.2	0.519		Spherical		266.4	192.5	75.9
cu_pct 1900: Model	87.8	262.8	167.3	normal score	0.999	0.300		0.249		Spherical		94.7	39.0	44.9	0.449		Spherical		316.9	185.3	108.0
cu_pct 1900: Model	87.8	262.8	167.3	data	0.003	0.001	0.407	0.001	0.288	Spherical		94.7	39.0	44.9	0.001	0.305	Spherical		316.9	185.3	108.0
cu_pct 1900: Model	17.0	288.9	138.0	data	0.003	0.001	0.381	0.001	0.247	Spherical		54.2	29.2	14.2	0.001	0.372	Spherical		266.4	192.5	75.9
cu_pct 2400: Model	17.0	288.9	138.0	normal score	0.999	0.324		0.228		Spherical		53.1	29.2	26.6	0.441		Spherical		188.0	109.4	146.7
cu_pct 2400: Model	17.0	288.9	138.0	data	0.165	0.072	0.434	0.043	0.260	Spherical		53.1	29.2	26.6	0.050	0.302	Spherical		188.0	109.4	146.7
cu_pct 24_26: Model	67.4	88.3	65.8	normal score	1.000	0.240		0.485		Spherical		42.8	77.0	35.4	0.275		Spherical		119.8	102.9	74.2
cu_pct 24_26: Model	67.4	88.3	65.8	data	0.098	0.033	0.335	0.048	0.489	Spherical		42.8	77.0	35.4	0.017	0.176	Spherical		119.8	102.9	74.2
cu_pct 2601: Model	17.0	288.9	138.0	data	0.017	0.007	0.434	0.003	0.179	Spherical		62.4	29.2	14.3	0.007	0.384	Spherical		206.4	166.9	139.0
cu_pct 2601: Model	17.0	288.9	138.0	normal score	0.999	0.324		0.122		Spherical		62.4	29.2	14.3	0.549		Spherical		206.4	166.9	139.0
cu_pct 2602: Model	83.8	79.8	4.6	normal score	0.998	0.324		0.121		Spherical		62.4	29.2	14.3	0.544		Spherical		206.4	166.9	139.0
cu_pct 2602: Model	83.8	79.8	4.6	data	0.052	0.023	0.431	0.009	0.178	Spherical		62.4	29.2	14.3	0.020	0.383	Spherical		206.4	166.9	139.0
cu_pct 2900: Model	11.5	344.8	67.1	data	0.002	0.001	0.380	0.000	0.247	Spherical		54.2	29.2	14.2	0.001	0.373	Spherical		266.4	192.5	64.5

table continues…

General	Direction							Structure 1							Structure 2						
Variogram Name	Dip	Dip Azimuth	Pitch	Model Space	Variance	Nugget	Normalised Nugget	Sill	Normalised Sill	Structure	Alpha	Major	Semi-major	Minor	Sill	Normalised Sill	Structure	Alpha	Major	Semi-major	Minor
cu_pct 2900: Model	11.5	344.8	67.1	normal score	0.999	0.278		0.201		Spherical		54.2	29.2	14.2	0.522		Spherical		266.4	192.5	64.5
cu_pct 32_33_36: Model	44.9	73.7	174.3	data	0.042	0.009	0.203	0.018	0.427	Spherical		19.6	29.2	43.1	0.016	0.368	Spherical		278.9	227.2	152.0
cu_pct 32_33_36: Model	44.9	73.7	174.3	normal score	1.000	0.138		0.365		Spherical		19.6	29.2	43.1	0.494		Spherical		278.9	227.2	152.0
cu_pct 3500: Model	21.4	231.4	32.6	data	0.009	0.003	0.380	0.002	0.184	Spherical		30.0	29.2	19.3	0.004	0.434	Spherical		213.9	192.5	92.8
cu_pct 3500: Model	21.4	231.4	32.6	normal score	0.999	0.278		0.141		Spherical		30.0	29.2	19.3	0.578		Spherical		213.9	192.5	92.8
cu_pct 3900: Model	44.3	85.5	170.1	normal score	0.999	0.138		0.097		Spherical		30.2	42.8	21.6	0.766		Spherical		300.2	300.2	95.4
cu_pct 3900: Model	44.3	85.5	170.1	data	0.003	0.001	0.203	0.001	0.165	Spherical		30.2	42.8	21.6	0.002	0.633	Spherical		300.2	300.2	95.4
cu_pct 4601: Model	87.9	281.2	0.8	normal score	0.999	0.378		0.311		Spherical		62.4	17.5	25.5	0.298		Spherical		211.9	116.3	60.5
cu_pct 4601: Model	87.9	281.2	0.8	data	0.010	0.005	0.493	0.003	0.303	Spherical		62.4	17.5	25.5	0.002	0.196	Spherical		211.9	116.3	60.5
cu_pct 4901: Model	78.8	95.6	176.7	data	0.001	0.000	0.296	0.001	0.503	Spherical		62.4	17.5	25.5	0.000	0.198	Spherical		162.1	150.8	131.7
cu_pct 4901: Model	78.8	95.6	176.7	normal score	0.999	0.209		0.479		Spherical		62.4	17.5	25.5	0.307		Spherical		162.1	150.8	131.7
cu_pct 5601: Model	17.5	355.3	99.1	normal score	0.997	0.138		0.858		Spherical		151.3	121.6	54.0							
cu_pct 5601: Model	17.5	355.3	99.1	data	0.020	0.004	0.202	0.016	0.795	Spherical		151.3	121.6	54.0							
cu_pct 5602: Model	17.5	355.3	87.4	normal score	0.994	0.137		0.855		Spherical		141.5	121.6	54.0							
cu_pct 5602: Model	17.5	355.3	87.4	data	0.017	0.003	0.202	0.014	0.794	Spherical		141.5	121.6	54.0							
cu_pct 5603: Model	17.5	355.3	77.8	normal score	0.988	0.136		0.850		Spherical		141.5	121.6	54.0							
cu_pct 5603: Model	17.5	355.3	77.8	data	0.004	0.001	0.201	0.004	0.791	Spherical		141.5	121.6	54.0							
cu_pct 5604: Model	17.5	355.3	85.0	normal score	0.981	0.135		0.834		Spherical		141.5	121.6	54.0							
cu_pct 5604: Model	17.5	355.3	85.0	data	0.005	0.001	0.197	0.004	0.783	Spherical		141.5	121.6	54.0							
cu_pct 5605: Model	17.5	355.3	92.3	normal score	0.985	0.136		0.846		Spherical		141.5	121.6	54.0							
cu_pct 5605: Model	17.5	355.3	92.3	data	0.013	0.003	0.200	0.010	0.789	Spherical		141.5	121.6	54.0							
cu_pct 5901_5902: Model	22.2	311.6	151.5	normal score	0.999	0.154		0.411		Spherical		94.7	39.0	24.1	0.436		Spherical		316.9	136.1	65.7

table continues…

General	Direction							Structure 1							Structure 2						
Variogram Name	Dip	Dip Azimuth	Pitch	Model Space	Variance	Nugget	Normalised Nugget	Sill	Normalised Sill	Structure	Alpha	Major	Semi-major	Minor	Sill	Normalised Sill	Structure	Alpha	Major	Semi-major	Minor
cu_pct 5901_5902: Model	19.6	324.9	110.1	normal score	0.999	0.138		0.278		Spherical		62.4	29.2	12.2	0.578		Spherical		252.3	203.3	64.4
cu_pct 5901_5902: Model	19.6	324.9	110.1	data	0.002	0.000	0.203	0.001	0.377	Spherical		62.4	29.2	12.2	0.001	0.417	Spherical		252.3	203.3	64.4
cu_pct 5901_5902: Model	22.2	311.6	151.5	data	0.002	0.001	0.225	0.001	0.482	Spherical		94.7	39.0	24.1	0.001	0.294	Spherical		316.9	136.1	65.7
cu_pct 6900_7500: Model	6.1	31.7	23.4	normal score	0.999	0.138		0.248		Spherical		62.4	29.2	26.3	0.611		Spherical		252.3	203.3	134.7
cu_pct 6900_7500: Model	6.1	31.7	23.4	data	0.008	0.002	0.203	0.003	0.351	Spherical		62.4	29.2	26.3	0.004	0.444	Spherical		252.3	203.3	134.7
cu_pct 7900: Model	74.2	77.7	6.7	normal score	0.999	0.152		0.351		Spherical		30.0	30.0	40.0	0.491		Spherical		278.9	227.2	122.7
cu_pct 7900: Model	74.2	77.7	6.7	data	0.005	0.001	0.221	0.002	0.413	Spherical		30.0	30.0	40.0	0.002	0.362	Spherical		278.9	227.2	122.7
cu_pct 8900: Model	15.5	205.0	63.8	normal score	0.999	0.256		0.239		Spherical		30.0	25.0	17.0	0.505		Spherical		278.9	227.2	78.8
cu_pct 8900: Model	15.5	205.0	63.8	data	0.002	0.001	0.354	0.001	0.274	Spherical		30.0	25.0	17.0	0.001	0.372	Spherical		278.9	227.2	78.8
cu_pct 9900: Model	15.5	205.0	63.8	normal score	0.999	0.256		0.239		Spherical		30.0	25.0	17.0	0.505		Spherical		278.9	227.2	78.8
cu_pct 9900: Model	15.5	205.0	63.8	data	0.002	0.001	0.354	0.000	0.274	Spherical		30.0	25.0	17.0	0.001	0.372	Spherical		278.9	227.2	78.8
cu_pct in GM_FromMeshes: 2601: Model	64.5	239.2	167.0	normal score	0.999	0.340		0.320		Spherical		200.0	60.0	35.0	0.340		Spherical		1140.0	450.0	80.0
cu_pct in GM_FromMeshes: 3200: Model	88.7	90.5	153.9	normal score	1.000	0.100		0.520		Spherical		46.8	16.0	55.0	0.380		Spherical		425.0	290.0	110.0
cu_pct in GM_FromMeshes: 3301: Model	45.1	81.4	168.6	data	0.023	0.003	0.143	0.010	0.412	Spherical		50.0	85.0	25.0	0.010	0.446	Spherical		255.0	265.0	110.0
mo_pct 1000: Model	27.7	249.8	65.2	data	0.000	0.000	0.784	0.000	0.215	Spherical		82.7	49.2	25.5							
mo_pct 1000: Model	27.7	249.8	65.2	normal score	0.997	0.690		0.308		Spherical		82.7	49.2	25.5							
mo_pct 11_12_13_16: Model	17.0	288.9	154.3	normal score	1.000	0.168		0.193		Spherical		33.3	35.7	17.8	0.640		Spherical		303.8	240.2	151.7
mo_pct 11_12_13_16: Model	17.0	288.9	154.3	data	0.000	0.000	0.243	0.000	0.259	Spherical		33.3	35.7	17.8	0.000	0.498	Spherical		303.8	240.2	151.7
mo_pct 1900: Model	17.0	288.9	138.0	data	0.000	0.000	0.215	0.000	0.471	Spherical		56.2	39.2	32.2	0.000	0.316	Spherical		222.4	192.5	96.3
mo_pct 1900: Model	17.0	288.9	138.0	normal score	0.999	0.147		0.397		Spherical		56.2	39.2	32.2	0.458		Spherical		222.4	192.5	96.3
mo_pct 2400: Model	17.0	288.9	147.5	data	0.001	0.000	0.325	0.000	0.306	Spherical		54.0	45.4	37.4	0.000	0.368	Spherical		158.5	134.7	165.0
mo_pct 2400: Model	17.0	288.9	147.5	normal score	0.999	0.233		0.230		Spherical		54.0	45.4	37.4	0.537		Spherical		158.5	134.7	165.0

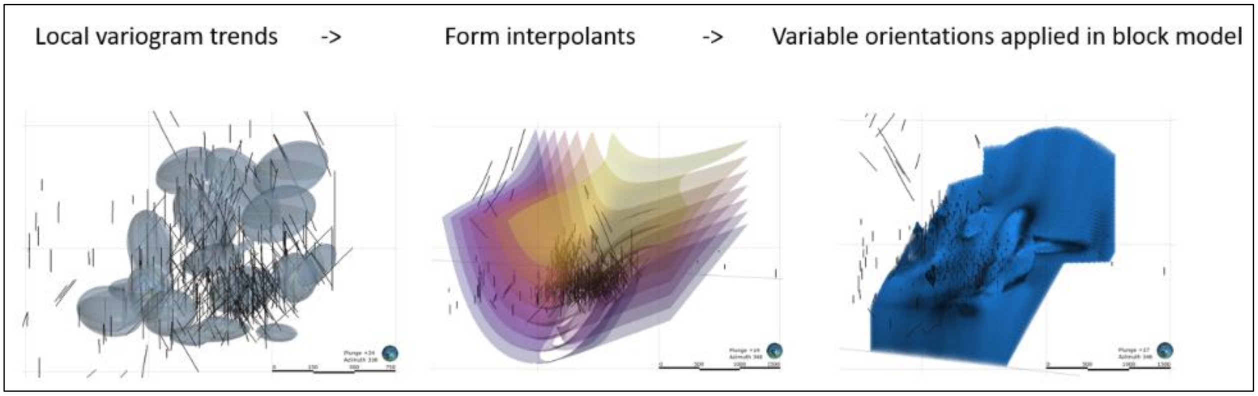
table continues…

General	Direction							Structure 1							Structure 2						
Variogram Name	Dip	Dip Azimuth	Pitch	Model Space	Variance	Nugget	Normalised Nugget	Sill	Normalised Sill	Structure	Alpha	Major	Semi-major	Minor	Sill	Normalised Sill	Structure	Alpha	Major	Semi-major	Minor
mo_pct 2601: Model	17.0	288.9	154.2	data	0.000	0.000	0.176	0.000	0.335	Spherical		62.4	16.2	28.3	0.000	0.490	Spherical		186.3	129.8	154.7
mo_pct 2601: Model	17.0	288.9	154.2	normal score	0.999	0.118		0.195		Spherical		62.4	16.2	28.3	0.688		Spherical		186.3	129.8	154.7
mo_pct 2602: Model	83.8	79.8	4.6	data	0.000	0.000	0.433	0.000	0.179	Spherical		62.4	29.2	14.3	0.000	0.384	Spherical		206.4	166.9	139.0
mo_pct 2602: Model	83.8	79.8	4.6	normal score	0.998	0.324		0.122		Spherical		62.4	29.2	14.3	0.548		Spherical		206.4	166.9	139.0
mo_pct 2900: Model	11.5	344.8	67.1	normal score	0.999	0.278		0.190		Spherical		54.2	29.2	49.8	0.528		Spherical		303.5	278.2	197.0
mo_pct 2900: Model	11.5	344.8	67.1	data	0.000	0.000	0.380	0.000	0.234	Spherical		54.2	29.2	49.8	0.000	0.384	Spherical		303.5	278.2	197.0
mo_pct 32_33_36: Model	44.9	73.7	69.1	data	0.001	0.000	0.229	0.000	0.332	Spherical		22.6	29.2	32.9	0.000	0.437	Spherical		181.6	177.1	175.4
mo_pct 32_33_36: Model	44.9	73.7	69.1	normal score	1.000	0.157		0.261		Spherical		22.6	29.2	32.9	0.577		Spherical		181.6	177.1	175.4
mo_pct 3500: Model	21.4	231.4	24.9	normal score	0.999	0.126		0.173		Spherical		30.0	92.5	38.6	0.701		Spherical		393.0	346.3	209.2
mo_pct 3500: Model	21.4	231.4	24.9	data	0.000	0.000	0.186	0.000	0.243	Spherical		30.0	92.5	38.6	0.000	0.571	Spherical		393.0	346.3	209.2
mo_pct 3900: Model	44.3	85.5	170.1	normal score	0.999	0.227		0.134		Spherical		30.2	9.2	8.5	0.640		Spherical		138.1	221.8	58.5
mo_pct 3900: Model	44.3	85.5	170.1	data	0.000	0.000	0.319	0.000	0.209	Spherical		30.2	9.2	8.5	0.000	0.473	Spherical		138.1	221.8	58.5
mo_pct 4601: Model	87.9	281.2	14.7	normal score	0.999	0.188		0.183		Spherical		62.4	12.0	30.4	0.630		Spherical		216.7	154.2	85.3
mo_pct 4601: Model	87.9	281.2	14.7	data	0.000	0.000	0.270	0.000	0.281	Spherical		62.4	12.0	30.4	0.000	0.451	Spherical		216.7	154.2	85.3
mo_pct 4901: Model	78.8	95.6	23.7	data	0.000	0.000	0.296	0.000	0.252	Spherical		62.4	32.4	25.5	0.000	0.451	Spherical		209.6	185.8	89.7
mo_pct 4901: Model	78.8	95.6	23.7	normal score	0.999	0.209		0.158		Spherical		62.4	32.4	25.5	0.631		Spherical		209.6	185.8	89.7
mo_pct 5601: Model	17.5	355.3	109.8	normal score	0.997	0.152		0.844		Spherical		151.3	121.6	54.0							
mo_pct 5601: Model	17.5	355.3	109.8	data	0.000	0.000	0.221	0.000	0.776	Spherical		151.3	121.6	54.0							
mo_pct 5602: Model	17.5	355.3	108.2	data	0.000	0.000	0.235	0.000	0.761	Spherical		141.5	121.6	54.0							
mo_pct 5602: Model	17.5	355.3	108.2	normal score	0.994	0.162		0.831		Spherical		141.5	121.6	54.0							
mo_pct 5603: Model	17.5	355.3	77.8	data	0.000	0.000	0.136	0.000	0.856	Spherical		141.5	121.6	54.0							
mo_pct 5603: Model	17.5	355.3	77.8	normal score	0.988	0.090		0.897		Spherical		141.5	121.6	54.0							

table continues…

General	Direction							Structure 1							Structure 2						
Variogram Name	Dip	Dip Azimuth	Pitch	Model Space	Variance	Nugget	Normalised Nugget	Sill	Normalised Sill	Structure	Alpha	Major	Semi-major	Minor	Sill	Normalised Sill	Structure	Alpha	Major	Semi-major	Minor
mo_pct 5604: Model	17.5	355.3	85.0	data	0.005	0.001	0.197	0.004	0.783	Spherical		141.5	121.6	54.0							
mo_pct 5604: Model	17.5	355.3	85.0	normal score	0.981	0.135		0.834		Spherical		141.5	121.6	54.0							
mo_pct 5605: Model	17.5	355.3	92.3	normal score	0.985	0.294		0.689		Spherical		141.5	121.6	54.0							
mo_pct 5605: Model	17.5	355.3	92.3	data	0.000	0.000	0.400	0.000	0.590	Spherical		141.5	121.6	54.0							
mo_pct 5901_5902: Model	19.6	324.9	110.1	data	0.000	0.000	0.348	0.000	0.280	Spherical		62.4	29.2	8.9	0.000	0.371	Spherical		252.3	203.3	127.9
mo_pct 5901_5902: Model	19.6	324.9	110.1	normal score	0.999	0.251		0.222		Spherical		62.4	29.2	8.9	0.525		Spherical		252.3	203.3	127.9
mo_pct 6900_7500: Model	6.1	31.7	23.4	data	0.000	0.000	0.203	0.000	0.351	Spherical		62.4	29.2	26.3	0.000	0.444	Spherical		252.3	203.3	134.7
mo_pct 6900_7500: Model	6.1	31.7	23.4	normal score	0.999	0.138		0.248		Spherical		62.4	29.2	26.3	0.611		Spherical		252.3	203.3	134.7
mo_pct 7900: Model	74.2	77.7	6.7	data	0.000	0.000	0.221	0.000	0.438	Spherical		30.0	28.6	40.0	0.000	0.340	Spherical		278.9	275.9	122.7
mo_pct 7900: Model	74.2	77.7	6.7	normal score	0.999	0.152		0.379		Spherical		30.0	28.6	40.0	0.467		Spherical		278.9	275.9	122.7
mo_pct 8900: Model	15.5	205.0	63.8	normal score	0.999	0.447		0.346		Spherical		30.0	25.0	21.1	0.207		Spherical		278.9	227.2	63.1
mo_pct 8900: Model	15.5	205.0	63.8	data	0.000	0.000	0.565	0.000	0.300	Spherical		30.0	25.0	21.1	0.000	0.135	Spherical		278.9	227.2	63.1
mo_pct 9900: Model	15.5	205.0	63.8	data	0.000	0.000	0.418	0.000	0.209	Spherical		30.0	25.0	17.0	0.000	0.373	Spherical		278.9	227.2	78.8
mo_pct 9900: Model	15.5	205.0	63.8	normal score	0.999	0.310		0.184		Spherical		30.0	25.0	17.0	0.505		Spherical		278.9	227.2	78.8
mo_pct Blk1: Model	26.0	262.0	107.8	normal score	0.999	0.146		0.419		Spherical		25.7	32.8	39.4	0.431		Spherical		139.3	160.5	162.9
mo_pct Blk1: Model	26.0	262.0	107.8	data	2.069	0.443	0.214	0.997	0.482	Spherical		25.7	32.8	39.4	0.625	0.302	Spherical		139.3	160.5	162.9
mo_pct: Model	5.3	225.6	35.8	data	0.000	0.000	0.062	0.000	0.264	Spherical		28.8	26.0	14.3	0.000	0.528	Spherical		408.9	227.1	177.6
mo_pct: Model	5.3	225.6	35.8	normal score	1.000	0.053		0.222		Spherical		28.8	26.0	14.3	0.536		Spherical		408.9	227.1	177.6

Figure 14-4: Development of Local Orientation Model



Source: Red Pennant (2021)

14.8 Estimation/Interpolation Methods

Copper, molybdenum, silver, and gold were estimated using ordinary kriging (OK) with inverse distance and NN estimates generated for validation (see Table 14-8). he estimation was done in a single pass. Search ellipses for copper, molybdenum, silver, and gold interpolation assumed ranges up to twice the maximum range of the variogram for the second structure. A block discretization of 5 x 5 x 3 steps (X, Y, Z) was used. Search parameters are shown in Table 14-9.

Table 14-8: Grade Interpolant Parameters

General				Value Capping		
Name	Domain	Values	Estimate Name	Lower Bound	Upper Bound	Estimate Type
ID, ag2_gt 1000	GM_Consolidated_from_meshes: 1000	ag2_gt	ag2_gt 1000	0.001	2.500	IDW
ID, ag2_gt 11_12_13_16	GM_Consolidated_from_meshes: 11_12_13_16	ag2_gt	ag2_gt 11_12_13_16	0.001	20.000	IDW
ID, ag2_gt 1900	GM_FromMeshes: 1900	ag2_gt	ag2_gt 1900	0.001	2.000	IDW
ID, ag2_gt 2400	GM_FromMeshes: 2400	ag2_gt	ag2_gt 2400	0.001	12.000	IDW
ID, ag2_gt 2601	GM_FromMeshes: 2601	ag2_gt	ag2_gt 2601	0.001	3.000	IDW
ID, ag2_gt 2602	GM_FromMeshes: 2602	ag2_gt	ag2_gt 2602	0.001	6.000	IDW
ID, ag2_gt 2900	GM_FromMeshes: 2900	ag2_gt	ag2_gt 2900	0.001	1.100	IDW
ID, ag2_gt 32_33_36	GM_Consolidated_from_meshes: 32_33_36	ag2_gt	ag2_gt 32_33_36	0.001	30.000	IDW
ID, ag2_gt 3500	GM_FromMeshes: 3500	ag2_gt	ag2_gt 3500	0.001	2.000	IDW
ID, ag2_gt 3900	GM_Consolidated_from_meshes: 3900	ag2_gt	ag2_gt 3900	0.001	10.000	IDW
ID, ag2_gt 4601	GM_FromMeshes: 4601	ag2_gt	ag2_gt 4601	0.001	2.000	IDW
ID, ag2_gt 4901	GM_FromMeshes: 4901	ag2_gt	ag2_gt 4901	0.001	1.000	IDW
ID, ag2_gt 5601	GM_FromMeshes: 5601	ag2_gt	ag2_gt 5601	0.001	3.000	IDW
ID, ag2_gt 5602	GM_FromMeshes: 5602	ag2_gt	ag2_gt 5602	0.001	0.900	IDW
ID, ag2_gt 5603	GM_FromMeshes: 5603	ag2_gt	ag2_gt 5603	0.001	1.236	IDW
ID, ag2_gt 5604	GM_FromMeshes: 5604	ag2_gt	ag2_gt 5604	0.001	4.866	IDW
ID, ag2_gt 5605	GM_FromMeshes: 5605	ag2_gt	ag2_gt 5605			IDW
ID, ag2_gt 5901_5902	GM_Consolidated_from_meshes: 5901	ag2_gt	ag2_gt 5901_5902	0.001	4.000	IDW

table continues…

General			Estimate Name	Value Capping		Estimate Type
Name	Domain	Values		Lower Bound	Upper Bound	
ID, ag2_gt 6900_7500	GM_Consolidated_from_meshes: 6900_7500	ag2_gt	ag2_gt 6900_7500	0.001	3.000	IDW
ID, ag2_gt 7900	GM_Consolidated_from_meshes: 7900	ag2_gt	ag2_gt 7900	0.001	2.000	IDW
ID, ag2_gt 8900	GM_FromMeshes: 8900	ag2_gt	ag2_gt 8900	0.001	4.000	IDW
ID, ag2_gt 9900	GM_FromMeshes: 9900	ag2_gt	ag2_gt 9900	0.001	6.000	IDW
ID, au2_gt 1000	GM_Consolidated_from_meshes: 1000	au2_gt	au2_gt 1000	0.001	0.600	IDW
ID, au2_gt 11_12_13_16	GM_Consolidated_from_meshes: 11_12_13_16	au2_gt	au2_gt 11_12_13_16	0.001	3.000	IDW
ID, au2_gt 1900	GM_FromMeshes: 1900	au2_gt	au2_gt 1900	0.001	0.270	IDW
ID, au2_gt 2400	GM_FromMeshes: 2400	au2_gt	au2_gt 2400	0.001	1.000	IDW
ID, au2_gt 2601	GM_FromMeshes: 2601	au2_gt	au2_gt 2601	0.001	0.300	IDW
ID, au2_gt 2602	GM_FromMeshes: 2602	au2_gt	au2_gt 2602	0.001	0.700	IDW
ID, au2_gt 2900	GM_FromMeshes: 2900	au2_gt	au2_gt 2900	0.001	0.120	IDW
ID, au2_gt 32_33_36	GM_Consolidated_from_meshes: 32_33_36	au2_gt	au2_gt 32_33_36	0.001	3.000	IDW
ID, au2_gt 3500	GM_FromMeshes: 3500	au2_gt	au2_gt 3500	0.001	0.400	IDW
ID, au2_gt 3900	GM_Consolidated_from_meshes: 3900	au2_gt	au2_gt 3900	0.001	0.250	IDW
ID, au2_gt 4601	GM_FromMeshes: 4601	au2_gt	au2_gt 4601	0.001	0.250	IDW
ID, au2_gt 4901	GM_FromMeshes: 4901	au2_gt	au2_gt 4901	0.001	0.200	IDW
ID, au2_gt 5601	GM_FromMeshes: 5601	au2_gt	au2_gt 5601	0.001	0.300	IDW
ID, au2_gt 5602	GM_FromMeshes: 5602	au2_gt	au2_gt 5602	0.001	0.200	IDW
ID, au2_gt 5603	GM_FromMeshes: 5603	au2_gt	au2_gt 5603	0.001	0.150	IDW

table continues…

General			Estimate Name	Value Capping		Estimate Type
Name	Domain	Values		Lower Bound	Upper Bound	
ID, au2_gt 5604	GM_FromMeshes: 5604	au2_gt	au2_gt 5604	0.001	0.140	IDW
ID, au2_gt 5605	GM_FromMeshes: 5605	au2_gt	au2_gt 5605	0.001	0.250	IDW
ID, au2_gt 5901_5902	GM_Consolidated_from_meshes: 5901	au2_gt	au2_gt 5901_5902	0.001	0.250	IDW
ID, au2_gt 6900_7500	GM_Consolidated_from_meshes: 6900_7500	au2_gt	au2_gt 6900_7500	0.001	0.300	IDW
ID, au2_gt 7900	GM_Consolidated_from_meshes: 7900	au2_gt	au2_gt 7900	0.001	0.200	IDW
ID, au2_gt 8900	GM_FromMeshes: 8900	au2_gt	au2_gt 8900	0.001	0.100	IDW
ID, au2_gt 9900	GM_FromMeshes: 9900	au2_gt	au2_gt 9900	0.001	0.100	IDW
ID, cu_pct 1000	GM_Consolidated_from_meshes: 1000	cu_pct	cu_pct 1000	0.001	0.500	IDW
ID, cu_pct 11_12_13_16	GM_Consolidated_from_meshes: 11_12_13_16	cu_pct	cu_pct 11_12_13_16	0.001	1.300	IDW
ID, cu_pct 1900	GM_FromMeshes: 1900	cu_pct	cu_pct 1900	0.001	0.383	IDW
ID, cu_pct 2400	GM_FromMeshes: 2400	cu_pct	cu_pct 2400	0.001	2.000	IDW
ID, cu_pct 2601	GM_FromMeshes: 2601	cu_pct	cu_pct 2601	0.001	0.650	IDW
ID, cu_pct 2602	GM_FromMeshes: 2602	cu_pct	cu_pct 2602	0.001	2.000	IDW
ID, cu_pct 2900	GM_FromMeshes: 2900	cu_pct	cu_pct 2900	0.001	0.310	IDW
ID, cu_pct 32_33_36	GM_Consolidated_from_meshes: 32_33_36	cu_pct	cu_pct 32_33_36	0.001	1.300	IDW
ID, cu_pct 3500	GM_FromMeshes: 3500	cu_pct	cu_pct 3500	0.001	0.310	IDW
ID, cu_pct 3900	GM_Consolidated_from_meshes: 3900	cu_pct	cu_pct 3900	0.001	0.320	IDW
ID, cu_pct 4601	GM_FromMeshes: 4601	cu_pct	cu_pct 4601	0.001	0.600	IDW
ID, cu_pct 4901	GM_FromMeshes: 4901	cu_pct	cu_pct 4901	0.001	0.184	IDW

table continues…

General				Value Capping		
Name	Domain	Values	Estimate Name	Lower Bound	Upper Bound	Estimate Type
ID, cu_pct 5601	GM_FromMeshes: 5601	cu_pct	cu_pct 5601	0.001	1.090	IDW
ID, cu_pct 5602	GM_FromMeshes: 5602	cu_pct	cu_pct 5602	0.001	0.658	IDW
ID, cu_pct 5603	GM_FromMeshes: 5603	cu_pct	cu_pct 5603	0.001	0.246	IDW
ID, cu_pct 5604	GM_FromMeshes: 5604	cu_pct	cu_pct 5604	0.001	0.216	IDW
ID, cu_pct 5605	GM_FromMeshes: 5605	cu_pct	cu_pct 5605	0.001	0.495	IDW
ID, cu_pct 5901_5902	GM_Consolidated_from_meshes: 5901	cu_pct	cu_pct 5901_5902	0.001	0.400	IDW
ID, cu_pct 6900_7500	GM_Consolidated_from_meshes: 6900_7500	cu_pct	cu_pct 6900_7500	0.001	0.400	IDW
ID, cu_pct 7900	GM_Consolidated_from_meshes: 7900	cu_pct	cu_pct 7900	0.001	0.310	IDW
ID, cu_pct 8900	GM_FromMeshes: 8900	cu_pct	cu_pct 8900	0.001	0.200	IDW
ID, cu_pct 9900	GM_FromMeshes: 9900	cu_pct	cu_pct 9900	0.001	0.200	IDW
ID, mo_pct 1000	GM_Consolidated_from_meshes: 1000	mo_pct	mo_pct 1000	0.001	0.030	IDW
ID, mo_pct 11_12_13_16	GM_Consolidated_from_meshes: 11_12_13_16	mo_pct	mo_pct 11_12_13_16	0.001	0.300	IDW
ID, mo_pct 1900	GM_FromMeshes: 1900	mo_pct	mo_pct 1900	0.001	0.050	IDW
ID, mo_pct 2400	GM_FromMeshes: 2400	mo_pct	mo_pct 2400	0.001	0.200	IDW
ID, mo_pct 2601	GM_FromMeshes: 2601	mo_pct	mo_pct 2601	0.001	0.060	IDW
ID, mo_pct 2602	GM_FromMeshes: 2602	mo_pct	mo_pct 2602	0.001	2.000	IDW
ID, mo_pct 2900	GM_FromMeshes: 2900	mo_pct	mo_pct 2900	0.001	0.015	IDW
ID, mo_pct 32_33_36	GM_Consolidated_from_meshes: 32_33_36	mo_pct	mo_pct 32_33_36	0.001	0.200	IDW
ID, mo_pct 3500	GM_FromMeshes: 3500	mo_pct	mo_pct 3500	0.001	0.055	IDW

table continues…

General			Estimate Name	Value Capping		Estimate Type
Name	Domain	Values		Lower Bound	Upper Bound	
ID, mo_pct 3900	GM_Consolidated_from_meshes: 3900	mo_pct	mo_pct 3900	0.001	0.025	IDW
ID, mo_pct 4601	GM_FromMeshes: 4601	mo_pct	mo_pct 4601	0.001	0.050	IDW
ID, mo_pct 4901	GM_FromMeshes: 4901	mo_pct	mo_pct 4901	0.001	0.008	IDW
ID, mo_pct 5601	GM_FromMeshes: 5601	mo_pct	mo_pct 5601	0.001	0.040	IDW
ID, mo_pct 5602	GM_FromMeshes: 5602	mo_pct	mo_pct 5602	0.001	0.018	IDW
ID, mo_pct 5603	GM_FromMeshes: 5603	mo_pct	mo_pct 5603	0.001	0.014	IDW
ID, mo_pct 5604	GM_FromMeshes: 5604	mo_pct	mo_pct 5604	0.001	0.216	IDW
ID, mo_pct 5605	GM_FromMeshes: 5605	mo_pct	mo_pct 5605	0.001	0.008	IDW
ID, mo_pct 5901_5902	GM_Consolidated_from_meshes: 5901	mo_pct	mo_pct 5901_5902	0.001	0.030	IDW
ID, mo_pct 6900_7500	GM_Consolidated_from_meshes: 6900_7500	mo_pct	mo_pct 6900_7500	0.001	0.015	IDW
ID, mo_pct 7900	GM_Consolidated_from_meshes: 7900	mo_pct	mo_pct 7900	0.001	0.020	IDW
ID, mo_pct 8900	GM_FromMeshes: 8900	mo_pct	mo_pct 8900	0.001	0.015	IDW
ID, mo_pct 9900	GM_FromMeshes: 9900	mo_pct	mo_pct 9900	0.001	0.015	IDW
Kr, ag2_gt	GM_Consolidated_from_meshes: Boundary	ag2_gt	ag2_gt			K
Kr, ag2_gt 1000	GM_Consolidated_from_meshes: 1000	ag2_gt	ag2_gt 1000	0.001	2.500	K
Kr, ag2_gt 11_12_13_16	GM_Consolidated_from_meshes: 11_12_13_16	ag2_gt	ag2_gt 11_12_13_16	0.001	20.000	K
Kr, ag2_gt 1900	GM_FromMeshes: 1900	ag2_gt	ag2_gt 1900	0.001	2.000	K
Kr, ag2_gt 2400	GM_FromMeshes: 2400	ag2_gt	ag2_gt 2400	0.001	12.000	K
Kr, ag2_gt 2601	GM_FromMeshes: 2601	ag2_gt	ag2_gt 2601	0.001	3.000	K

table continues…

General				Value Capping		
Name	Domain	Values	Estimate Name	Lower Bound	Upper Bound	Estimate Type
Kr, ag2_gt 2602	GM_FromMeshes: 2602	ag2_gt	ag2_gt 2602	0.001	6.000	K
Kr, ag2_gt 2900	GM_FromMeshes: 2900	ag2_gt	ag2_gt 2900	0.001	1.100	K
Kr, ag2_gt 32_33_36	GM_Consolidated_from_meshes: 32_33_36	ag2_gt	ag2_gt 32_33_36	0.001	30.000	K
Kr, ag2_gt 3500	GM_FromMeshes: 3500	ag2_gt	ag2_gt 3500	0.001	2.000	K
Kr, ag2_gt 3900	GM_Consolidated_from_meshes: 3900	ag2_gt	ag2_gt 3900	0.001	10.000	K
Kr, ag2_gt 4601	GM_FromMeshes: 4601	ag2_gt	ag2_gt 4601	0.001	2.000	K
Kr, ag2_gt 4901	GM_FromMeshes: 4901	ag2_gt	ag2_gt 4901	0.001	1.000	K
Kr, ag2_gt 5601	GM_FromMeshes: 5601	ag2_gt	ag2_gt 5601	0.001	3.000	K
Kr, ag2_gt 5602	GM_FromMeshes: 5602	ag2_gt	ag2_gt 5602	0.001	0.900	K
Kr, ag2_gt 5603	GM_FromMeshes: 5603	ag2_gt	ag2_gt 5603	0.001	1.236	K
Kr, ag2_gt 5604	GM_FromMeshes: 5604	ag2_gt	ag2_gt 5604	0.001	4.866	K
Kr, ag2_gt 5605	GM_FromMeshes: 5605	ag2_gt	ag2_gt 5605			K
Kr, ag2_gt 5901_5902	GM_Consolidated_from_meshes: 5901	ag2_gt	ag2_gt 5901_5902	0.001	4.000	K
Kr, ag2_gt 6900_7500	GM_Consolidated_from_meshes: 6900_7500	ag2_gt	ag2_gt 6900_7500	0.001	3.000	K
Kr, ag2_gt 7900	GM_Consolidated_from_meshes: 7900	ag2_gt	ag2_gt 7900	0.001	2.000	K
Kr, ag2_gt 8900	GM_FromMeshes: 8900	ag2_gt	ag2_gt 8900	0.001	4.000	K
Kr, ag2_gt 9900	GM_FromMeshes: 9900	ag2_gt	ag2_gt 9900	0.001	6.000	K
Kr, au2_gt	GM_Consolidated_from_meshes: Boundary	au2_gt	au2_gt			K
Kr, au2_gt 1000	GM_Consolidated_from_meshes: 1000	au2_gt	au2_gt 1000	0.001	0.600	K

table continues…

General			Estimate Name	Value Capping		Estimate Type
Name	Domain	Values		Lower Bound	Upper Bound	
Kr, au2_gt 11_12_13_16	GM_Consolidated_from_meshes: 11_12_13_16	au2_gt	au2_gt 11_12_13_16	0.001	3.000	K
Kr, au2_gt 1900	GM_FromMeshes: 1900	au2_gt	au2_gt 1900	0.001	0.270	K
Kr, au2_gt 2400	GM_FromMeshes: 2400	au2_gt	au2_gt 2400	0.001	1.000	K
Kr, au2_gt 2601	GM_FromMeshes: 2601	au2_gt	au2_gt 2601	0.001	0.300	K
Kr, au2_gt 2602	GM_FromMeshes: 2602	au2_gt	au2_gt 2602	0.001	0.700	K
Kr, au2_gt 2900	GM_FromMeshes: 2900	au2_gt	au2_gt 2900	0.001	0.120	K
Kr, au2_gt 32_33_36	GM_Consolidated_from_meshes: 32_33_36	au2_gt	au2_gt 32_33_36	0.001	3.000	K
Kr, au2_gt 3500	GM_FromMeshes: 3500	au2_gt	au2_gt 3500	0.001	0.400	K
Kr, au2_gt 3900	GM_Consolidated_from_meshes: 3900	au2_gt	au2_gt 3900	0.001	0.250	K
Kr, au2_gt 4601	GM_FromMeshes: 4601	au2_gt	au2_gt 4601	0.001	0.250	K
Kr, au2_gt 4901	GM_FromMeshes: 4901	au2_gt	au2_gt 4901	0.001	0.200	K
Kr, au2_gt 5601	GM_FromMeshes: 5601	au2_gt	au2_gt 5601	0.001	0.300	K
Kr, au2_gt 5602	GM_FromMeshes: 5602	au2_gt	au2_gt 5602	0.001	0.200	K
Kr, au2_gt 5603	GM_FromMeshes: 5603	au2_gt	au2_gt 5603	0.001	0.150	K
Kr, au2_gt 5604	GM_FromMeshes: 5604	au2_gt	au2_gt 5604	0.001	0.140	K
Kr, au2_gt 5605	GM_FromMeshes: 5605	au2_gt	au2_gt 5605	0.001	0.250	K
Kr, au2_gt 5901_5902	GM_Consolidated_from_meshes: 5901	au2_gt	au2_gt 5901_5902	0.001	0.250	K
Kr, au2_gt 6900_7500	GM_Consolidated_from_meshes: 6900_7500	au2_gt	au2_gt 6900_7500	0.001	0.300	K
Kr, au2_gt 7900	GM_Consolidated_from_meshes: 7900	au2_gt	au2_gt 7900	0.001	0.200	K

table continues…

General				Value Capping		
Name	Domain	Values	Estimate Name	Lower Bound	Upper Bound	Estimate Type
Kr, au2_gt 8900	GM_FromMeshes: 8900	au2_gt	au2_gt 8900	0.001	0.100	K
Kr, au2_gt 9900	GM_FromMeshes: 9900	au2_gt	au2_gt 9900	0.001	0.100	K
Kr, au2_gt Blk1	GM_Faults fault block 1: Unknown	au2_gt	au2_gt Blk1	0.001	20.000	K
Kr, cu_pct 1000	GM_Consolidated_from_meshes: 1000	cu_pct	cu_pct 1000	0.001	0.500	K
Kr, cu_pct 11_12_13_16	GM_Consolidated_from_meshes: 11_12_13_16	cu_pct	cu_pct 11_12_13_16	0.001	1.300	K
Kr, cu_pct 1900	GM_FromMeshes: 1900	cu_pct	cu_pct 1900	0.001	0.383	K
Kr, cu_pct 2400	GM_FromMeshes: 2400	cu_pct	cu_pct 2400	0.001	2.000	K
Kr, cu_pct 24_26	GM_Consolidated_from_meshes: 24_26	cu_pct	cu_pct 24_26			K
Kr, cu_pct 2601	GM_FromMeshes: 2601	cu_pct	cu_pct 2601	0.001	0.650	K
Kr, cu_pct 2601 P1	GM_FromMeshes: 2601	cu_pct	cu_pct in GM_FromMeshes: 2601			K
Kr, cu_pct 2601 P2	GM_FromMeshes: 2601	cu_pct	cu_pct in GM_FromMeshes: 2601			K
Kr, cu_pct 2601 P3	GM_FromMeshes: 2601	cu_pct	cu_pct in GM_FromMeshes: 2601			K
Kr, cu_pct 2602	GM_FromMeshes: 2602	cu_pct	cu_pct 2602	0.001	2.000	K
Kr, cu_pct 2900	GM_FromMeshes: 2900	cu_pct	cu_pct 2900	0.001	0.310	K
Kr, cu_pct 3200 P1	GM_FromMeshes: 3200	cu_pct	cu_pct in GM_FromMeshes: 3200			K
Kr, cu_pct 3200 P2	GM_FromMeshes: 3200	cu_pct	cu_pct in GM_FromMeshes: 3200			K
Kr, cu_pct 3200 P3	GM_FromMeshes: 3200	cu_pct	cu_pct in GM_FromMeshes: 3200			K
Kr, cu_pct 32_33_36	GM_Consolidated_from_meshes: 32_33_36	cu_pct	cu_pct 32_33_36	0.001	1.300	K
Kr, cu_pct 3500	GM_FromMeshes: 3500	cu_pct	cu_pct 3500	0.001	0.310	K

table continues…

General				Value Capping		
Name	Domain	Values	Estimate Name	Lower Bound	Upper Bound	Estimate Type
Kr, cu_pct 3900	GM_Consolidated_from_meshes: 3900	cu_pct	cu_pct 3900	0.001	0.320	K
Kr, cu_pct 4601	GM_FromMeshes: 4601	cu_pct	cu_pct 4601	0.001	0.600	K
Kr, cu_pct 4901	GM_FromMeshes: 4901	cu_pct	cu_pct 4901	0.001	0.184	K
Kr, cu_pct 5601	GM_FromMeshes: 5601	cu_pct	cu_pct 5601	0.001	1.090	K
Kr, cu_pct 5602	GM_FromMeshes: 5602	cu_pct	cu_pct 5602	0.001	0.658	K
Kr, cu_pct 5603	GM_FromMeshes: 5603	cu_pct	cu_pct 5603	0.001	0.246	K
Kr, cu_pct 5604	GM_FromMeshes: 5604	cu_pct	cu_pct 5604	0.001	0.216	K
Kr, cu_pct 5605	GM_FromMeshes: 5605	cu_pct	cu_pct 5605	0.001	0.495	K
Kr, cu_pct 5901_5902	GM_Consolidated_from_meshes: 5901	cu_pct	cu_pct 5901_5902	0.001	0.400	K
Kr, cu_pct 6900_7500	GM_Consolidated_from_meshes: 6900_7500	cu_pct	cu_pct 6900_7500	0.001	0.400	K
Kr, cu_pct 7900	GM_Consolidated_from_meshes: 7900	cu_pct	cu_pct 7900	0.001	0.310	K
Kr, cu_pct 8900	GM_FromMeshes: 8900	cu_pct	cu_pct 8900	0.001	0.200	K
Kr, cu_pct 9900	GM_FromMeshes: 9900	cu_pct	cu_pct 9900	0.001	0.200	K
Kr, mo_pct 1000	GM_Consolidated_from_meshes: 1000	mo_pct	mo_pct 1000	0.001	0.030	K
Kr, mo_pct 11_12_13_16	GM_Consolidated_from_meshes: 11_12_13_16	mo_pct	mo_pct 11_12_13_16	0.001	0.300	K
Kr, mo_pct 1900	GM_FromMeshes: 1900	mo_pct	mo_pct 1900	0.001	0.050	K
Kr, mo_pct 2400	GM_FromMeshes: 2400	mo_pct	mo_pct 2400	0.001	0.200	K
Kr, mo_pct 2601	GM_FromMeshes: 2601	mo_pct	mo_pct 2601	0.001	0.060	K
Kr, mo_pct 2602	GM_FromMeshes: 2602	mo_pct	mo_pct 2602	0.001	2.000	K

table continues…

General			Estimate Name	Value Capping		Estimate Type
Name	Domain	Values		Lower Bound	Upper Bound	
Kr, mo_pct 2900	GM_FromMeshes: 2900	mo_pct	mo_pct 2900	0.001	0.015	K
Kr, mo_pct 32_33_36	GM_Consolidated_from_meshes: 32_33_36	mo_pct	mo_pct 32_33_36	0.001	0.200	K
Kr, mo_pct 3500	GM_FromMeshes: 3500	mo_pct	mo_pct 3500	0.001	0.055	K
Kr, mo_pct 3900	GM_Consolidated_from_meshes: 3900	mo_pct	mo_pct 3900	0.001	0.025	K
Kr, mo_pct 4601	GM_FromMeshes: 4601	mo_pct	mo_pct 4601	0.001	0.050	K
Kr, mo_pct 4901	GM_FromMeshes: 4901	mo_pct	mo_pct 4901	0.001	0.008	K
Kr, mo_pct 5601	GM_FromMeshes: 5601	mo_pct	mo_pct 5601	0.001	0.040	K
Kr, mo_pct 5602	GM_FromMeshes: 5602	mo_pct	mo_pct 5602	0.001	0.018	K
Kr, mo_pct 5603	GM_FromMeshes: 5603	mo_pct	mo_pct 5603	0.001	0.014	K
Kr, mo_pct 5604	GM_FromMeshes: 5604	mo_pct	mo_pct 5604	0.001	0.048	K
Kr, mo_pct 5605	GM_FromMeshes: 5605	mo_pct	mo_pct 5605	0.001	0.008	K
Kr, mo_pct 5901_5902	GM_Consolidated_from_meshes: 5901	mo_pct	mo_pct 5901_5902	0.001	0.030	K
Kr, mo_pct 6900_7500	GM_Consolidated_from_meshes: 6900_7500	mo_pct	mo_pct 6900_7500	0.001	0.015	K
Kr, mo_pct 7900	GM_Consolidated_from_meshes: 7900	mo_pct	mo_pct 7900	0.001	0.020	K
Kr, mo_pct 8900	GM_FromMeshes: 8900	mo_pct	mo_pct 8900	0.001	0.015	K
Kr, mo_pct 9900	GM_FromMeshes: 9900	mo_pct	mo_pct 9900	0.001	0.015	K
NN, ag2_gt 1000	GM_Consolidated_from_meshes: 1000	ag2_gt	ag2_gt 1000	0.001	2.500	NN
NN, ag2_gt 11_12_13_16	GM_Consolidated_from_meshes: 11_12_13_16	ag2_gt	ag2_gt 11_12_13_16	0.001	20.000	NN
NN, ag2_gt 1900	GM_FromMeshes: 1900	ag2_gt	ag2_gt 1900	0.001	2.000	NN

table continues…

General				Value Capping		
Name	Domain	Values	Estimate Name	Lower Bound	Upper Bound	Estimate Type
NN, ag2_gt 2400	GM_FromMeshes: 2400	ag2_gt	ag2_gt 2400	0.001	12.000	NN
NN, ag2_gt 2601	GM_FromMeshes: 2601	ag2_gt	ag2_gt 2601	0.001	3.000	NN
NN, ag2_gt 2602	GM_FromMeshes: 2602	ag2_gt	ag2_gt 2602	0.001	6.000	NN
NN, ag2_gt 2900	GM_FromMeshes: 2900	ag2_gt	ag2_gt 2900	0.001	1.100	NN
NN, ag2_gt 32_33_36	GM_Consolidated_from_meshes: 32_33_36	ag2_gt	ag2_gt 32_33_36	0.001	30.000	NN
NN, ag2_gt 3500	GM_FromMeshes: 3500	ag2_gt	ag2_gt 3500	0.001	2.000	NN
NN, ag2_gt 3900	GM_Consolidated_from_meshes: 3900	ag2_gt	ag2_gt 3900	0.001	10.000	NN
NN, ag2_gt 4601	GM_FromMeshes: 4601	ag2_gt	ag2_gt 4601	0.001	2.000	NN
NN, ag2_gt 4901	GM_FromMeshes: 4901	ag2_gt	ag2_gt 4901	0.001	1.000	NN
NN, ag2_gt 5601	GM_FromMeshes: 5601	ag2_gt	ag2_gt 5601	0.001	3.000	NN
NN, ag2_gt 5602	GM_FromMeshes: 5602	ag2_gt	ag2_gt 5602	0.001	0.900	NN
NN, ag2_gt 5603	GM_FromMeshes: 5603	ag2_gt	ag2_gt 5603	0.001	1.236	NN
NN, ag2_gt 5604	GM_FromMeshes: 5604	ag2_gt	ag2_gt 5604	0.001	4.866	NN
NN, ag2_gt 5605	GM_FromMeshes: 5605	ag2_gt	ag2_gt 5605			NN
NN, ag2_gt 5901_5902	GM_Consolidated_from_meshes: 5901	ag2_gt	ag2_gt 5901_5902	0.001	4.000	NN
NN, ag2_gt 6900_7500	GM_Consolidated_from_meshes: 6900_7500	ag2_gt	ag2_gt 6900_7500	0.001	3.000	NN
NN, ag2_gt 7900	GM_Consolidated_from_meshes: 7900	ag2_gt	ag2_gt 7900	0.001	2.000	NN
NN, ag2_gt 8900	GM_FromMeshes: 8900	ag2_gt	ag2_gt 8900	0.001	4.000	NN
NN, ag2_gt 9900	GM_FromMeshes: 9900	ag2_gt	ag2_gt 9900	0.001	6.000	NN

table continues…

General				Value Capping		
Name	Domain	Values	Estimate Name	Lower Bound	Upper Bound	Estimate Type
NN, au2_gt 1000	GM_Consolidated_from_meshes: 1000	au2_gt	au2_gt 1000	0.001	0.600	NN
NN, au2_gt 11_12_13_16	GM_Consolidated_from_meshes: 11_12_13_16	au2_gt	au2_gt 11_12_13_16	0.001	3.000	NN
NN, au2_gt 1900	GM_FromMeshes: 1900	au2_gt	au2_gt 1900	0.001	0.270	NN
NN, au2_gt 2400	GM_FromMeshes: 2400	au2_gt	au2_gt 2400	0.001	1.000	NN
NN, au2_gt 2601	GM_FromMeshes: 2601	au2_gt	au2_gt 2601	0.001	0.300	NN
NN, au2_gt 2602	GM_FromMeshes: 2602	au2_gt	au2_gt 2602	0.001	0.700	NN
NN, au2_gt 2900	GM_FromMeshes: 2900	au2_gt	au2_gt 2900	0.001	0.120	NN
NN, au2_gt 32_33_36	GM_Consolidated_from_meshes: 32_33_36	au2_gt	au2_gt 32_33_36	0.001	3.000	NN
NN, au2_gt 3500	GM_FromMeshes: 3500	au2_gt	au2_gt 3500	0.001	0.400	NN
NN, au2_gt 3900	GM_Consolidated_from_meshes: 3900	au2_gt	au2_gt 3900	0.001	0.250	NN
NN, au2_gt 4601	GM_FromMeshes: 4601	au2_gt	au2_gt 4601	0.001	0.250	NN
NN, au2_gt 4901	GM_FromMeshes: 4901	au2_gt	au2_gt 4901	0.001	0.200	NN
NN, au2_gt 5601	GM_FromMeshes: 5601	au2_gt	au2_gt 5601	0.001	0.300	NN
NN, au2_gt 5602	GM_FromMeshes: 5602	au2_gt	au2_gt 5602	0.001	0.200	NN
NN, au2_gt 5603	GM_FromMeshes: 5603	au2_gt	au2_gt 5603	0.001	0.150	NN
NN, au2_gt 5604	GM_FromMeshes: 5604	au2_gt	au2_gt 5604	0.001	0.140	NN
NN, au2_gt 5605	GM_FromMeshes: 5605	au2_gt	au2_gt 5605	0.001	0.250	NN
NN, au2_gt 5901_5902	GM_Consolidated_from_meshes: 5901	au2_gt	au2_gt 5901_5902	0.001	0.250	NN
NN, au2_gt 6900_7500	GM_Consolidated_from_meshes: 6900_7500	au2_gt	au2_gt 6900_7500	0.001	0.300	NN

table continues…

General			Estimate Name	Value Capping		Estimate Type
Name	Domain	Values		Lower Bound	Upper Bound	
NN, au2_gt 7900	GM_Consolidated_from_meshes: 7900	au2_gt	au2_gt 7900	0.001	0.200	NN
NN, au2_gt 8900	GM_FromMeshes: 8900	au2_gt	au2_gt 8900	0.001	0.100	NN
NN, au2_gt 9900	GM_FromMeshes: 9900	au2_gt	au2_gt 9900	0.001	0.100	NN
NN, cu_pct 1000	GM_Consolidated_from_meshes: 1000	cu_pct	cu_pct 1000	0.001	0.500	NN
NN, cu_pct 11_12_13_16	GM_Consolidated_from_meshes: 11_12_13_16	cu_pct	cu_pct 11_12_13_16	0.001	1.300	NN
NN, cu_pct 1900	GM_FromMeshes: 1900	cu_pct	cu_pct 1900	0.001	0.383	NN
NN, cu_pct 2400	GM_FromMeshes: 2400	cu_pct	cu_pct 2400	0.001	2.000	NN
NN, cu_pct 2601	GM_FromMeshes: 2601	cu_pct	cu_pct 2601	0.001	0.650	NN
NN, cu_pct 2602	GM_FromMeshes: 2602	cu_pct	cu_pct 2602	0.001	2.000	NN
NN, cu_pct 2900	GM_FromMeshes: 2900	cu_pct	cu_pct 2900	0.001	0.310	NN
NN, cu_pct 32_33_36	GM_Consolidated_from_meshes: 32_33_36	cu_pct	cu_pct 32_33_36	0.001	1.300	NN
NN, cu_pct 3500	GM_FromMeshes: 3500	cu_pct	cu_pct 3500	0.001	0.310	NN
NN, cu_pct 3900	GM_Consolidated_from_meshes: 3900	cu_pct	cu_pct 3900	0.001	0.320	NN
NN, cu_pct 4601	GM_FromMeshes: 4601	cu_pct	cu_pct 4601	0.001	0.600	NN
NN, cu_pct 4901	GM_FromMeshes: 4901	cu_pct	cu_pct 4901	0.001	0.184	NN
NN, cu_pct 5601	GM_FromMeshes: 5601	cu_pct	cu_pct 5601	0.001	1.090	NN
NN, cu_pct 5602	GM_FromMeshes: 5602	cu_pct	cu_pct 5602	0.001	0.658	NN
NN, cu_pct 5603	GM_FromMeshes: 5603	cu_pct	cu_pct 5603	0.001	0.246	NN
NN, cu_pct 5604	GM_FromMeshes: 5604	cu_pct	cu_pct 5604	0.001	0.216	NN

table continues…

General				Value Capping		
Name	Domain	Values	Estimate Name	Lower Bound	Upper Bound	Estimate Type
NN, cu_pct 5605	GM_FromMeshes: 5605	cu_pct	cu_pct 5605	0.001	0.495	NN
NN, cu_pct 5901_5902	GM_Consolidated_from_meshes: 5901	cu_pct	cu_pct 5901_5902	0.001	0.400	NN
NN, cu_pct 6900_7500	GM_Consolidated_from_meshes: 6900_7500	cu_pct	cu_pct 6900_7500	0.001	0.400	NN
NN, cu_pct 7900	GM_Consolidated_from_meshes: 7900	cu_pct	cu_pct 7900	0.001	0.310	NN
NN, cu_pct 8900	GM_FromMeshes: 8900	cu_pct	cu_pct 8900	0.001	0.200	NN
NN, cu_pct 9900	GM_FromMeshes: 9900	cu_pct	cu_pct 9900	0.001	0.200	NN
NN, mo_pct 1000	GM_Consolidated_from_meshes: 1000	mo_pct	mo_pct 1000	0.001	0.030	NN
NN, mo_pct 11_12_13_16	GM_Consolidated_from_meshes: 11_12_13_16	mo_pct	mo_pct 11_12_13_16	0.001	0.300	NN
NN, mo_pct 1900	GM_FromMeshes: 1900	mo_pct	mo_pct 1900	0.001	0.050	NN
NN, mo_pct 2400	GM_FromMeshes: 2400	mo_pct	mo_pct 2400	0.001	0.200	NN
NN, mo_pct 2601	GM_FromMeshes: 2601	mo_pct	mo_pct 2601	0.001	0.060	NN
NN, mo_pct 2602	GM_FromMeshes: 2602	mo_pct	mo_pct 2602	0.001	2.000	NN
NN, mo_pct 2900	GM_FromMeshes: 2900	mo_pct	mo_pct 2900	0.001	0.015	NN
NN, mo_pct 32_33_36	GM_Consolidated_from_meshes: 32_33_36	mo_pct	mo_pct 32_33_36	0.001	0.200	NN
NN, mo_pct 3500	GM_FromMeshes: 3500	mo_pct	mo_pct 3500	0.001	0.055	NN
NN, mo_pct 3900	GM_Consolidated_from_meshes: 3900	mo_pct	mo_pct 3900	0.001	0.025	NN
NN, mo_pct 4601	GM_FromMeshes: 4601	mo_pct	mo_pct 4601	0.001	0.050	NN
NN, mo_pct 4901	GM_FromMeshes: 4901	mo_pct	mo_pct 4901	0.001	0.008	NN
NN, mo_pct 5601	GM_FromMeshes: 5601	mo_pct	mo_pct 5601	0.001	0.040	NN

table continues…

General			Estimate Name	Value Capping		Estimate Type
Name	Domain	Values		Lower Bound	Upper Bound	
NN, mo_pct 5602	GM_FromMeshes: 5602	mo_pct	mo_pct 5602	0.001	0.018	NN
NN, mo_pct 5603	GM_FromMeshes: 5603	mo_pct	mo_pct 5603	0.001	0.014	NN
NN, mo_pct 5604	GM_FromMeshes: 5604	mo_pct	mo_pct 5604	0.001	0.048	NN
NN, mo_pct 5605	GM_FromMeshes: 5605	mo_pct	mo_pct 5605	0.001	0.008	NN
NN, mo_pct 5901_5902	GM_Consolidated_from_meshes: 5901	mo_pct	mo_pct 5901_5902	0.001	0.030	NN
NN, mo_pct 6900_7500	GM_Consolidated_from_meshes: 6900_7500	mo_pct	mo_pct 6900_7500	0.001	0.015	NN
NN, mo_pct 7900	GM_Consolidated_from_meshes: 7900	mo_pct	mo_pct 7900	0.001	0.020	NN
NN, mo_pct 8900	GM_FromMeshes: 8900	mo_pct	mo_pct 8900	0.001	0.015	NN
NN, mo_pct 9900	GM_FromMeshes: 9900	mo_pct	mo_pct 9900	0.001	0.015	NN

The estimation was done in a single pass. Search ellipses for copper, molybdenum, silver, and gold interpolation assumed ranges up to twice the maximum range of the variogram for the second structure. A block discretization of 5 x 5 x 3 steps (X, Y, Z) was used. Search parameters are shown in Table 14-9.

Table 14-9: Estimation Search Parameters

General			Ellipsoid Ranges			Ellipsoid Directions				Number of Samples	
Name	Domain	Values	Maximum	Intermediate	Minimum	Dip	Dip Azimuth	Pitch	Variable Orientation	Minimum	Maximum
ID, ag2_gt 1000	GM_Consolidated_from_meshes: 1000	ag2_gt	160	100	50				Variable Topo	5	11
ID, ag2_gt 11_12_13_16	GM_Consolidated_from_meshes: 11_12_13_16	ag2_gt	270	320	320				Variable Orientation	7	11
ID, ag2_gt 1900	GM_FromMeshes: 1900	ag2_gt	440	480	120				Variable Orientation	7	11
ID, ag2_gt 2400	GM_FromMeshes: 2400	ag2_gt	250	120	150				Variable Orientation	7	11
ID, ag2_gt 2601	GM_FromMeshes: 2601	ag2_gt	370	300	120				Variable Orientation	7	11
ID, ag2_gt 2602	GM_FromMeshes: 2602	ag2_gt	400	300	220				Variable Orientation	7	11
ID, ag2_gt 2900	GM_FromMeshes: 2900	ag2_gt	500	470	250				Variable Orientation	7	11
ID, ag2_gt 32_33_36	GM_Consolidated_from_meshes: 32_33_36	ag2_gt	320	240	220				Variable Orientation	7	11
ID, ag2_gt 3500	GM_FromMeshes: 3500	ag2_gt	400	400	200				Variable Orientation	7	11
ID, ag2_gt 3900	GM_Consolidated_from_meshes: 3900	ag2_gt	270	280	280				Variable Orientation	7	11
ID, ag2_gt 4601	GM_FromMeshes: 4601	ag2_gt	320	180	120				Variable Orientation	7	11
ID, ag2_gt 4901	GM_FromMeshes: 4901	ag2_gt	230	150	150				Variable Orientation	7	11
ID, ag2_gt 5601	GM_FromMeshes: 5601	ag2_gt	300	240	110				Variable Orientation	7	11
ID, ag2_gt 5602	GM_FromMeshes: 5602	ag2_gt	280	240	110				Variable Orientation	7	11
ID, ag2_gt 5603	GM_FromMeshes: 5603	ag2_gt	280	240	110				Variable Orientation	7	11
ID, ag2_gt 5604	GM_FromMeshes: 5604	ag2_gt	280	240	110				Variable Orientation	4	11
ID, ag2_gt 5605	GM_FromMeshes: 5605	ag2_gt	280	240	110				Variable Orientation	5	11
ID, ag2_gt 5901_5902	GM_Consolidated_from_meshes: 5901	ag2_gt	280	400	240				Variable Orientation	7	11
ID, ag2_gt 6900_7500	GM_Consolidated_from_meshes: 6900_7500	ag2_gt	420	400	200				Variable Orientation	7	11
ID, ag2_gt 7900	GM_Consolidated_from_meshes: 7900	ag2_gt	520	480	160				Variable Orientation	7	11
ID, ag2_gt 8900	GM_FromMeshes: 8900	ag2_gt	500	450	140				Variable Orientation	7	11
ID, ag2_gt 9900	GM_FromMeshes: 9900	ag2_gt	360	240	100				Variable Orientation	7	11
ID, au2_gt 1000	GM_Consolidated_from_meshes: 1000	au2_gt	160	100	50				Variable Topo	5	11
ID, au2_gt 11_12_13_16	GM_Consolidated_from_meshes: 11_12_13_16	au2_gt	225	340	260				Variable Orientation	7	11
ID, au2_gt 1900	GM_FromMeshes: 1900	au2_gt	500	360	120				Variable Orientation	7	11
ID, au2_gt 2400	GM_FromMeshes: 2400	au2_gt	260	140	240				Variable Orientation	7	11

table continues…

General			Ellipsoid Ranges			Ellipsoid Directions				Number of Samples	
Name	Domain	Values	Maximum	Intermediate	Minimum	Dip	Dip Azimuth	Pitch	Variable Orientation	Minimum	Maximum
ID, au2_gt 2601	GM_FromMeshes: 2601	au2_gt	400	300	240				Variable Orientation	7	11
ID, au2_gt 2602	GM_FromMeshes: 2602	au2_gt	400	300	220				Variable Orientation	7	11
ID, au2_gt 2900	GM_FromMeshes: 2900	au2_gt	500	200	270				Variable Orientation	7	11
ID, au2_gt 32_33_36	GM_Consolidated_from_meshes: 32_33_36	au2_gt	470	360	240				Variable Orientation	7	11
ID, au2_gt 3500	GM_FromMeshes: 3500	au2_gt	400	350	250				Variable Orientation	7	11
ID, au2_gt 3900	GM_Consolidated_from_meshes: 3900	au2_gt	550	550	150				Variable Orientation	7	11
ID, au2_gt 4601	GM_FromMeshes: 4601	au2_gt	320	500	120				Variable Orientation	7	11
ID, au2_gt 4901	GM_FromMeshes: 4901	au2_gt	320	300	260				Variable Orientation	7	11
ID, au2_gt 5601	GM_FromMeshes: 5601	au2_gt	300	240	110				Variable Orientation	7	11
ID, au2_gt 5602	GM_FromMeshes: 5602	au2_gt	280	240	110				Variable Orientation	7	11
ID, au2_gt 5603	GM_FromMeshes: 5603	au2_gt	280	240	110				Variable Orientation	7	11
ID, au2_gt 5604	GM_FromMeshes: 5604	au2_gt	280	240	110				Variable Orientation	4	11
ID, au2_gt 5605	GM_FromMeshes: 5605	au2_gt	280	240	110				Variable Orientation	5	11
ID, au2_gt 5901_5902	GM_Consolidated_from_meshes: 5901	au2_gt	440	330	250				Variable Orientation	7	11
ID, au2_gt 6900_7500	GM_Consolidated_from_meshes: 6900_7500	au2_gt	500	400	270				Variable Orientation	7	11
ID, au2_gt 7900	GM_Consolidated_from_meshes: 7900	au2_gt	550	800	240				Variable Orientation	7	11
ID, au2_gt 8900	GM_FromMeshes: 8900	au2_gt	550	450	80				Variable Orientation	7	11
ID, au2_gt 9900	GM_FromMeshes: 9900	au2_gt	550	450	220				Variable Orientation	7	11
ID, cu_pct 1000	GM_Consolidated_from_meshes: 1000	cu_pct	160	100	50				Variable Topo	5	11
ID, cu_pct 11_12_13_16	GM_Consolidated_from_meshes: 11_12_13_16	cu_pct	269.2	203.3	131.7				Variable Orientation	7	11
ID, cu_pct 1900	GM_FromMeshes: 1900	cu_pct	500	400	150				Variable Orientation	7	11
ID, cu_pct 2400	GM_FromMeshes: 2400	cu_pct	188	109.4	146.7				Variable Orientation	7	11
ID, cu_pct 2601	GM_FromMeshes: 2601	cu_pct	400	300	240				Variable Orientation	7	11
ID, cu_pct 2602	GM_FromMeshes: 2602	cu_pct	400	300	220				Variable Orientation	7	11
ID, cu_pct 2900	GM_FromMeshes: 2900	cu_pct	500	400	120				Variable Orientation	7	11
ID, cu_pct 32_33_36	GM_Consolidated_from_meshes: 32_33_36	cu_pct	550	450	300				Variable Orientation	7	11
ID, cu_pct 3500	GM_FromMeshes: 3500	cu_pct	400	380	180				Variable Orientation	7	11
ID, cu_pct 3900	GM_Consolidated_from_meshes: 3900	cu_pct	550	550	180				Variable Orientation	7	11

table continues…

General			Ellipsoid Ranges			Ellipsoid Directions				Number of Samples	
Name	Domain	Values	Maximum	Intermediate	Minimum	Dip	Dip Azimuth	Pitch	Variable Orientation	Minimum	Maximum
ID, cu_pct 4601	GM_FromMeshes: 4601	cu_pct	400	230	120				Variable Orientation	7	11
ID, cu_pct 4901	GM_FromMeshes: 4901	cu_pct	300	300	250				Variable Orientation	7	11
ID, cu_pct 5601	GM_FromMeshes: 5601	cu_pct	300	240	110				Variable Orientation	7	11
ID, cu_pct 5602	GM_FromMeshes: 5602	cu_pct	280	240	110				Variable Orientation	7	11
ID, cu_pct 5603	GM_FromMeshes: 5603	cu_pct	280	240	110				Variable Orientation	7	11
ID, cu_pct 5604	GM_FromMeshes: 5604	cu_pct	280	240	110				Variable Orientation	4	11
ID, cu_pct 5605	GM_FromMeshes: 5605	cu_pct	280	240	110				Variable Orientation	5	11
ID, cu_pct 5901_5902	GM_Consolidated_from_meshes: 5901	cu_pct	500	400	120				Variable Orientation	7	11
ID, cu_pct 6900_7500	GM_Consolidated_from_meshes: 6900_7500	cu_pct	500	400	250				Variable Orientation	7	11
ID, cu_pct 7900	GM_Consolidated_from_meshes: 7900	cu_pct	500	450	240				Variable Orientation	7	11
ID, cu_pct 8900	GM_FromMeshes: 8900	cu_pct	550	450	150				Variable Orientation	7	11
ID, cu_pct 9900	GM_FromMeshes: 9900	cu_pct	550	450	150				Variable Orientation	7	11
ID, mo_pct 1000	GM_Consolidated_from_meshes: 1000	mo_pct	160	100	50				Variable Topo	5	11
ID, mo_pct 11_12_13_16	GM_Consolidated_from_meshes: 11_12_13_16	mo_pct	600	480	300				Variable Orientation	7	11
ID, mo_pct 1900	GM_FromMeshes: 1900	mo_pct	440	400	200				Variable Orientation	7	11
ID, mo_pct 2400	GM_FromMeshes: 2400	mo_pct	300	260	330				Variable Orientation	7	11
ID, mo_pct 2601	GM_FromMeshes: 2601	mo_pct	350	260	300				Variable Orientation	7	11
ID, mo_pct 2602	GM_FromMeshes: 2602	mo_pct	400	300	220				Variable Orientation	7	11
ID, mo_pct 2900	GM_FromMeshes: 2900	mo_pct	600	550	400				Variable Orientation	7	11
ID, mo_pct 32_33_36	GM_Consolidated_from_meshes: 32_33_36	mo_pct	360	350	350				Variable Orientation	7	11
ID, mo_pct 3500	GM_FromMeshes: 3500	mo_pct	780	700	400				Variable Orientation	7	11
ID, mo_pct 3900	GM_Consolidated_from_meshes: 3900	mo_pct	270	440	120				Variable Orientation	7	11
ID, mo_pct 4601	GM_FromMeshes: 4601	mo_pct	430	300	170				Variable Orientation	7	11
ID, mo_pct 4901	GM_FromMeshes: 4901	mo_pct	400	360	200				Variable Orientation	7	11
ID, mo_pct 5601	GM_FromMeshes: 5601	mo_pct	300	240	110				Variable Orientation	7	11
ID, mo_pct 5602	GM_FromMeshes: 5602	mo_pct	280	240	110				Variable Orientation	7	11
ID, mo_pct 5603	GM_FromMeshes: 5603	mo_pct	280	240	110				Variable Orientation	7	11
ID, mo_pct 5604	GM_FromMeshes: 5604	mo_pct	280	240	110				Variable Orientation	4	11

table continues…

General			Ellipsoid Ranges			Ellipsoid Directions				Number of Samples	
Name	Domain	Values	Maximum	Intermediate	Minimum	Dip	Dip Azimuth	Pitch	Variable Orientation	Minimum	Maximum
ID, mo_pct 5605	GM_FromMeshes: 5605	mo_pct	280	240	110				Variable Orientation	5	11
ID, mo_pct 5901_5902	GM_Consolidated_from_meshes: 5901	mo_pct	500	400	250				Variable Orientation	7	11
ID, mo_pct 6900_7500	GM_Consolidated_from_meshes: 6900_7500	mo_pct	500	400	260				Variable Orientation	7	11
ID, mo_pct 7900	GM_Consolidated_from_meshes: 7900	mo_pct	500	500	250				Variable Orientation	7	11
ID, mo_pct 8900	GM_FromMeshes: 8900	mo_pct	500	450	140				Variable Orientation	7	11
ID, mo_pct 9900	GM_FromMeshes: 9900	mo_pct	550	450	150				Variable Orientation	7	11
Kr, ag2_gt 1000	GM_Consolidated_from_meshes: 1000	ag2_gt	160	100	50				Variable Topo	5	11
Kr, ag2_gt 11_12_13_16	GM_Consolidated_from_meshes: 11_12_13_16	ag2_gt	270	320	320				Variable Orientation	7	11
Kr, ag2_gt 1900	GM_FromMeshes: 1900	ag2_gt	440	480	120				Variable Orientation	7	11
Kr, ag2_gt 2400	GM_FromMeshes: 2400	ag2_gt	250	120	150				Variable Orientation	7	11
Kr, ag2_gt 2601	GM_FromMeshes: 2601	ag2_gt	370	300	120				Variable Orientation	7	11
Kr, ag2_gt 2602	GM_FromMeshes: 2602	ag2_gt	400	300	220				Variable Orientation	7	11
Kr, ag2_gt 2900	GM_FromMeshes: 2900	ag2_gt	500	470	250				Variable Orientation	7	11
Kr, ag2_gt 32_33_36	GM_Consolidated_from_meshes: 32_33_36	ag2_gt	320	240	220				Variable Orientation	7	11
Kr, ag2_gt 3500	GM_FromMeshes: 3500	ag2_gt	400	400	200				Variable Orientation	7	11
Kr, ag2_gt 3900	GM_Consolidated_from_meshes: 3900	ag2_gt	270	280	280				Variable Orientation	7	11
Kr, ag2_gt 4601	GM_FromMeshes: 4601	ag2_gt	320	180	120				Variable Orientation	7	11
Kr, ag2_gt 4901	GM_FromMeshes: 4901	ag2_gt	230	150	150				Variable Orientation	7	11
Kr, ag2_gt 5601	GM_FromMeshes: 5601	ag2_gt	300	240	110				Variable Orientation	7	11
Kr, ag2_gt 5602	GM_FromMeshes: 5602	ag2_gt	280	240	110				Variable Orientation	7	11
Kr, ag2_gt 5603	GM_FromMeshes: 5603	ag2_gt	280	240	110				Variable Orientation	7	11
Kr, ag2_gt 5604	GM_FromMeshes: 5604	ag2_gt	280	240	110				Variable Orientation	4	11
Kr, ag2_gt 5605	GM_FromMeshes: 5605	ag2_gt	280	240	110				Variable Orientation	5	11
Kr, ag2_gt 5901_5902	GM_Consolidated_from_meshes: 5901	ag2_gt	280	400	240				Variable Orientation	7	11
Kr, ag2_gt 6900_7500	GM_Consolidated_from_meshes: 6900_7500	ag2_gt	420	400	200				Variable Orientation	7	11
Kr, ag2_gt 7900	GM_Consolidated_from_meshes: 7900	ag2_gt	520	480	160				Variable Orientation	7	11
Kr, ag2_gt 8900	GM_FromMeshes: 8900	ag2_gt	500	450	140				Variable Orientation	7	11
Kr, ag2_gt 9900	GM_FromMeshes: 9900	ag2_gt	360	240	100				Variable Orientation	7	11

table continues…

General			Ellipsoid Ranges			Ellipsoid Directions				Number of Samples	
Name	Domain	Values	Maximum	Intermediate	Minimum	Dip	Dip Azimuth	Pitch	Variable Orientation	Minimum	Maximum
Kr, au2_gt 1000	GM_Consolidated_from_meshes: 1000	au2_gt	160	100	50				Variable Topo	5	11
Kr, au2_gt 11_12_13_16	GM_Consolidated_from_meshes: 11_12_13_16	au2_gt	225	340	260				Variable Orientation	7	11
Kr, au2_gt 1900	GM_FromMeshes: 1900	au2_gt	500	360	120				Variable Orientation	7	11
Kr, au2_gt 2400	GM_FromMeshes: 2400	au2_gt	260	1.4	240				Variable Orientation	7	11
Kr, au2_gt 2601	GM_FromMeshes: 2601	au2_gt	400	300	240				Variable Orientation	7	11
Kr, au2_gt 2602	GM_FromMeshes: 2602	au2_gt	400	300	220				Variable Orientation	7	11
Kr, au2_gt 2900	GM_FromMeshes: 2900	au2_gt	500	200	270				Variable Orientation	7	11
Kr, au2_gt 32_33_36	GM_Consolidated_from_meshes: 32_33_36	au2_gt	470	360	240				Variable Orientation	7	11
Kr, au2_gt 3500	GM_FromMeshes: 3500	au2_gt	400	350	250				Variable Orientation	7	11
Kr, au2_gt 3900	GM_Consolidated_from_meshes: 3900	au2_gt	550	550	150				Variable Orientation	7	11
Kr, au2_gt 4601	GM_FromMeshes: 4601	au2_gt	320	500	120				Variable Orientation	7	11
Kr, au2_gt 4901	GM_FromMeshes: 4901	au2_gt	320	300	260				Variable Orientation	7	11
Kr, au2_gt 5601	GM_FromMeshes: 5601	au2_gt	300	240	110				Variable Orientation	7	11
Kr, au2_gt 5602	GM_FromMeshes: 5602	au2_gt	280	240	110				Variable Orientation	7	11
Kr, au2_gt 5603	GM_FromMeshes: 5603	au2_gt	280	240	110				Variable Orientation	7	11
Kr, au2_gt 5604	GM_FromMeshes: 5604	au2_gt	280	240	110				Variable Orientation	4	11
Kr, au2_gt 5605	GM_FromMeshes: 5605	au2_gt	280	240	110				Variable Orientation	5	11
Kr, au2_gt 5901_5902	GM_Consolidated_from_meshes: 5901	au2_gt	440	330	250				Variable Orientation	7	11
Kr, au2_gt 6900_7500	GM_Consolidated_from_meshes: 6900_7500	au2_gt	500	400	270				Variable Orientation	7	11
Kr, au2_gt 7900	GM_Consolidated_from_meshes: 7900	au2_gt	550	800	240				Variable Orientation	7	11
Kr, au2_gt 8900	GM_FromMeshes: 8900	au2_gt	550	450	80				Variable Orientation	7	11
Kr, au2_gt 9900	GM_FromMeshes: 9900	au2_gt	550	450	220				Variable Orientation	7	11
Kr, cu_pct 1000	GM_Consolidated_from_meshes: 1000	cu_pct	160	100	50				Variable Topo	5	11
Kr, cu_pct 11_12_13_16	GM_Consolidated_from_meshes: 11_12_13_16	cu_pct	269.2	203.3	131.7				Variable Orientation	7	11
Kr, cu_pct 1900	GM_FromMeshes: 1900	cu_pct	500	400	150				Variable Orientation	7	11
Kr, cu_pct 2400	GM_FromMeshes: 2400	cu_pct	188	109.4	146.7	17.0	288.9	138.0		7	11
Kr, cu_pct 24_26	GM_Consolidated_from_meshes: 24_26	cu_pct	218.2	130.9	43.6	0.0	0.0	90.0		4	20
Kr, cu_pct 2601	GM_FromMeshes: 2601	cu_pct	400	300	240				Variable Orientation	7	11

table continues…

General			Ellipsoid Ranges			Ellipsoid Directions				Number of Samples	
Name	Domain	Values	Maximum	Intermediate	Minimum	Dip	Dip Azimuth	Pitch	Variable Orientation	Minimum	Maximum
Kr, cu_pct 2601 P1	GM_FromMeshes: 2601	cu_pct	80	80	25	64.5	234.7	167.0		7	11
Kr, cu_pct 2601 P2	GM_FromMeshes: 2601	cu_pct	200	80	90	64.5	234.7	167.0		4	11
Kr, cu_pct 2601 P3	GM_FromMeshes: 2601	cu_pct	400	160	180	64.5	234.7	167.0		2	11
Kr, cu_pct 2602	GM_FromMeshes: 2602	cu_pct	400	300	220				Variable Orientation	7	11
Kr, cu_pct 2900	GM_FromMeshes: 2900	cu_pct	500	400	120				Variable Orientation	7	11
Kr, cu_pct 3200 P1	GM_FromMeshes: 3200	cu_pct	140	100	55	59.9	90.5	156.9		7	11
Kr, cu_pct 3200 P2	GM_FromMeshes: 3200	cu_pct	300	100	200	59.9	90.5	156.9		4	11
Kr, cu_pct 3200 P3	GM_FromMeshes: 3200	cu_pct	600	200	400	59.9	90.5	156.9		2	11
Kr, cu_pct 32_33_36	GM_Consolidated_from_meshes: 32_33_36	cu_pct	278.9	227.2	152				Variable Orientation	7	11
Kr, cu_pct 3500	GM_FromMeshes: 3500	cu_pct	400	380	180				Variable Orientation	7	11
Kr, cu_pct 3900	GM_Consolidated_from_meshes: 3900	cu_pct	550	550	180				Variable Orientation	7	11
Kr, cu_pct 4601	GM_FromMeshes: 4601	cu_pct	400	230	120				Variable Orientation	7	11
Kr, cu_pct 4901	GM_FromMeshes: 4901	cu_pct	162.1	150.8	131.7				Variable Orientation	7	11
Kr, cu_pct 5601	GM_FromMeshes: 5601	cu_pct	300	240	110				Variable Orientation	7	11
Kr, cu_pct 5602	GM_FromMeshes: 5602	cu_pct	280	240	110				Variable Orientation	7	11
Kr, cu_pct 5603	GM_FromMeshes: 5603	cu_pct	280	240	110				Variable Orientation	7	11
Kr, cu_pct 5604	GM_FromMeshes: 5604	cu_pct	280	240	110				Variable Orientation	4	11
Kr, cu_pct 5605	GM_FromMeshes: 5605	cu_pct	280	240	110				Variable Orientation	5	11
Kr, cu_pct 5901_5902	GM_Consolidated_from_meshes: 5901	cu_pct	500	400	120				Variable Orientation	7	11
Kr, cu_pct 6900_7500	GM_Consolidated_from_meshes: 6900_7500	cu_pct	500	400	240				Variable Orientation	7	11
Kr, cu_pct 7900	GM_Consolidated_from_meshes: 7900	cu_pct	500	450	240				Variable Orientation	7	11
Kr, cu_pct 8900	GM_FromMeshes: 8900	cu_pct	550	450	150				Variable Orientation	7	11
Kr, cu_pct 9900	GM_FromMeshes: 9900	cu_pct	550	450	150				Variable Orientation	7	11
Kr, mo_pct 1000	GM_Consolidated_from_meshes: 1000	mo_pct	160	100	50				Variable Topo	5	11
Kr, mo_pct 11_12_13_16	GM_Consolidated_from_meshes: 11_12_13_16	mo_pct	600	480	300				Variable Orientation	7	11
Kr, mo_pct 1900	GM_FromMeshes: 1900	mo_pct	440	400	200				Variable Orientation	7	11
Kr, mo_pct 2400	GM_FromMeshes: 2400	mo_pct	300	260	330				Variable Orientation	7	11
Kr, mo_pct 2601	GM_FromMeshes: 2601	mo_pct	360	260	300				Variable Orientation	7	11

table continues…

General			Ellipsoid Ranges			Ellipsoid Directions				Number of Samples	
Name	Domain	Values	Maximum	Intermediate	Minimum	Dip	Dip Azimuth	Pitch	Variable Orientation	Minimum	Maximum
Kr, mo_pct 2602	GM_FromMeshes: 2602	mo_pct	400	300	220				Variable Orientation	7	11
Kr, mo_pct 2900	GM_FromMeshes: 2900	mo_pct	600	550	400				Variable Orientation	7	11
Kr, mo_pct 32_33_36	GM_Consolidated_from_meshes: 32_33_36	mo_pct	360	350	350				Variable Orientation	7	11
Kr, mo_pct 3500	GM_FromMeshes: 3500	mo_pct	780	700	400				Variable Orientation	7	11
Kr, mo_pct 3900	GM_Consolidated_from_meshes: 3900	mo_pct	270	440	120				Variable Orientation	7	11
Kr, mo_pct 4601	GM_FromMeshes: 4601	mo_pct	430	300	170				Variable Orientation	7	11
Kr, mo_pct 4901	GM_FromMeshes: 4901	mo_pct	400	360	200				Variable Orientation	7	11
Kr, mo_pct 5601	GM_FromMeshes: 5601	mo_pct	300	240	110				Variable Orientation	7	11
Kr, mo_pct 5602	GM_FromMeshes: 5602	mo_pct	280	240	110				Variable Orientation	7	11
Kr, mo_pct 5603	GM_FromMeshes: 5603	mo_pct	280	240	110				Variable Orientation	7	11
Kr, mo_pct 5604	GM_FromMeshes: 5604	mo_pct	280	240	110				Variable Orientation	4	11
Kr, mo_pct 5605	GM_FromMeshes: 5605	mo_pct	280	240	110				Variable Orientation	5	11
Kr, mo_pct 5901_5902	GM_Consolidated_from_meshes: 5901	mo_pct	500	400	250				Variable Orientation	7	11
Kr, mo_pct 6900_7500	GM_Consolidated_from_meshes: 6900_7500	mo_pct	500	400	260				Variable Orientation	7	11
Kr, mo_pct 7900	GM_Consolidated_from_meshes: 7900	mo_pct	500	500	250				Variable Orientation	7	11
Kr, mo_pct 8900	GM_FromMeshes: 8900	mo_pct	500	450	140				Variable Orientation	7	11
Kr, mo_pct 9900	GM_FromMeshes: 9900	mo_pct	550	450	150				Variable Orientation	7	11
NN, ag2_gt 1000	GM_Consolidated_from_meshes: 1000	ag2_gt	160	100	50	27.7	249.8	65.2			
NN, ag2_gt 11_12_13_16	GM_Consolidated_from_meshes: 11_12_13_16	ag2_gt	270	320	320	26.0	262.0	107.8			
NN, ag2_gt 1900	GM_FromMeshes: 1900	ag2_gt	440	480	120	17.0	288.9	138.0			
NN, ag2_gt 2400	GM_FromMeshes: 2400	ag2_gt	250	120	150	17.0	288.9	147.5			
NN, ag2_gt 2601	GM_FromMeshes: 2601	ag2_gt	370	300	120	17.0	288.9	154.2			
NN, ag2_gt 2602	GM_FromMeshes: 2602	ag2_gt	400	300	220	83.8	79.8	4.6			
NN, ag2_gt 2900	GM_FromMeshes: 2900	ag2_gt	500	470	250	11.5	344.8	67.1			
NN, ag2_gt 32_33_36	GM_Consolidated_from_meshes: 32_33_36	ag2_gt	320	240	220	44.9	73.7	69.1			
NN, ag2_gt 3500	GM_FromMeshes: 3500	ag2_gt	400	400	200	21.4	231.4	24.9			
NN, ag2_gt 3900	GM_Consolidated_from_meshes: 3900	ag2_gt	270	280	280	44.3	85.5	170.1			
NN, ag2_gt 4601	GM_FromMeshes: 4601	ag2_gt	320	180	120	87.9	281.2	14.7			

table continues…

General			Ellipsoid Ranges			Ellipsoid Directions			Variable Orientation	Number of Samples	
Name	Domain	Values	Maximum	Intermediate	Minimum	Dip	Dip Azimuth	Pitch		Minimum	Maximum
NN, ag2_gt 4901	GM_FromMeshes: 4901	ag2_gt	230	150	150	78.8	95.6	23.7			
NN, ag2_gt 5601	GM_FromMeshes: 5601	ag2_gt	300	240	110	17.5	355.3	109.8			
NN, ag2_gt 5602	GM_FromMeshes: 5602	ag2_gt	280	240	110	17.5	355.3	108.2			
NN, ag2_gt 5603	GM_FromMeshes: 5603	ag2_gt	280	240	110	17.5	355.3	77.8			
NN, ag2_gt 5604	GM_FromMeshes: 5604	ag2_gt	280	240	110	17.5	355.3	85.0			
NN, ag2_gt 5605	GM_FromMeshes: 5605	ag2_gt	280	240	110	17.0	288.9	138.0			
NN, ag2_gt 5901_5902	GM_Consolidated_from_meshes: 5901	ag2_gt	280	400	240	19.6	324.9	110.1			
NN, ag2_gt 6900_7500	GM_Consolidated_from_meshes: 6900_7500	ag2_gt	420	400	200	6.1	31.7	23.4			
NN, ag2_gt 7900	GM_Consolidated_from_meshes: 7900	ag2_gt	520	480	160	74.2	77.7	104.7			
NN, ag2_gt 8900	GM_FromMeshes: 8900	ag2_gt	500	450	140	74.2	77.7	6.7			
NN, ag2_gt 9900	GM_FromMeshes: 9900	ag2_gt	360	240	100	15.5	205.0	63.8			
NN, au2_gt 1000	GM_Consolidated_from_meshes: 1000	au2_gt	160	100	50	27.7	249.8	65.2			
NN, au2_gt 11_12_13_16	GM_Consolidated_from_meshes: 11_12_13_16	au2_gt	225	340	260	15.4	342.5	0.0			
NN, au2_gt 1900	GM_FromMeshes: 1900	au2_gt	500	360	120	17.0	288.9	138.0			
NN, au2_gt 2400	GM_FromMeshes: 2400	au2_gt	260	140	240	17.0	288.9	138.0			
NN, au2_gt 2601	GM_FromMeshes: 2601	au2_gt	400	300	240	17.0	288.9	138.0			
NN, au2_gt 2602	GM_FromMeshes: 2602	au2_gt	400	300	220	83.8	79.8	4.6			
NN, au2_gt 2900	GM_FromMeshes: 2900	au2_gt	500	200	270	11.5	344.8	67.1			
NN, au2_gt 32_33_36	GM_Consolidated_from_meshes: 32_33_36	au2_gt	470	360	240	44.9	73.7	174.3			
NN, au2_gt 3500	GM_FromMeshes: 3500	au2_gt	400	350	250	21.4	231.4	32.6			
NN, au2_gt 3900	GM_Consolidated_from_meshes: 3900	au2_gt	550	550	150	44.3	85.5	170.1			
NN, au2_gt 4601	GM_FromMeshes: 4601	au2_gt	320	500	120	87.9	281.2	0.8			
NN, au2_gt 4901	GM_FromMeshes: 4901	au2_gt	320	300	260	78.8	95.6	176.7			
NN, au2_gt 5601	GM_FromMeshes: 5601	au2_gt	300	240	110	17.0	288.9	138.0			
NN, au2_gt 5602	GM_FromMeshes: 5602	au2_gt	280	240	110	17.5	355.3	87.4			
NN, au2_gt 5603	GM_FromMeshes: 5603	au2_gt	280	240	110	17.5	355.3	77.8			
NN, au2_gt 5604	GM_FromMeshes: 5604	au2_gt	280	240	110	17.5	355.3	85.0			
NN, au2_gt 5605	GM_FromMeshes: 5605	au2_gt	280	240	110	17.5	355.3	92.3			

table continues…

General			Ellipsoid Ranges			Ellipsoid Directions				Number of Samples	
Name	Domain	Values	Maximum	Intermediate	Minimum	Dip	Dip Azimuth	Pitch	Variable Orientation	Minimum	Maximum
NN, au2_gt 5901_5902	GM_Consolidated_from_meshes: 5901	au2_gt	440	330	250	19.6	324.9	65.3			
NN, au2_gt 6900_7500	GM_Consolidated_from_meshes: 6900_7500	au2_gt	500	400	270	6.1	31.7	23.4			
NN, au2_gt 7900	GM_Consolidated_from_meshes: 7900	au2_gt	550	800	240	74.2	77.7	6.7			
NN, au2_gt 8900	GM_FromMeshes: 8900	au2_gt	550	450	80	74.2	77.7	6.7			
NN, au2_gt 9900	GM_FromMeshes: 9900	au2_gt	550	450	220	15.5	205.0	63.8			
NN, cu_pct 1000	GM_Consolidated_from_meshes: 1000	cu_pct	160	100	50	27.7	249.8	65.2			
NN, cu_pct 11_12_13_16	GM_Consolidated_from_meshes: 11_12_13_16	cu_pct	269.2	203.3	131.7	17.0	288.9	138.0			
NN, cu_pct 1900	GM_FromMeshes: 1900	cu_pct	500	400	150	17.0	288.9	138.0			
NN, cu_pct 2400	GM_FromMeshes: 2400	cu_pct	188	109.4	146.7	17.0	288.9	138.0			
NN, cu_pct 2601	GM_FromMeshes: 2601	cu_pct	400	300	240	17.0	288.9	138.0			
NN, cu_pct 2602	GM_FromMeshes: 2602	cu_pct	400	300	220	83.8	79.8	4.6			
NN, cu_pct 2900	GM_FromMeshes: 2900	cu_pct	500	400	120	11.5	344.8	67.1			
NN, cu_pct 32_33_36	GM_Consolidated_from_meshes: 32_33_36	cu_pct	550	450	300	44.9	73.7	174.3			
NN, cu_pct 3500	GM_FromMeshes: 3500	cu_pct	400	380	180	21.4	231.4	32.6			
NN, cu_pct 3900	GM_Consolidated_from_meshes: 3900	cu_pct	550	550	180	44.3	85.5	170.1			
NN, cu_pct 4601	GM_FromMeshes: 4601	cu_pct	400	230	120	87.9	281.2	0.8			
NN, cu_pct 4901	GM_FromMeshes: 4901	cu_pct	300	300	250	78.8	95.6	176.7			
NN, cu_pct 5601	GM_FromMeshes: 5601	cu_pct	300	240	110	17.0	288.9	138.0			
NN, cu_pct 5602	GM_FromMeshes: 5602	cu_pct	280	240	110	17.5	355.3	87.4			
NN, cu_pct 5603	GM_FromMeshes: 5603	cu_pct	280	240	110	17.5	355.3	77.8			
NN, cu_pct 5604	GM_FromMeshes: 5604	cu_pct	280	240	110	17.5	355.3	85.0			
NN, cu_pct 5605	GM_FromMeshes: 5605	cu_pct	280	240	110	17.5	355.3	92.3			
NN, cu_pct 5901_5902	GM_Consolidated_from_meshes: 5901	cu_pct	500	400	120	19.6	324.9	110.1			
NN, cu_pct 6900_7500	GM_Consolidated_from_meshes: 6900_7500	cu_pct	500	400	240	6.1	31.7	23.4			
NN, cu_pct 7900	GM_Consolidated_from_meshes: 7900	cu_pct	500	450	240	74.2	77.7	6.7			
NN, cu_pct 8900	GM_FromMeshes: 8900	cu_pct	550	450	150	74.2	77.7	6.7			
NN, cu_pct 9900	GM_FromMeshes: 9900	cu_pct	550	450	150	15.5	205.0	63.8			
NN, mo_pct 1000	GM_Consolidated_from_meshes: 1000	mo_pct	160	100	50	27.7	249.8	65.2			

table continues…

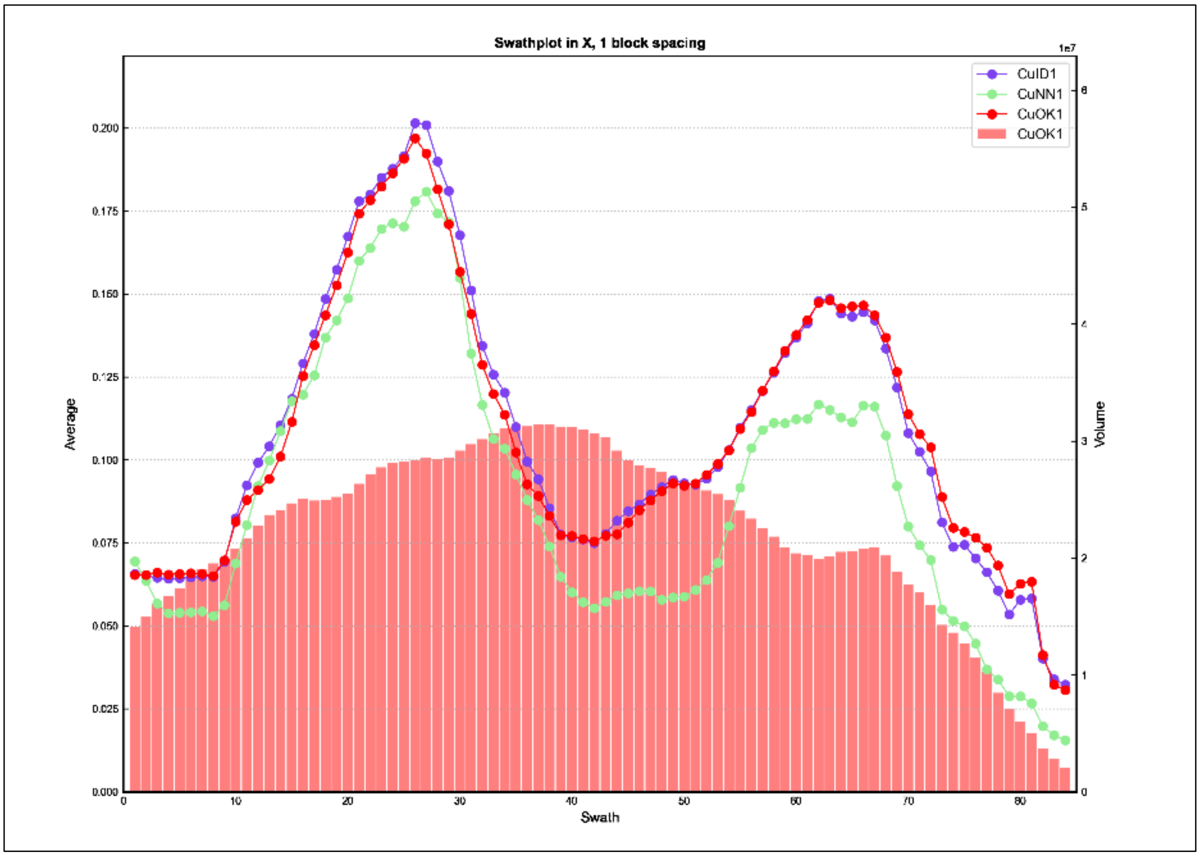
General			Ellipsoid Ranges			Ellipsoid Directions			Variable Orientation	Number of Samples	
Name	Domain	Values	Maximum	Intermediate	Minimum	Dip	Dip Azimuth	Pitch		Minimum	Maximum
NN, mo_pct 11_12_13_16	GM_Consolidated_from_meshes: 11_12_13_16	mo_pct	600	480	300	17.0	288.9	154.3			
NN, mo_pct 1900	GM_FromMeshes: 1900	mo_pct	440	400	200	17.0	288.9	138.0			
NN, mo_pct 2400	GM_FromMeshes: 2400	mo_pct	300	260	330	17.0	288.9	147.5			
NN, mo_pct 2601	GM_FromMeshes: 2601	mo_pct	360	260	300	17.0	288.9	154.2			
NN, mo_pct 2602	GM_FromMeshes: 2602	mo_pct	400	300	220	83.8	79.8	4.6			
NN, mo_pct 2900	GM_FromMeshes: 2900	mo_pct	600	550	400	11.5	344.8	67.1			
NN, mo_pct 32_33_36	GM_Consolidated_from_meshes: 32_33_36	mo_pct	360	350	350	44.9	73.7	69.1			
NN, mo_pct 3500	GM_FromMeshes: 3500	mo_pct	780	700	400	21.4	231.4	24.9			
NN, mo_pct 3900	GM_Consolidated_from_meshes: 3900	mo_pct	270	440	120	44.3	85.5	170.1			
NN, mo_pct 4601	GM_FromMeshes: 4601	mo_pct	430	300	170	87.9	281.2	14.7			
NN, mo_pct 4901	GM_FromMeshes: 4901	mo_pct	400	360	200	78.8	95.6	23.7			
NN, mo_pct 5601	GM_FromMeshes: 5601	mo_pct	300	240	110	17.5	355.3	109.8			
NN, mo_pct 5602	GM_FromMeshes: 5602	mo_pct	280	240	110	17.5	355.3	108.2			
NN, mo_pct 5603	GM_FromMeshes: 5603	mo_pct	280	240	110	17.5	355.3	77.8			
NN, mo_pct 5604	GM_FromMeshes: 5604	mo_pct	280	240	110	17.5	355.3	85.0			
NN, mo_pct 5605	GM_FromMeshes: 5605	mo_pct	280	240	110	17.5	355.3	92.3			
NN, mo_pct 5901_5902	GM_Consolidated_from_meshes: 5901	mo_pct	500	400	250	19.6	324.9	110.1			
NN, mo_pct 6900_7500	GM_Consolidated_from_meshes: 6900_7500	mo_pct	500	400	260	6.1	31.7	23.4			
NN, mo_pct 7900	GM_Consolidated_from_meshes: 7900	mo_pct	500	500	250	74.2	77.7	6.7			
NN, mo_pct 8900	GM_FromMeshes: 8900	mo_pct	500	450	140	74.2	77.7	6.7			
NN, mo_pct 9900	GM_FromMeshes: 9900	mo_pct	550	450	150	15.5	205.0	63.8			

14.9 Block Model Validation

Several validation techniques have been utilized to ensure that the estimates are reasonable.

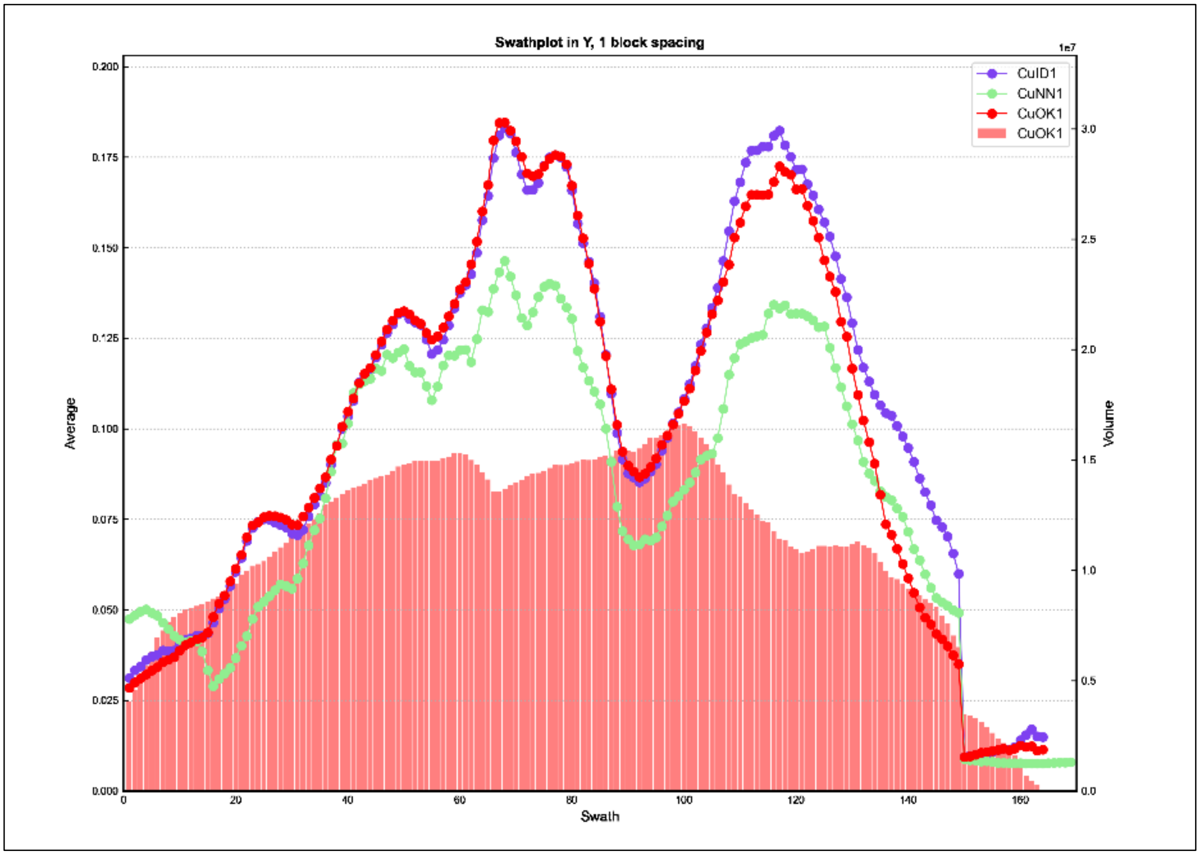
- Swath plots comparing composite grade to the kriged estimate in corridors in the X, Y, and Z directions. Comparison was also made with inverse distance to the second power estimates and NN estimates (representing declustered composite grades). Examples are shown in Figure 14-5 to Figure 14-7.
- Comparison with block estimates from the previous mineral resource estimates in 2011 and 2018.
- Visual comparisons on section and in plan (examples are shown in Figure 14-8 to Figure 14-11).
- Comparison of grade–tonnage curves for the kriged estimates, the previous mineral resource estimates, and the inverse distance estimates (see Figure 14-12 to Figure 14-15).

Figure 14-5: Copper Eastings Swathplot Example (All Estimated Blocks, Bars Represent Number of Blocks)



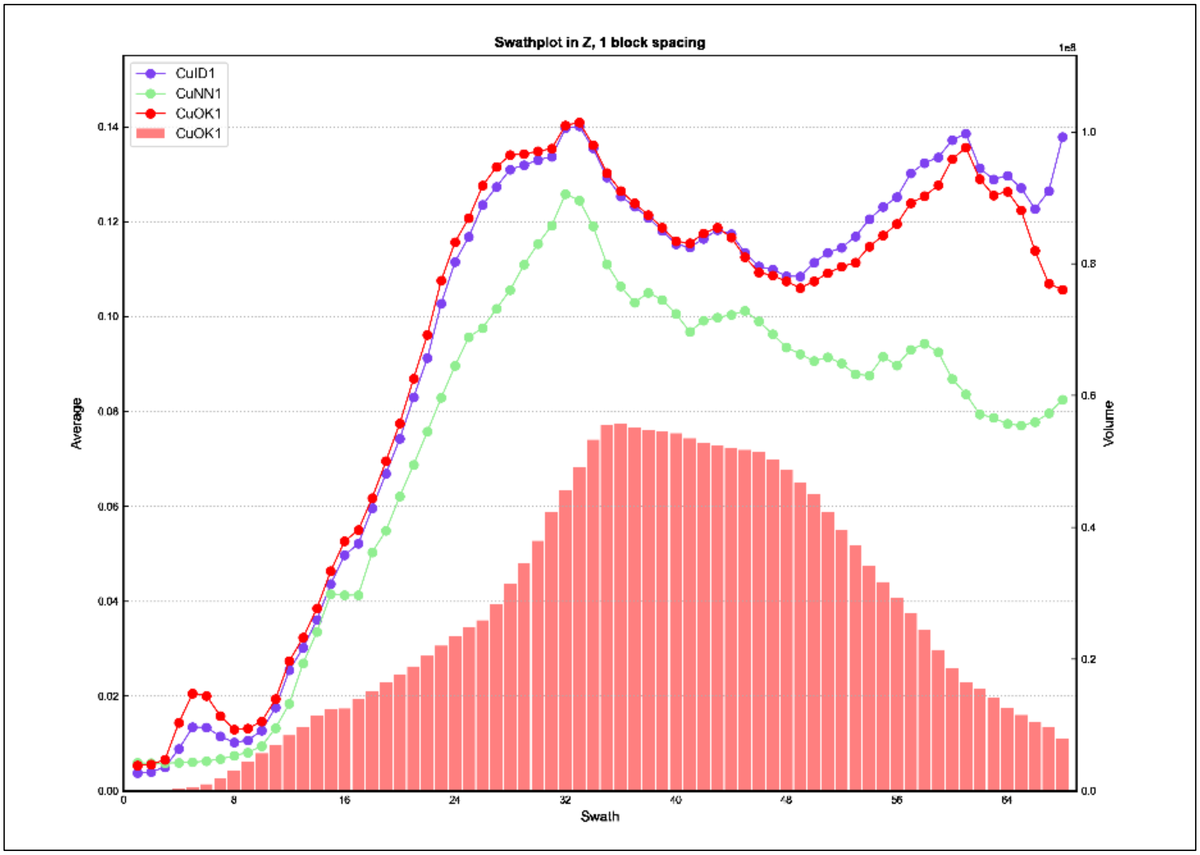
Source: Red Pennant

Figure 14-6: Copper Northings Swathplot Example



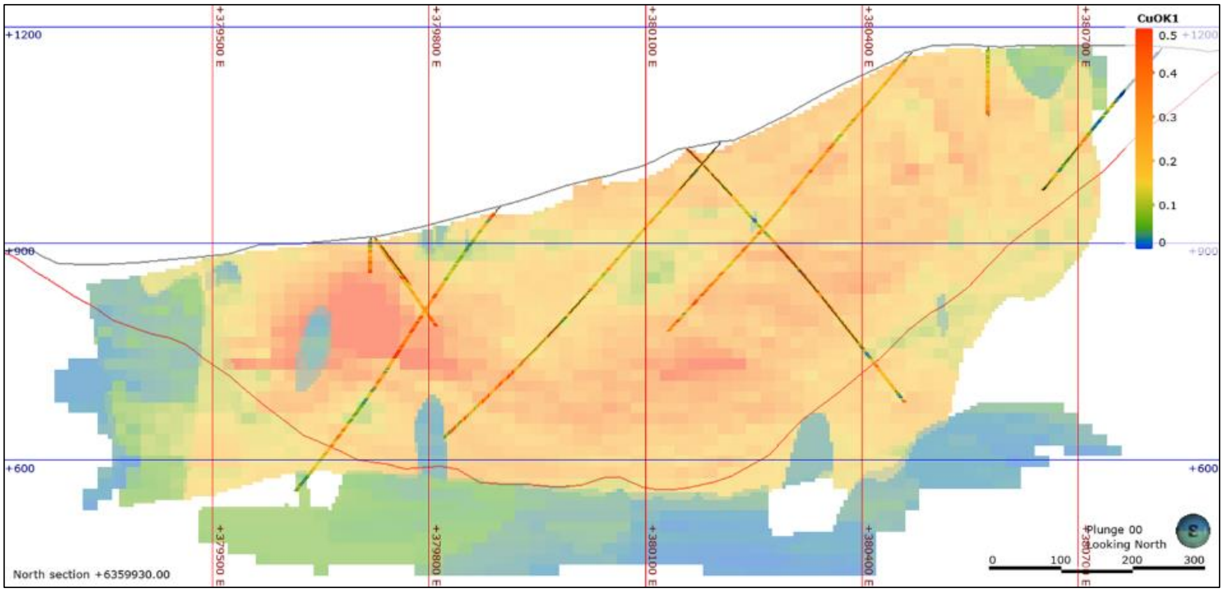
Source: Red Pennant

Figure 14-7: Copper Elevations Swathplot Example



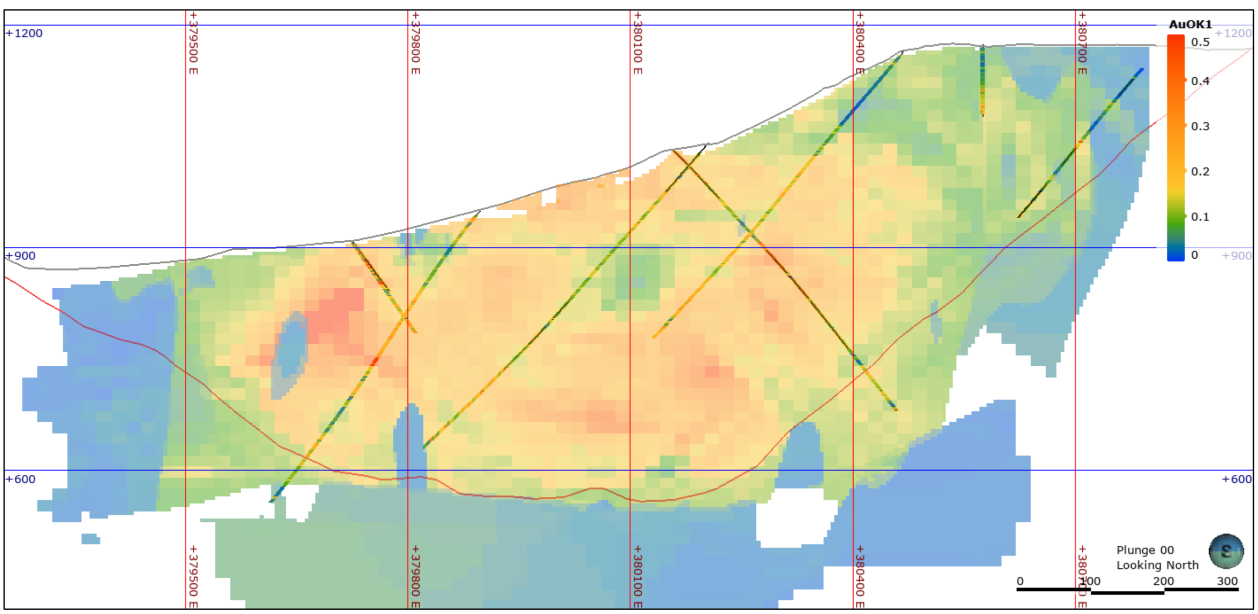
Source: Red Pennant

Figure 14-8: Copper Kriged Block Estimates and Drill Data West-East Section Y+6,359,930 (50 m wide)



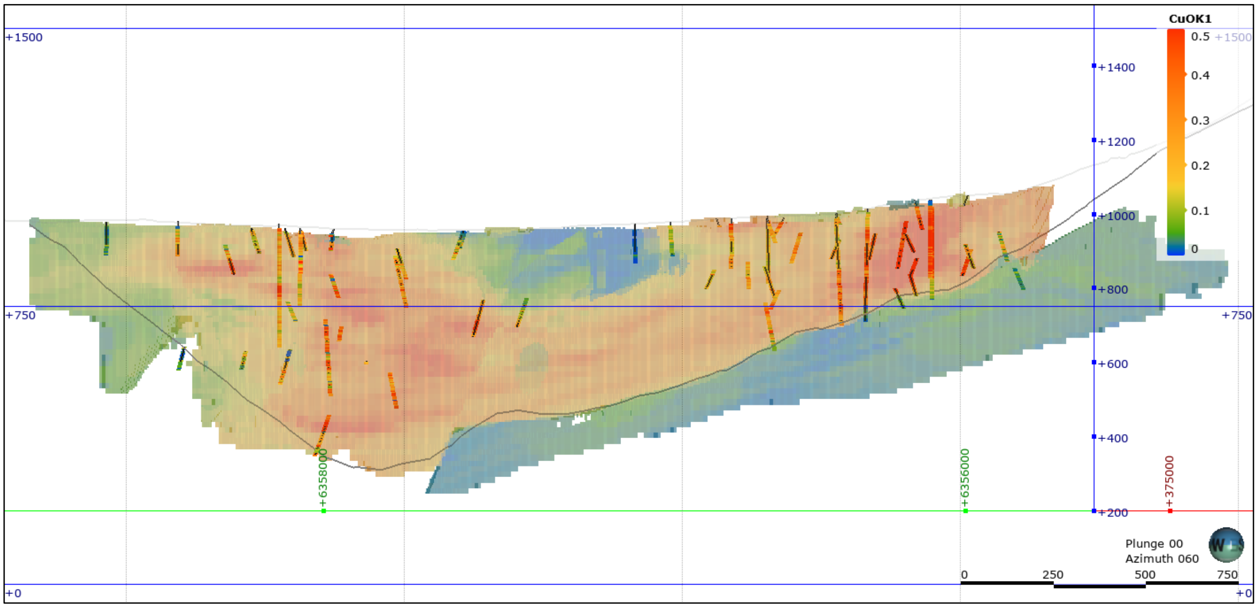
Source: Red Pennant

Figure 14-9: Gold Kriged Block Estimates and Drill Data West-East Section Y+6,359,930



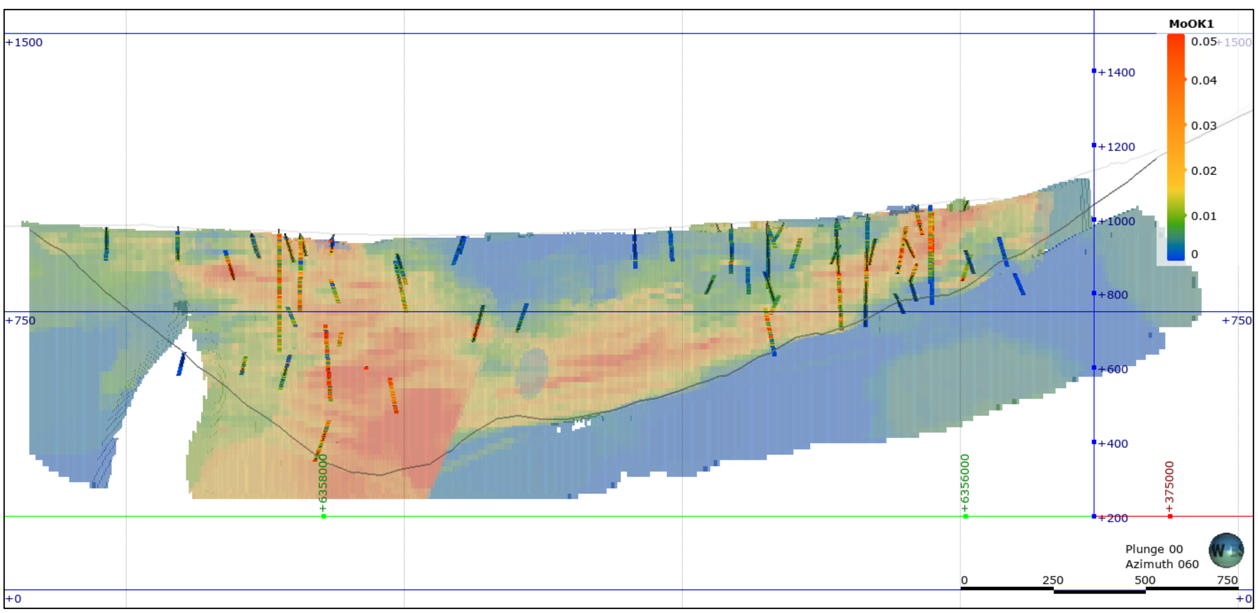
Source: Red Pennant

Figure 14-10: Copper Kriged Block Estimates and Drill Data Paramount-Liard NNW-SSE Section (X+379,872 Y+6,360,170 Looking to 060, 50 m wide)



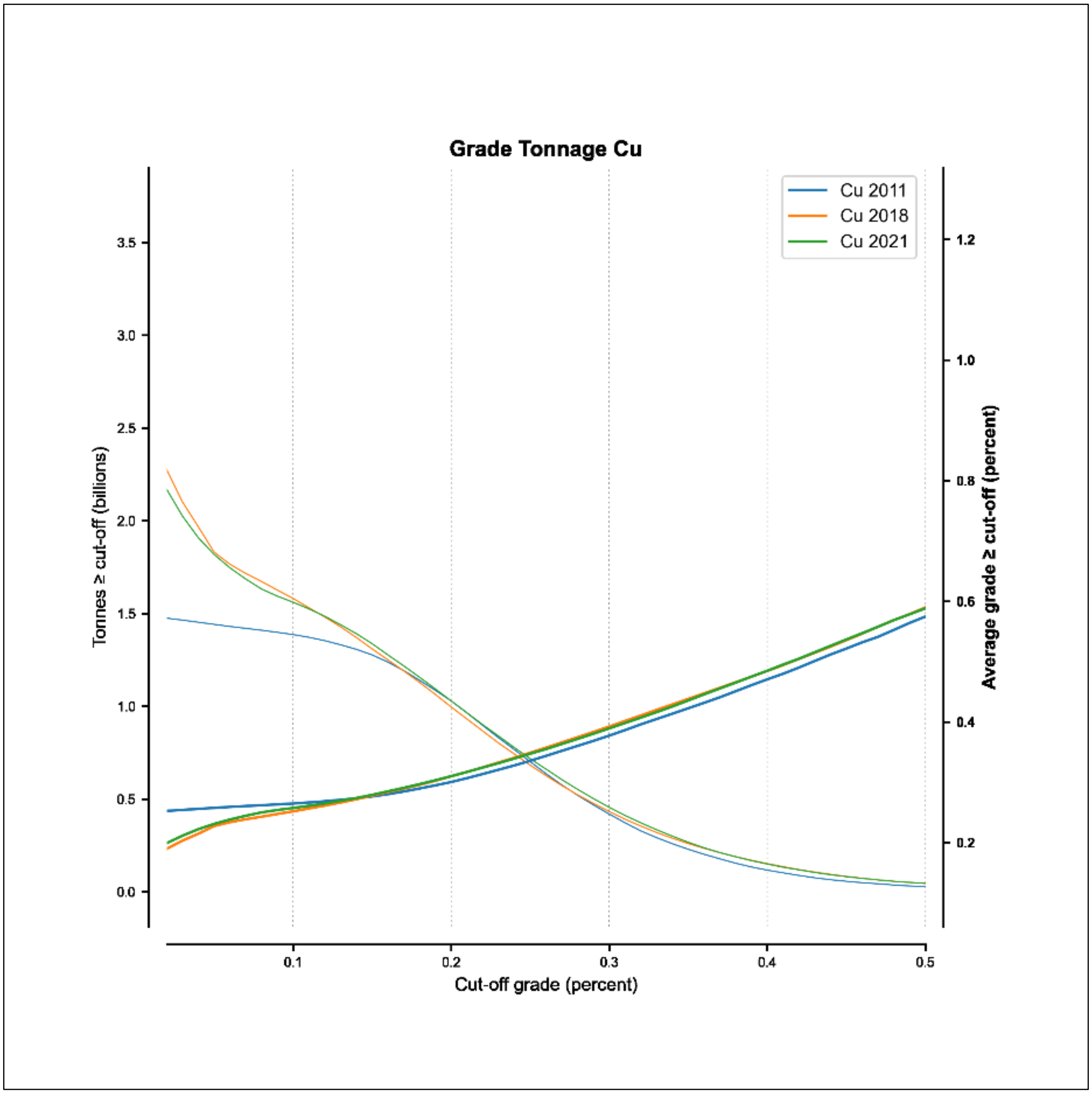
Source: Red Pennant

Figure 14-11: Molybdenum Kriged Block Estimates and Drilled Data Paramount-Liard NNW-SSE Section (X+379,872 Y+6,360,170 Looking to 060)



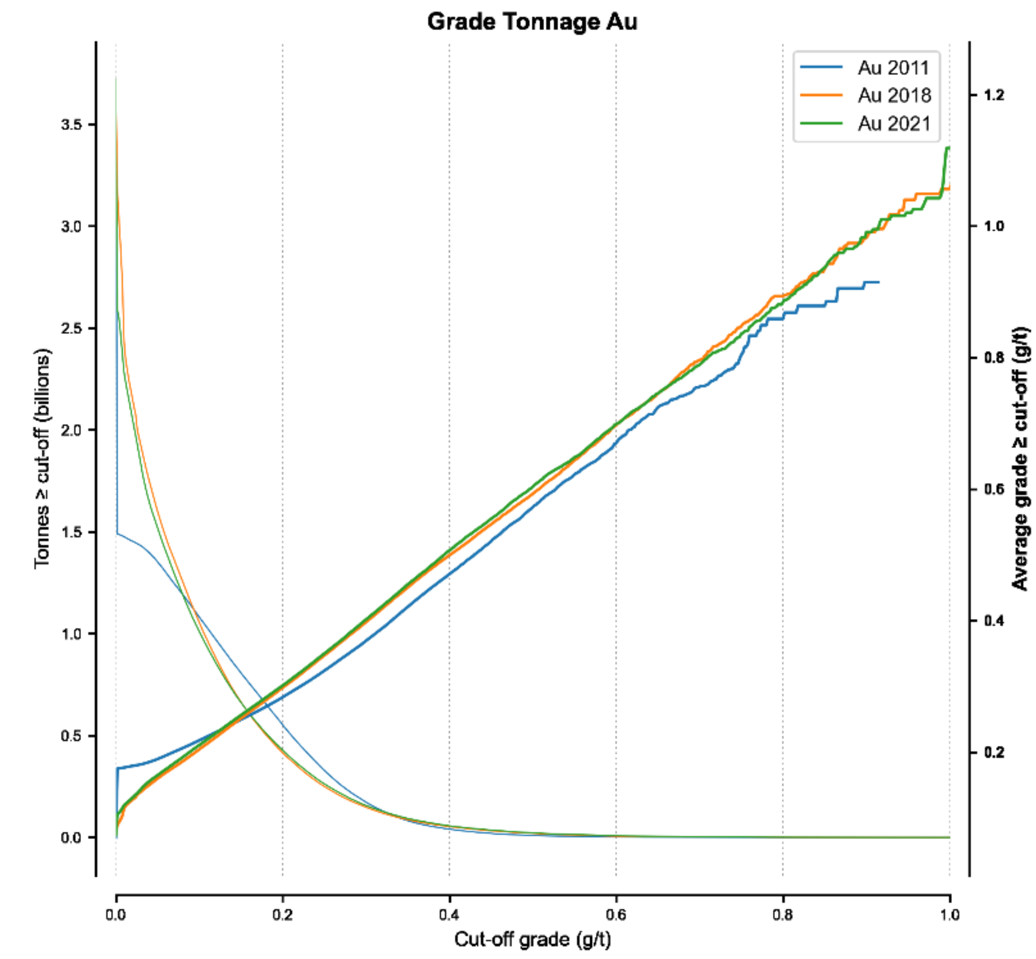
Source: Red Pennant

Figure 14-12: Grade Tonnage Comparison for Copper Block Estimates for the Current Estimates and Previous 2018 and 2011 Estimates



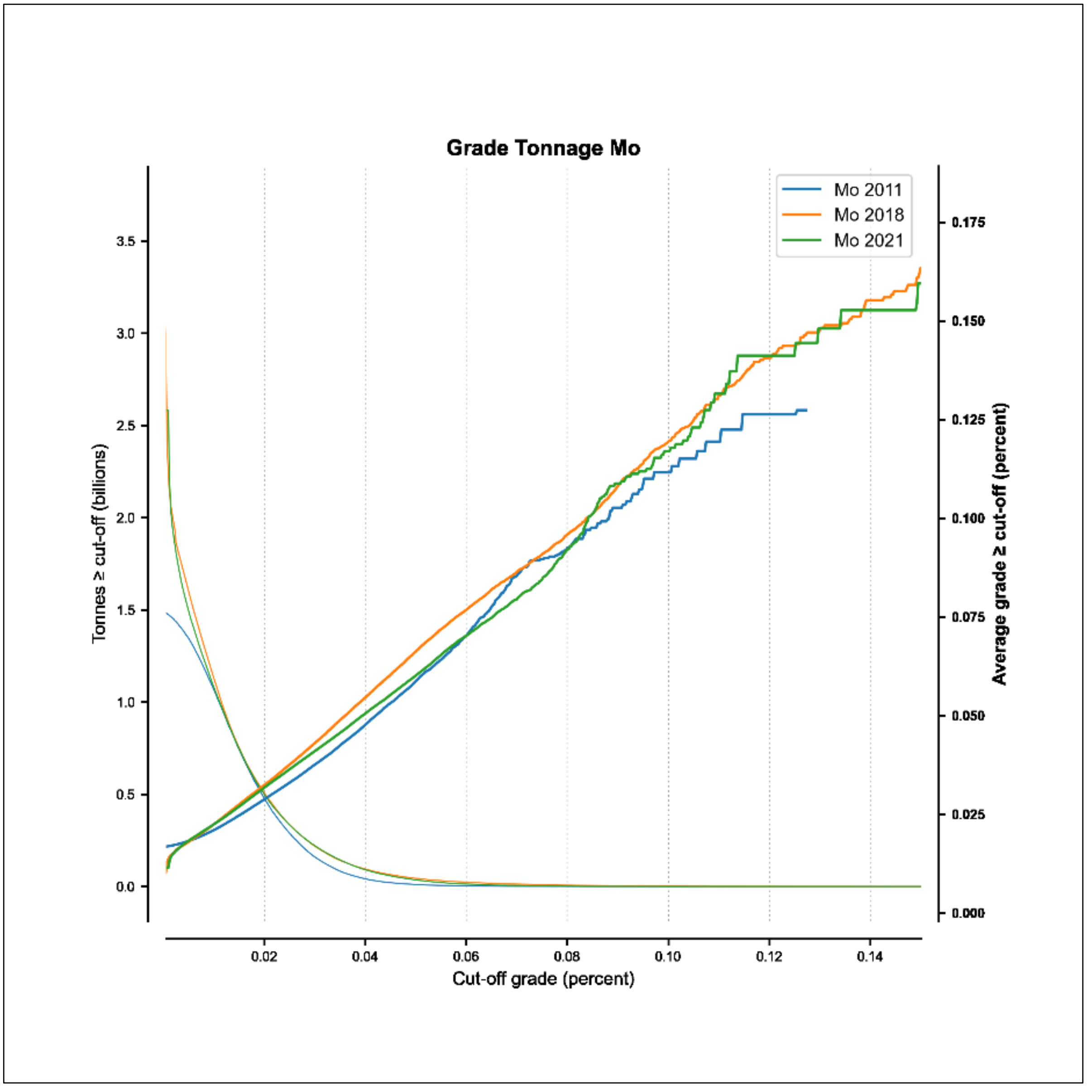
Source: Red Pennant

Figure 14-13: Grade Tonnage Comparison for Gold Block Estimates for the Current Estimates and Previous 2018 and 2011 Estimates



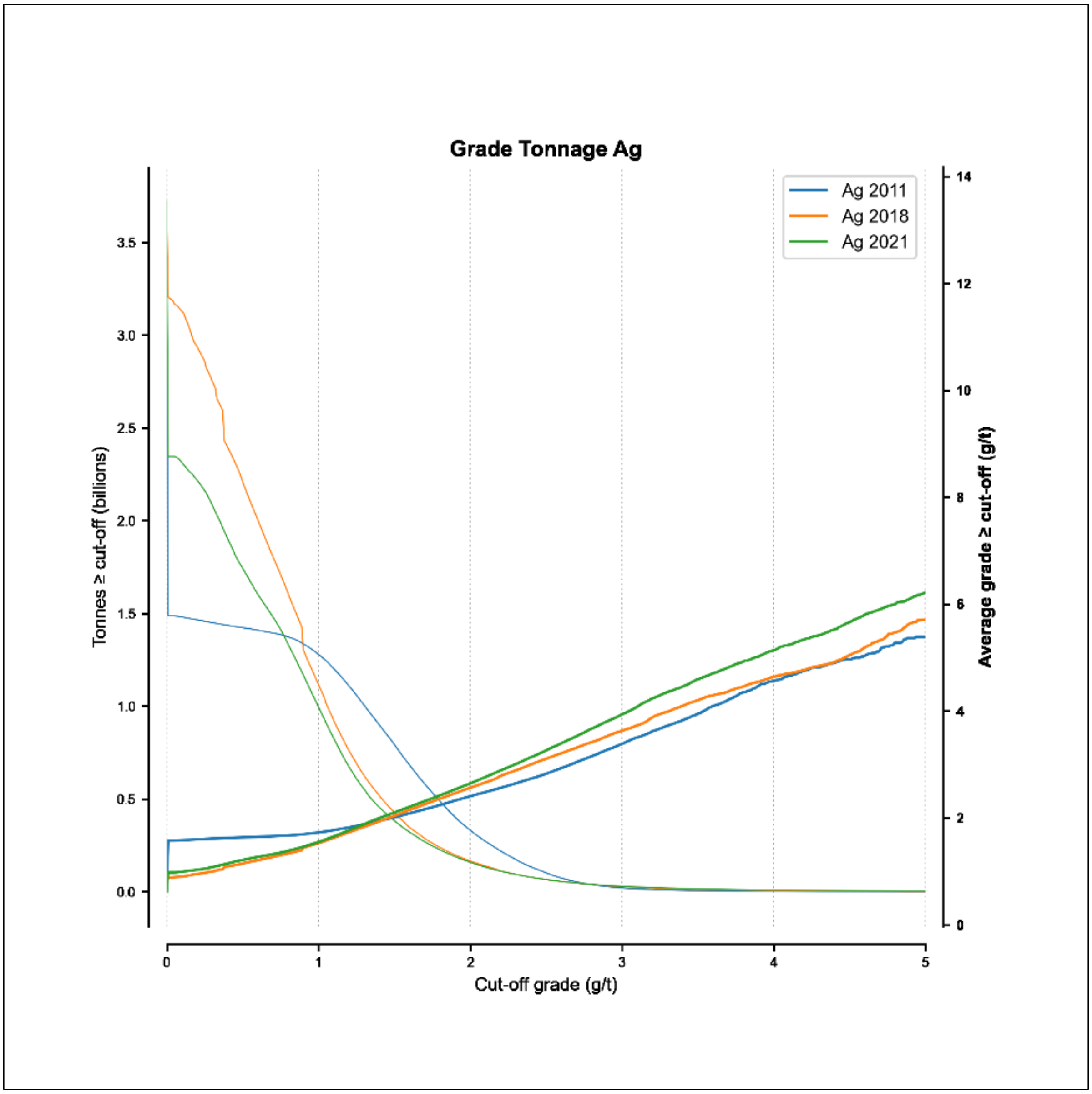
Source: Red Pennant

Figure 14-14: Grade Tonnage Comparison for Molybdenum Block Estimates for the Current Estimates and Previous 2018 and 2011 Estimates.



Source: Red Pennant

Figure 14-15: Grade Tonnage Comparison for Silver Block Estimates for the Current Estimates and Previous 2018 and 2011 Estimates



Source: Red Pennant

The validations indicated the block model estimates are reasonable.

14.10 Classification of Mineral Resources

The Kriging Slope of Regression was generated during the estimation and is a measure of confidence related to distance from data in the context of the spatial uncertainty (corresponding with the variogram models). The copper Kriging Slope of Regression was used as the main driver of uncertainty.

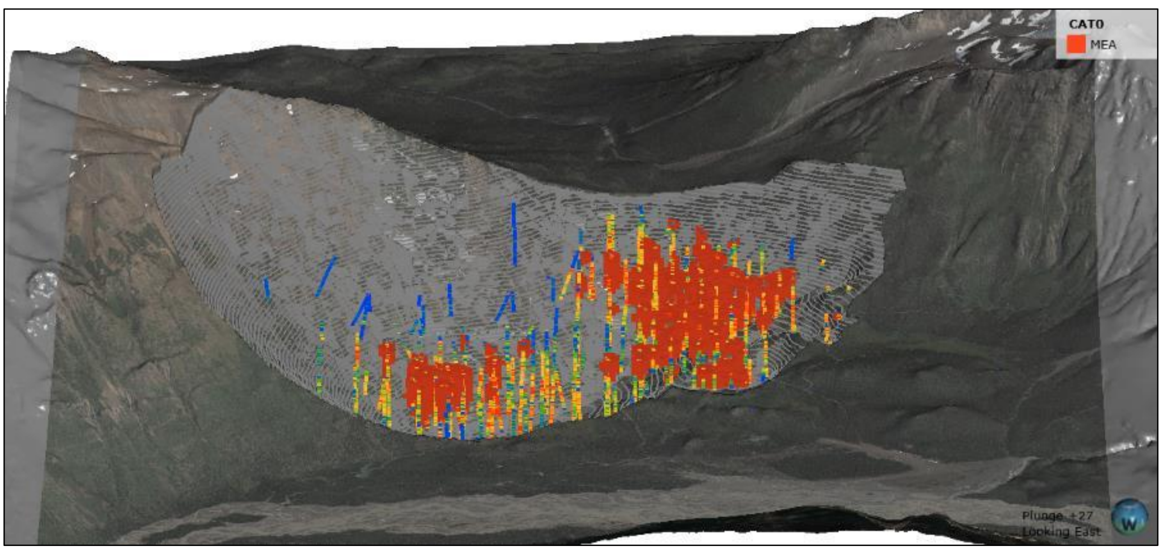
Measured, indicated, and inferred mineral resources were classified using the parameters outlined in Table 14-10.

Table 14-10: Confidence Classification Criteria

Categories	Average Distance to Estimation Composites	Slope of Regression
Measured	<48 m	>0.85
Indicated	48 - 128 m	0.6 - 0.85
Inferred	128 - 500 m	0.05 - 0.6

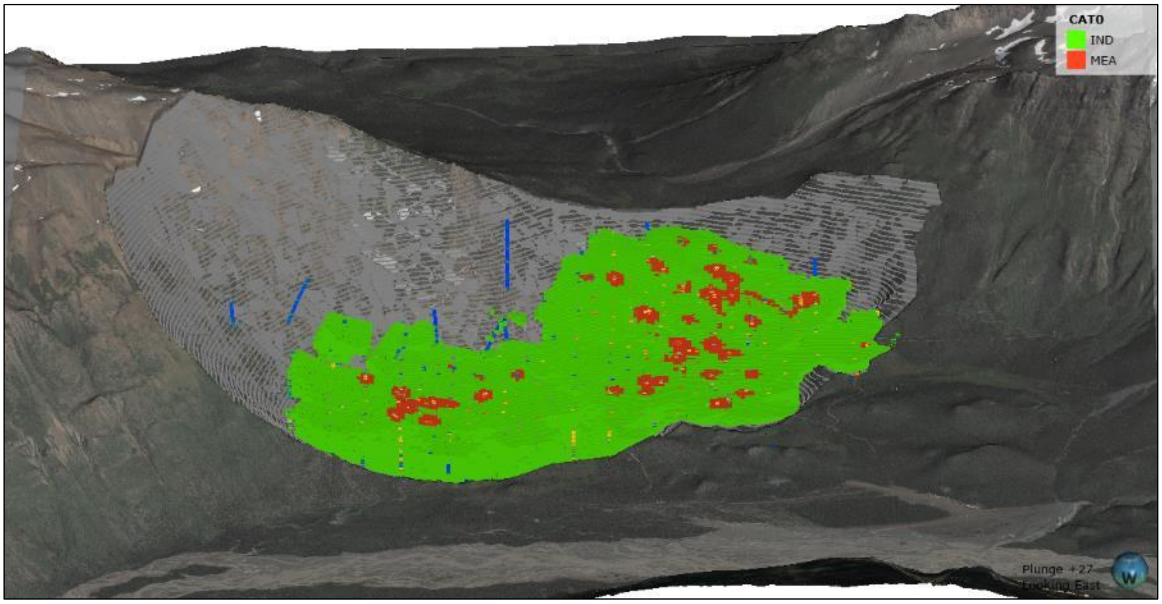
The extents of measured, indicated, and inferred mineral resources within the conceptual pit are illustrated in Figure 14-16 to Figure 14-18, inclusive.

Figure 14-16: View of Measured Mineral Resource Looking East



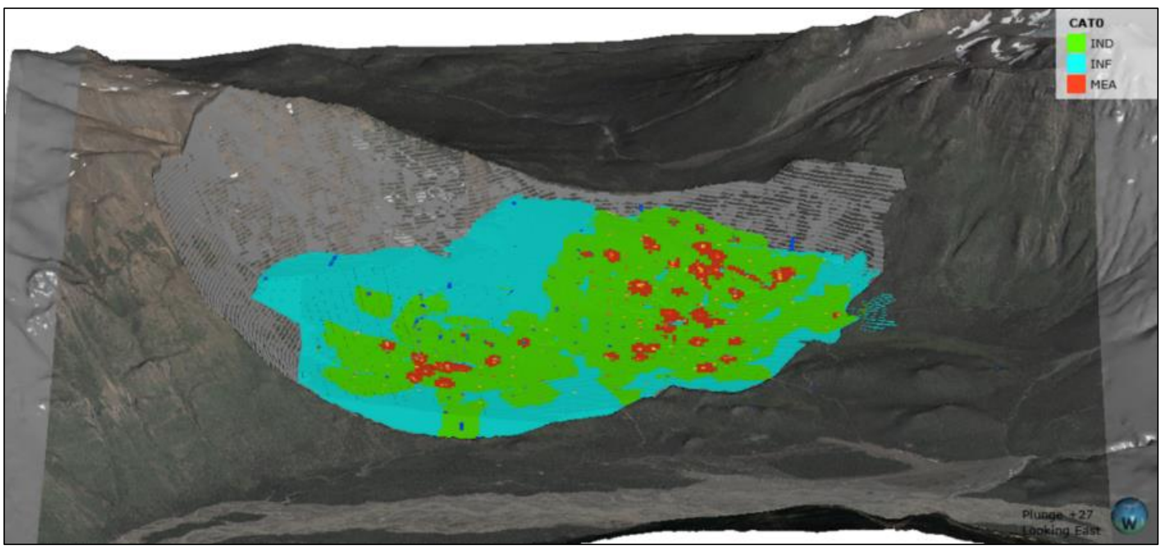
Source: Red Pennant

Figure 14-17: View of Measured and Indicated Mineral Resources Looking East



Source: Red Pennant

Figure 14-18: View of Measured, Indicated, and Inferred Mineral Resources Looking East



Source: Red Pennant

14.11 Reasonable Prospects of Eventual Economic Extraction

Mineral Resources were constrained using a conceptual ultimate pit shell based on the input parameters listed in Table 14-11. These values were used to calculate a block-by-block NSR for copper and molybdenum concentrates. The parameters and pit shell are the same as the pit shell used to constrain the mineral resources in 2018.

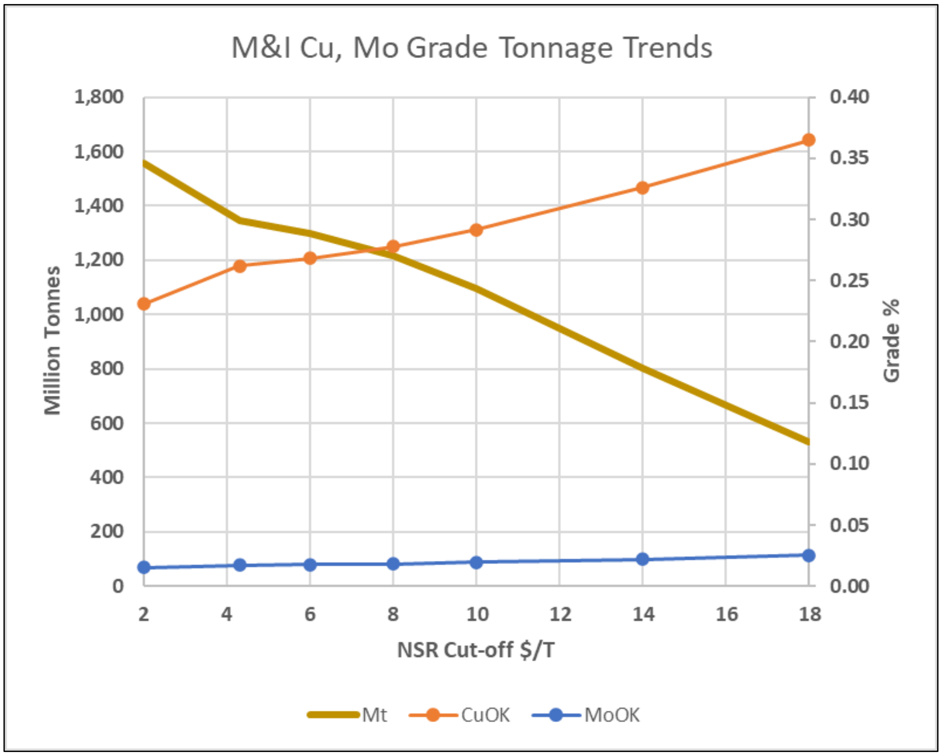
Table 14-11: Conceptual Pit Shell Input Parameters

	Units	Values
Metal Prices		
Copper	USD$/lb	3.00
Molybdenum	USD$/lb	10.00
Gold	USD$/oz	1,200.00
Silver	USD$/oz	20.00
Mill Feed Rate		
Tonnes per annum	Mt	47.50
Financial Analysis Assumptions		
Discount rate	%	8.00
Exchange rate	$CAD/$US	1.20
Copper Concentrate		
Copper concentrate grade	%	28.00
Copper recovery	%	90.00*
Molybdenum Concentrate		
Molybdenum concentrate grade	%	50.00
Molybdenum recovery	%	85.00*
Unit Costs		
Processing and general and administrative costs	USD$/t	4.31
Mining cost	USD$/t	1.95
Pit Slopes Whittle		
All units (overall slope angle [OSA])	degrees	40 to 44

14.12 Resource Sensitivity to Cut-off

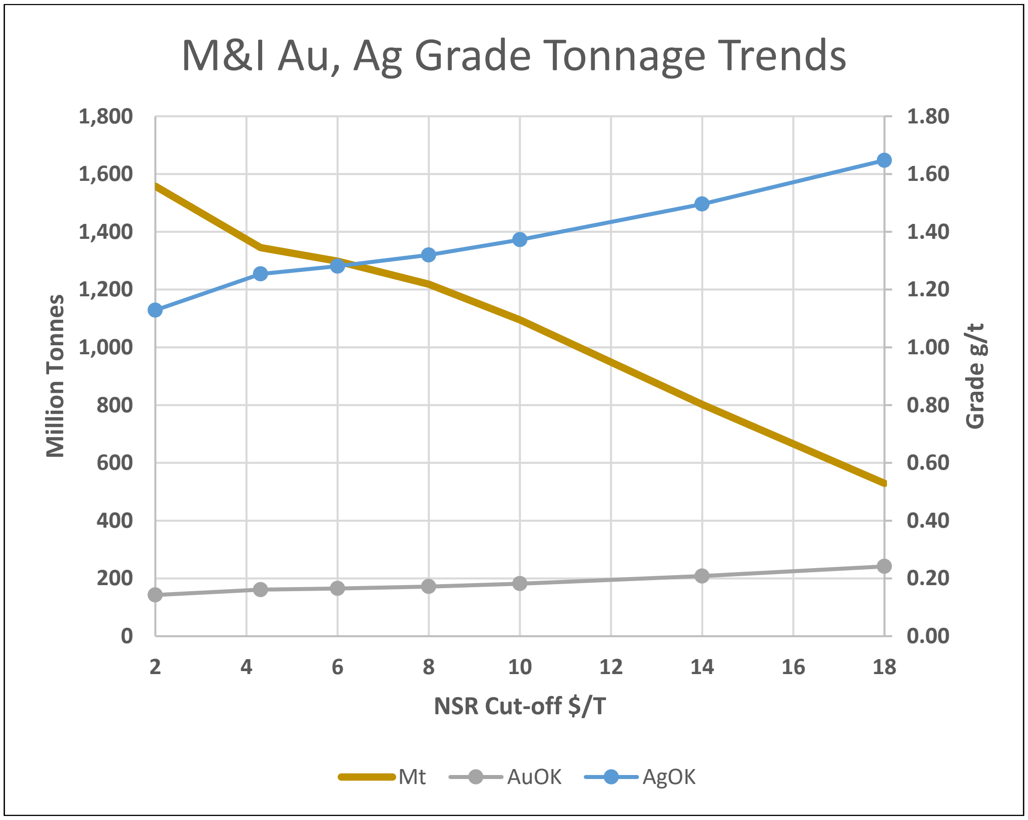
The tonnage and grade sensitivities at different cut-off (NSR) values are illustrated in Figure 14-19 and Figure 14-20, for measured and indicated mineral resources and in Figure 14-21 and Figure 14-22, for inferred mineral resources within the conceptual resource pit shell.

Figure 14-19: Measured and Indicated Cu, Mo Grade, and Tonnage Trends at Different NSR Cut-offs



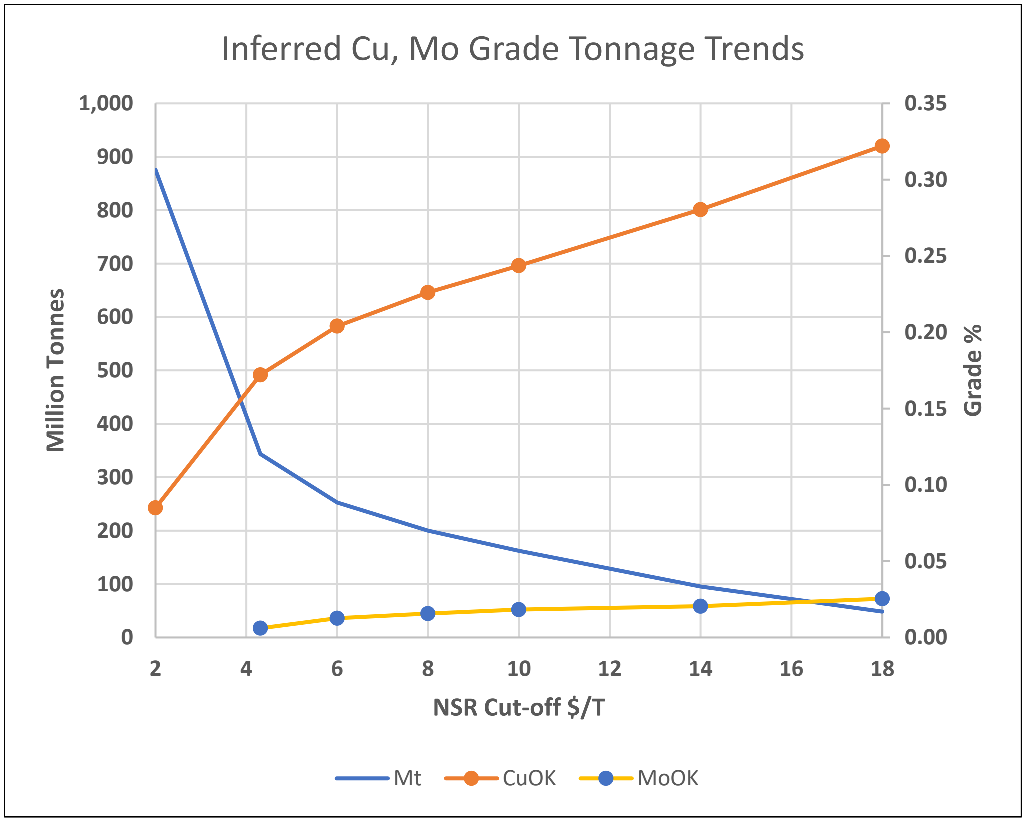
Source: Red Pennant

Figure 14-20: Measured and Indicated Au, Ag Grade, and Tonnage Trends at Different NSR Cut-offs



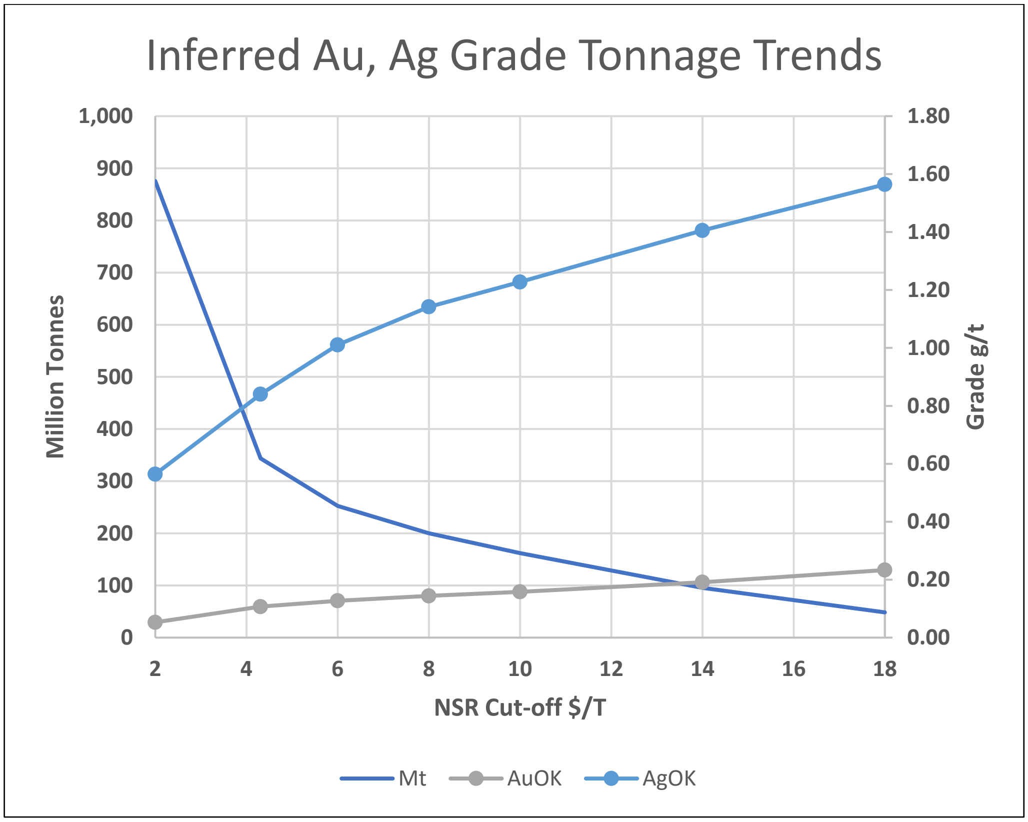
Source: Red Pennant

Figure 14-21: Inferred Material Cu, Mo Grade, and Tonnage Trends at Different NSR Cut-offs



Source: Red Pennant

Figure 14-22: Inferred Material Au, Ag Grade, and Tonnage Trends at Different NSR Cut-offs



Source: Red Pennant

14.13 Mineral Resource Statement

The Mineral Resource was estimated in January 2021 and constrained within an optimized ultimate pit shell and are summarized in Table 14-12.

Table 14-12: Mineral Resource Statement

Category	Mass	Average Value				Material Content			
		Cu	Au	Mo	Ag	Cu	Au	Mo	Ag
	Mt	%	g/t	%	g/t	million lb	million t. oz	million lb	million t. oz
Measured	176	0.32	0.22	0.018	1.46	1,262	1.28	71	8.26
Indicated	1,169	0.25	0.15	0.017	1.22	6,503	5.69	440	46.00
Total M&I	**1,346**	**0.26**	**0.16**	0.017	**1.25**	**7,764**	**6.97**	**511**	**54.25**
Inferred	344	0.17	0.11	0.013	0.84	1,303	1.18	96	9.28

Notes:
1. Mineral Resources are reported using the 2014 CIM Definition Standards.
2. The QP for the estimate is Mr. Michael F. O'Brien, P.Geo., Red Pennant Resources Geoscience. Mineral Resources have an effective date of 15 January 2021.
3. Mineral Resources are reported within a conceptual constraining pit shell that includes the following input parameters: $3/lb Cu, $10/lb Mo, $1,200/oz Au, $20/oz Ag, mining cost of CAD$1.95/t mined, processing cost of CAD$4.94/t processed and pit slope angles that vary from 40–44º. Metal recoveries; Cu 86.6%, Au 73%, Mo 58.8%, Ag 48.3%.
4. Mineral Resources are reported using a net smelter return cut-off of USD$4.31/t, and a CAD$ to USD$ exchange rate of 1.20. 5. Metal prices are in $US.
6. Tonnes are metric tonnes, with copper and molybdenum grades as percentages, and gold and silver grades as gram per tonne units. Copper and molybdenum metal content is reported in lbs and gold and silver content is reported in troy ounces.
7. Totals may not sum due to rounding.

14.14 Factors That May Affect the Mineral Resource Estimate

Areas of uncertainty that may materially impact the Mineral Resource estimates include:

- Changes to long-term metal price assumptions
- Changes in local interpretations of mineralization geometry, fault geometry, and continuity of mineralized zones
- Changes to the NSR used to constrain the estimates
- Changes to the regression equation used to fill in missing gold and silver values
- Changes to metallurgical recovery assumptions
- Changes to the input assumptions used to derive the conceptual open pit outlines used to constrain the estimate
- Changes to resource classification approach
- Variations in geotechnical, hydrogeological, and mining assumptions

- Changes to environmental, permitting, and social license assumptions

14.15 Comments on Section 14.0

The QP believes that the Mineral Resources have been estimated using good industry practice and conform to the 2014 CIM Definition Standards. Mineral Resources are constrained by reasonable open pit mining assumptions.

There are no other known environmental, legal, title, taxation, socioeconomic, marketing, political or other relevant factors that would materially affect the estimation of Mineral Resources that are not discussed in this Report.

15.0 MINERAL RESERVE ESTIMATE

A Mineral Reserve has not been estimated for the Project as part of this PEA.

A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource.

16.0 MINING METHODS

16.1 Introduction

The 2021 PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the 2021 PEA results will be realized.

The 2013 mine planning completed by MMTS was based on the 3D block model created by Tetra Tech. A 130,000 tpd throughput was used in 2013 FS as the mineral processing rate for the project. The 2013 FS focused on a 21 year mine life.

The 2013 FS included approximately 1.0 Bt of mineralized material and 2.0 Bt of waste with a LOM strip ratio of 2.01 W:O. Due to the LOM strip ratio, the project required three RSFs which resulted in significant waste haulage over the life of the mine.

The 2021 mine design and production plan utilized the updated geological and resource models that provided a better understanding of the extent and boundaries of the deposit compared to the 2013 FS. The 2021 PEA takes into consideration the entire updated resource, including measured, indicated, and inferred material, where previously only measured and indicated resources were considered in the 2013 FS. The current work has led to a re-centering of the pit to the west, creating a significant reduction to the LOM strip ratio.

The mine plan includes a variable throughput rate for mineral processing based on the current lithological units and material hardness outlined in Table 17-2, Section 17.0 of this report. Haulage requirements have been updated, including the haulage of ROM NPAG material for the construction of the North and South TSF embankments.

With the redesign and re-centering of the LOM pit, the mine plan now includes approximately 1.0 Bt of waste and 1.0 Bt of mineralization for a LOM strip ratio of 1.0 W:O. With the significant reduction in waste t, the updated mine plan was able to remove the requirement of one of the RSFs. With the removal of the furthest RSF, the total waste haulage requirement has been significantly reduced along with a reduction in the total site footprint.

16.2 Block Model Description

A 20 m x 20 m x 15 m block model was used for pit shell generation, pit design, and production scheduling. Details of the resource estimation and block modeling are documented in Section 14.0 of this report. The updated resource estimate and Technical Report, titled, "Mineral Resource Estimate for the Schaft Creek Property, B.C., Canada" with an effective date of January 15, 2021, was prepared by H. Ghaffari, MASc., P.Eng. and J. Huang, PhD., P.Eng, of Tetra Tech and Michael F. O'Brian, P.Geo., of Red Pennant Geo Sciences.

16.3 2021 Mine Optimization

Optimization of the final pit and the interim phases for Schaft Creek were developed in the Whittle® software package. The current PEA study updated the NSR values into the block model and these NSR values were the input for the block value used in generating the final pit with approximately 1.0 Bt of mill feed. A comparison of the pit parameters between the 2013 FS and the current study are set out in Table 16-1.

Table 16-1: Pit Optimization Comparison Results

	Unit	2013 FS	2021 PEA
Waste	Mt	1,890	1,029
Mineralized Material	Mt	940.8	1,030
Total Tonnes	Mt	2,831	2,059
SR W/O		2.00	1.00

[1]Note: These are results from Whittle Pit which were then used to create the design pit.

Mining recovery and dilution were assumed to be 97.7% and 0.3%, respectively (Tetra Tech, 2013). Process recoveries used in pit optimizations are variable based on metal grades as shown in Table 16-3, and are discussed in detail in Section 13.0 of this report.

16.4 Design Parameters

16.4.1 NSR Calculation Parameters

Table 16-2 below outlines the key parameters used in the NSR calculation.

Table 16-2: Pit Optimization Parameters

Metal Prices		
Copper (USD$/lb)	3.15	USD$/lb
Molybdenum (USD$/lb)	10.00	USD$/lb
Gold (USD$/oz)	1,300.00	USD$/oz
Silver (USD$/oz)	20.00	USD$/oz
Copper Concentrate Terms		
Cu Con Grade	28.00	%
Payable Copper	96.50	%
Copper Refining Charge	0.09	USD$/lb
Copper Con TC	90.00	USD$/dmt
Gold Refining Charge	6.00	USD$/oz
Payable Metal		
Au Payable	96.50	%
Copper Concentrate Terms		
Silver Refining Charge	0.40	USD$/oz
Payable Metal		
Ag <=30g	0.00	%
Ag > 30g	90.00	%
Molybdenum Concentrate Terms		
Mo Con Grade	50	%
Payable Mo	99	%
Mo Refining Charge	1.5	USD$/lb
Mo Con TC	0	USD$/dmt
Transport Cost Copper Concentrate		
Freight (inland + Ocean)	76.67	USD$/wmt
Marketing & Others	0.50	USD$/wmt

table continues…

Insurance	0.15	%
Moisture	9.0	%
Total Freight	100.00	USD$/dmt
Transport Cost Molybdenum Concentrate		
Land Freight	55.89	USD$/wmt
Marketing & Others	0.5	USD$/wmt
Insurance	0.15	%
Moisture	5.0	%
Total Freight	200.00	USD$/dmt
Costs		
Mining Cost	2.37	USD$/t
Processing Cost	5.64	USD$/t

Table 16-3: Process Recoveries Used in Pit Optimization

Copper Concentrate		
Concentrate Grade	Copper Grade = 28% Cu	
Metal Recovery		
Copper Head	> 1.25% Cu	Copper Recovery = 96.5%
	0.15% to 1.25% Cu	Copper Recovery = 7.3552 x ln (Copper Head, %) + 92.872
	0.10% to 0.15% Cu	Copper Recovery = 70%
	0.05% to 0.10% Cu	Copper Recovery = 45%
	0.02% to 0.05% Cu	Copper Recovery = 10%
	< 0.02% Cu	Copper Recovery = 0%
Gold Head	> 5.0 g/t Au	Gold Recovery = 95%
	1.5 g/t to 5.0 g/t Au	Gold Recovery = 93%
	0.1 g/t to 1.5 g/t Au	Gold Recovery = 9.812 x ln (Gold Head, g/t) + 88.904
	0.05 g/t to 0.1 g/t Au	Gold Recovery = 50%
	0.02 g/t to 0.05 g/t Au	Gold Recovery = 10%
	< 0.02 g/t Au	Gold Recovery = 0%
Silver Head	> 8.0 g/t Ag	Silver Recovery = 92%
	1.0 g/t to 8.0 g/t Ag	Silver Recovery = 23.839 x ln (Silver Head, g/t) + 35.330
	0.5 g/t to 1.0 g/t Ag	Silver Recovery = 20%
	< 0.5 g/t Ag	Silver Recovery = 0%
Molybdenum Concentrate		
Concentrate Grade	Molybdenum Grade = 50% Mo	
Molybdenum Recovery		
Molybdenum Head	> 0.10% Mo	Recovery = 85%
	0.01% to 0.10% Mo	Recovery = 12.042 x ln (Molybdenum Head, %) + 107.12
	0.005% to 0.01% Mo	Recovery = 45%
	0.003% to 0.005% Mo	Recovery = 20%
	< 0.003% Mo	Recovery = 0%

16.4.2 Geotechnical Parameters

The geotechnical design parameters for the development of the optimized pit shells remained unchanged from the 2013 FS completed by Copper Fox.

The design criteria are shown in Table 16-4.

Table 16-4: Geotechnical Slope Parameters

From: Bearing	Overall Slope Angle degrees
0	40
115	44
200	44
250	44
320	40

16.5 Mine Plan

The 2021 mine plan for Schaft Creek is based on a conventional open pit truck and shovel operation. The mining fleet was based upon a combination of electric rope shovels, wheel loaders, and a fleet of haul trucks. The main mining fleet will be supported by ancillary equipment that will perform various activities, including pit maintenance, geotechnical support, and dewatering activities.

The stockpiling strategy designed for the Schaft Creek operation allows the mine to give priority to higher value material for processing and ensure that the required mill feed is maintained throughout the LOM. ROM stockpile material will be deposited at the stockpile with the primary haul truck fleet.

16.6 Mine Production Plans

16.6.1 Net Smelter Return Calculation

Using the parameters described in Section 16.4.1, the NSR value per tonne of mineralized material was calculated and incorporated into the block model. An NSR value of USD$4.31/t was determined to be the optimal cut-off. The NSR value was the key input for the optimization and scheduling process and was calculated using the metal recoveries to concentrate.

16.6.2 Cut-off Grade Policy

Once the optimal pit shell was established, the processing and G&A costs formed the basis for the cut-off grade determination. For subsequent mine scheduling exercises, a variable cut-off grade was applied on a year-by-year basis, based on the available Mineral Resource for the period. Using a variable cut-off strategy, including taking advantage of stock piling the lower grade material during the first five years (see Table 16-8 for details), results in an average mine head grade of 0.30% copper over the first five years of production.

16.6.3 Variable Mill Throughput

The base mill throughput was set at 133,000 t per day, for production planning a variable throughput dependent on the lithology of the rock type delivered to the mill was applied to determined the overall mill throughput in each period. Table-16-5 below outlines the throughput rates for each lithology as provided in Table 17-2, Section 17.0 of this report.

Table 16-5: Mill Throughput by Lithology

Lithology	Throughput tpd	LOM Total Mill Feed Mt
Breccia	>150,000	173.4
Intrusive	>150,000	103.0
Porphyry	149,000	46.2
Volcanic	133,000	707.5

16.6.4 Mine Haulage Requirements

Haulage routes designed for the LOM operation include internal pit hauling, hauls ex-pit to the mill, the east RSF, the west RSF and both the north and south tailings embankments. The estimated haulage hours were the basis for sizing the equipment fleet and the mine personnel requirements.

16.6.5 Rock Storage Facilities

The 2013 FS utilized three RSFs: east, west, and south. With the reduction in LOM waste material in the 2021 mine plan, there is sufficient capacity in the east and west RSFs to handle the volume of waste. The south RSF was removed from the mine plan as haul distances are greater than to the other two facilities. The south RSF represents a suitable location for any future expansion beyond the current 21 years LOM. The total capacity for the RSFs is 571.1 million m^3, which exceeds the required capacity of ~500 million m^3, see Table 16-6 for the individual capacities. The site layout for the pit and RSFs is shown in Figure 16-1.

Table 16-6: RSF Capacities

RSF	Capacity M^3 millions	Capacity Mt
East RSF	334.1	634.8
West RSF	237.1	450.3
Total	571.1	1085.1
South RSF	159.1	302.3

The estimated requirements of waste rock for tailings embankment construction and the placement schedule is shown in Table 16-7.

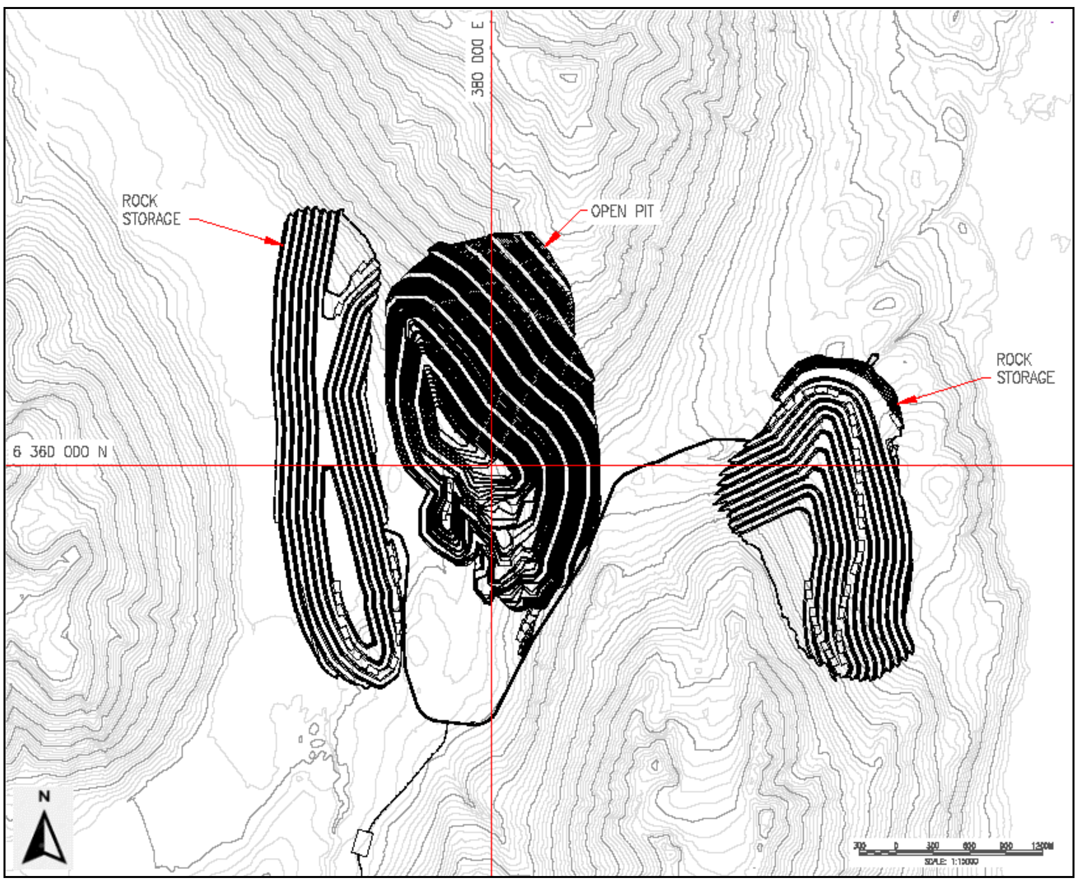
Table 16-7: Tailing Embankment Rock Placement Schedule

Tailings Embankment Waste Rock Placement			Total Tonnes Placed
Year	North Embankment m^3	South Embankment m^3	
-2	4,700,000	--	8,930,000
2	5,000,000	5,020,000	19,038,000
6	4,300,000	3,570,000	14,953,000
10	5,400,000	4,460,000	18,734,000
15	8,935,000	6,360,000	29,060,500
Total	28,335,000	19,410,000	90,715,500

Note: calculated loose density of 1.9 t/m^3, from an estimated 30% swell factor and in-situ density of 2.7 t/m^3

The Annual Mine Production Plan is shown in Table 16-8.

Figure 16-1: General Layout of Open Pit Area



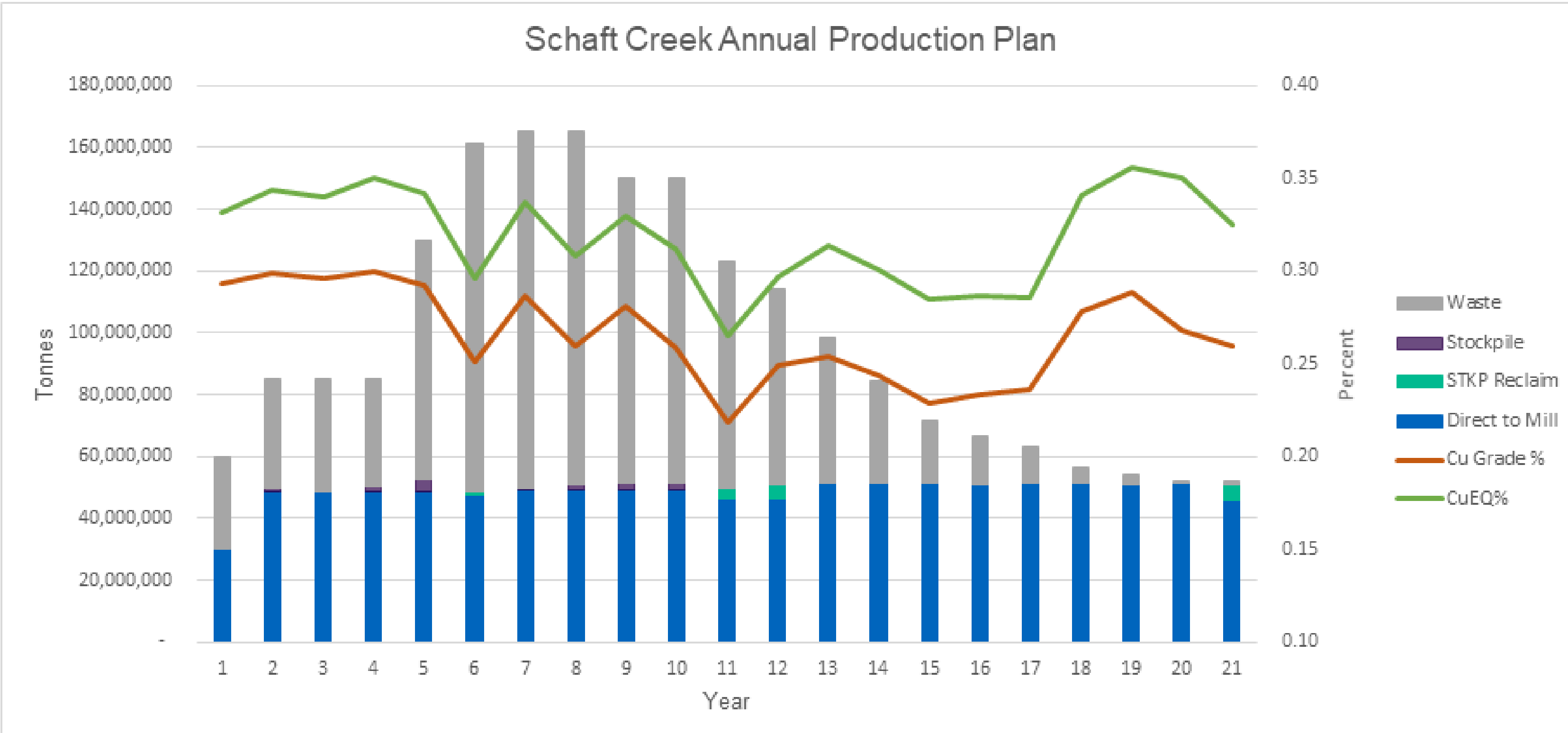
Source: Tetra Tech (2021)

Table 16-8: Annual Production Schedule

		Total	Period Name 1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21
Summary	Direct to Mill Tonnes	**1,016,082,744**	30,094,932	48,556,878	48,442,044	48,578,730	48,704,940	47,464,440	49,096,002	48,917,070	48,938,838	48,912,192	46,590,660	46,453,974	51,224,226	51,190,578	51,486,600	51,066,960	51,357,480	51,405,900	50,841,000	51,228,360	45,530,940
	Stockpile Tonnes	**14,123,898**	-	976,608	-	1,694,700	3,737,928	-	581,040	1,839,960	2,550,120	2,743,542	-	-	-	-	-	-	-	-	-	-	-
	Manual Reclaim (tonnes)	**14,123,898**	-	-	144,228	-	-	1,122,900	-	-	-	-	3,324,840	4,335,108	-	-	-	-	-	-	-	-	5,196,822
	Waste Tonnes	**1,029,292,068**	29,919,036	35,473,944	36,566,232	34,739,868	77,558,916	112,550,400	115,338,864	114,251,568	98,518,692	98,353,896	73,414,416	63,550,422	47,245,194	33,577,500	19,916,760	15,688,080	11,701,500	4,922,700	3,647,640	984,540	1,371,900
	Total Ore Tonnes	**1,030,206,642**	30,094,932	48,556,878	48,586,272	48,578,730	48,704,940	48,587,340	49,096,002	48,917,070	48,938,838	48,912,192	49,915,500	50,789,082	51,224,226	51,190,578	51,486,600	51,066,960	51,357,480	51,405,900	50,841,000	51,228,360	50,727,762
	Strip Ratio W/O	**1.00**	**0.99**	**0.72**	**0.75**	**0.69**	**1.48**	**2.37**	**2.32**	**2.25**	**1.91**	**1.90**	**1.58**	**1.37**	**0.92**	**0.66**	**0.39**	**0.31**	**0.23**	**0.10**	**0.07**	**0.02**	**0.03**
	Total Tonnes (Inc Reclaims)	**2,073,622,608**	60,013,968	85,007,430	85,152,504	85,013,298	130,001,784	161,137,740	165,015,906	165,008,598	150,007,650	150,009,630	123,329,916	114,339,504	98,469,420	84,768,078	71,403,360	66,755,040	63,058,980	56,328,600	54,488,640	52,164,480	52,148,082
	Total Tonnes (Exc Reclaims)	**2,059,498,710**	60,013,968	85,007,430	85,008,276	85,013,298	130,001,784	160,014,840	165,015,906	165,008,598	150,007,650	150,009,630	120,005,076	110,004,396	98,469,420	84,768,078	71,403,360	66,755,040	63,058,980	56,328,600	54,488,640	52,164,480	46,951,260
	mcost (WAvg)	**2.02**	1.62	1.53	1.68	2.09	1.73	1.77	1.76	1.85	1.85	1.89	1.84	2.17	2.00	2.14	2.37	2.52	2.73	2.87	2.91	2.93	2.79
Total Mill Feed	**Total Mill Feed**																						
	Tonnes	**1,030,206,642**	30,094,932	48,556,878	48,586,272	48,578,730	48,704,940	48,587,340	49,096,002	48,917,070	48,938,838	48,912,192	49,915,500	50,789,082	51,224,226	51,190,578	51,486,600	51,066,960	51,357,480	51,405,900	50,841,000	51,228,360	50,727,762
	Tonnes / day		82,451.87	133,032.54	133,113.07	133,092.41	133,438.19	133,116.00	134,509.59	134,019.37	134,079.01	134,006.01	136,754.79	139,148.17	140,340.35	140,248.16	141,059.18	139,909.48	140,705.42	140,838.08	139,290.41	140,351.67	138,980.17
	aggpt (WAvg)	**1.23**	1.00	1.25	1.32	1.29	1.17	1.09	1.22	1.17	1.30	1.12	1.18	1.33	1.31	1.28	1.19	1.19	1.15	1.29	1.38	1.23	1.23
	augpt (WAvg)	**0.16**	0.15	0.23	0.24	0.21	0.17	0.17	0.16	0.19	0.19	0.16	0.15	0.15	0.15	0.14	0.11	0.13	0.12	0.12	0.14	0.13	0.13
	cupct (WAvg)	**0.26**	0.29	0.30	0.30	0.30	0.29	0.25	0.29	0.26	0.28	0.26	0.22	0.25	0.25	0.24	0.23	0.23	0.24	0.28	0.29	0.27	0.26
	mopct (WAvg)	**0.02**	0.01	0.01	0.01	0.02	0.02	0.01	0.02	0.01	0.02	0.02	0.01	0.01	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.03	0.02
Direct To Mill	**Direct to Mill**																						
	Tonnes	**1,016,082,744**	30,094,932	48,556,878	48,442,044	48,578,730	48,704,940	47,464,440	49,096,002	48,917,070	48,938,838	48,912,192	46,590,660	46,453,974	51,224,226	51,190,578	51,486,600	51,066,960	51,357,480	51,405,900	50,841,000	51,228,360	45,530,940
	aggpt (WAvg)	**1.23**	1.00	1.25	1.32	1.29	1.17	1.09	1.22	1.17	1.30	1.12	1.15	1.36	1.31	1.28	1.19	1.19	1.15	1.29	1.38	1.23	1.23
	augpt (WAvg)	**0.16**	0.15	0.23	0.24	0.21	0.17	0.17	0.16	0.19	0.19	0.16	0.14	0.15	0.15	0.14	0.11	0.13	0.12	0.12	0.14	0.13	0.13
	cupct (WAvg)	**0.26**	0.29	0.30	0.30	0.30	0.29	0.25	0.29	0.26	0.28	0.26	0.21	0.25	0.25	0.24	0.23	0.23	0.24	0.28	0.29	0.27	0.26
	mopct (WAvg)	**0.02**	0.01	0.01	0.01	0.02	0.02	0.01	0.02	0.01	0.02	0.02	0.01	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.03	0.02
To Stockpile	**Stockpile Tonnes**																						
	Tonnes	**14,123,898**	-	976,608	-	1,694,700	3,737,928	-	581,040	1,839,960	2,550,120	2,743,542	-	-	-	-	-	-	-	-	-	-	-
	aggpt (WAvg)	**1.26**	-	0.94	-	1.65	1.36	-	1.12	1.09	0.82	1.54	-	-	-	-	-	-	-	-	-	-	-
	augpt (WAvg)	**0.16**	-	0.14	-	0.23	0.16	-	0.21	0.19	0.10	0.14	-	-	-	-	-	-	-	-	-	-	-
	cupct (WAvg)	**0.26**	-	0.24	-	0.36	0.29	-	0.21	0.22	0.23	0.24	-	-	-	-	-	-	-	-	-	-	-
	mopct (WAvg)	**0.01**	-	0.01	-	0.02	0.01	-	0.01	0.01	0.01	0.01	-	-	-	-	-	-	-	-	-	-	-
Reclaim to Mill	**Reclaim Tonnes**																						
	Tonnes	**14,123,898**	-	-	144,228	-	-	1,122,900	-	-	-	-	3,324,840	4,335,108	-	-	-	-	-	-	-	-	5,196,822
	aggpt (WAvg)	**1.26**	-	-	0.90	-	-	1.13	-	-	-	-	1.73	1.00	-	-	-	-	-	-	-	-	1.20
	augpt (WAvg)	**0.16**	-	-	0.12	-	-	0.16	-	-	-	-	0.24	0.14	-	-	-	-	-	-	-	-	0.12
	cupct (WAvg)	**0.26**	-	-	0.24	-	-	0.26	-	-	-	-	0.36	0.22	-	-	-	-	-	-	-	-	0.24
	mopct (WAvg)	**0.01**	-	-	0.01	-	-	0.01	-	-	-	-	0.02	0.01	-	-	-	-	-	-	-	-	0.01

The Schaft Creek deposit contains significant concentrations of gold, molybdenum and silver. These by-product credits contribute significantly to the LOM metal production. The annual mine productions expected average copper and copper equivalent grades are shown in Figure 16-2.

Figure 16-2: Annual Production Plan



16.7 Mining Equipment

The main criteria for equipment selection is the ability of the production fleet to meet the material movement requirements to achieve the mine plan. Additionally the loading and hauling equipment need to be a suitable match in terms the ability to load the haul trucks and for the number of passes per load.

16.7.1 Loading Equipment

The primary loading units will be four electric rope shovels with 45 m^3 buckets. These units were selected based on their proven productivity and ability to efficiently mine and load the selected haul trucks.

For secondary loading and stockpile reclaim, the primary equipment will be wheel loaders with 22 m^3 buckets. These units will also be used for waste mining throughout the mine life. These units have been selected based on their ability to dig 15 m benches as well as on their mobility.

16.7.2 Hauling Equipment

A fleet of 38 haul trucks (360 t) will be required during mining operations. Komatsu 980E AC diesel electric haul trucks or similar will be selected. These trucks are a match for the selected loading units.

16.7.3 Ancillary and Support Equipment

Various pieces of equipment will be required during LOM operations to support production activities. Support activities include haul road maintenance, mine dewatering, transporting operating supplies and transporting equipment. Table 16-9 below shows the major equipment list. Table 16-10 shows the major mine equipment purchase and replacement schedule during LOM.

Table 16-9: LOM Major Equipment Purchases

	Total
Drill - Electric - 311mm Operating	6
FEL Blast Hole Stemmer - 111kW -	2
Emulsion Loading Truck	2
Rope Shovel - Electric Operating	4
FEL Wheel Loader - 22m3 Operating	4
Haul Truck Operating	38
Dozer - 600Hp	2
Wheel Dozer	1
Dozer - 600Hp	2
Dozer - 350Hp	2
Fuel / Lube Truck - 4000gal -	2
Water Truck - 20 000gal -	1
Motor Grader	2
Cable Reeler	1
Excavator PC 390	2
Excavator PC 240	2
373 kW FEL - Tire Manipulator	1

16.7.4 Equipment Schedule

Table 16-10: Equipment Purchasing and Replacement Schedule

	Year1	Year 2	Year3	Year4	Year5	Year6	Year7	Year8	Year9	Year10	Year11	Year12	Year13	Year14	Year15	Year16	Year17	Year18	Year19	Year20	Year21
Drill - Electric - 311mm Fleet	2	1	-	-	2	1	-	1	-	1	-	-	-	-	1	-	1	-	-	-	-
FEL Blast Hole Stemmer - 111kW -	1	-	-	-	-	1	-	-	-	-	-	-	-	-	1	-	-	-	-	-	-
Emulsion Loading Truck	1	-	-	-	-	1	-	-	-	1	-	-	-	-	-	-	-	1	-	-	-
Rope Shovel - Electric Operating	2	-	-	-	1	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
FEL Wheel Loader - 22m3 Fleet	2	-	-	-	1	1	-	-	-	-	-	-	-	-	-	1	-	-	-	-	-
Haul Truck	11	3	2	7	3	8	1	3	-	2	-	3	3	-	-	1	1	-	-	-	-
Dozer - 600Hp	1	-	-	1	-	-	-	1	-	-	-	1	-	-	-	1	-	-	-	-	-
Wheel Dozer	1	-	-	-	-	1	-	-	-	-	-	1	-	-	-	-	-	1	-	-	-
Dozer - 600Hp	2	-	-	-	1	1	-	-	-	1	1	-	-	-	1	1	-	-	-	1	-
Dozer - 350Hp	2	-	-	-	-	1	1	-	-	-	-	1	1	-	-	1	1	-	-	-	-
Fuel / Lube Truck - 4000gal -	1	-	1	-	-	-	-	-	-	1	-	1	-	-	-	-	-	-	-	-	-
Water Truck - 20 000gal -	1	-	-	-	-	-	-	-	-	-	1	-	-	-	-	-	-	-	-	-	-
Motor Grader	1	1	-	-	-	-	1	1	-	-	-	-	1	1	-	-	-	-	1	-	-
Cable Reeler	1	-	-	-	-	-	-	-	-	-	-	1	-	-	-	-	-	-	-	-	-
Excavator PC 390	1	1	-	-	-	-	-	1	-	1	-	-	-	-	-	1	-	-	-	-	-
Excavator PC 240	1	1	-	-	-	-	-	1	-	1	-	-	-	-	-	1	-	-	-	-	-
373 kW FEL - Tire Manipulator	1	-	-	-	-	-	-	-	-	-	1	-	-	-	-	-	-	-	-	-	-
Tractor/Trailer - 170t	1	-	-	-	-	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-
Crew Cab Pickup -	10	-	2	3	-	-	-	-	2	2	2	2	-	-	-	-	-	3	-	-	-
Maintenance Truck - 1t -	1	-	-	-	-	-	-	-	-	-	-	1	-	-	-	-	-	-	-	-	-
Mine Rescue Fire Truck	1	-	-	-	-	-	-	-	-	-	-	1	-	-	-	-	-	-	-	-	-
Mine Rescue Ambulance	1	-	-	-	-	-	-	-	-	-	-	1	-	-	-	-	-	-	-	-	-
Gravel Truck	1	-	-	-	-	-	-	-	-	-	-	1	-	-	-	-	-	-	-	-	-
FEL Utility loader	1	-	-	-	-	-	-	-	-	-	-	1	-	-	-	-	-	-	-	-	-
Welding Truck	1	-	-	-	-	-	-	-	-	-	-	1	-	-	-	-	-	-	-	-	-
Powerline Truck	1	-	-	-	-	-	-	-	-	-	-	1	-	-	-	-	-	-	-	-	-
Forklift 10t	1	-	-	-	-	-	-	-	-	-	-	1	-	-	-	-	-	-	-	-	-
Forklift 30t	1	-	-	-	-	-	-	-	-	-	-	1	-	-	-	-	-	-	-	-	-
Screening and Crushing Plant	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Crew Bus	2	-	-	1	-	-	-	-	-	1	-	-	1	-	-	-	-	-	-	-	-
Pit Dewatering Pump 160Hp	2	1	1	1	-	-	-	-	1	-	1	-	-	-	-	-	-	1	-	-	-
Light Plant	3	2	1	1	-	-	-	-	-	2	-	1	-	1	1	-	-	-	-	-	-
10 tonne smooth Drum Roller	2	-	-	-	-	-	-	-	-	-	1	1	-	-	-	-	-	-	-	-	-

16.7.5 Mining Personnel

Table 16-11 shows the expected numbers of equipment operators, maintenance personnel, engineering, support, and management personnel.

Table 16-11: Mining Personnel Requirement

	Year 5	Year 10	Year 15	LOM
Shovel Operator	9	10	5	4
Loader Operator	9	10	5	4
Support Equipment	57	67	43	35
Haul Truck Operator	84	115	73	61
Drill Operator	14	16	8	5
Blast Crew	3	3	3	1
Cable Crew	5	5	5	5
Maintenance	68	83	51	40
Mine Services	5	5	5	5
Technical Services	20	20	20	17
Operations Supervision	14	14	14	8
Maintenance Supervision	14	14	14	9
Total	**302**	**362**	**246**	**194**

These numbers are based on 4 shifts, 365 days per year on a 12-hour per day roster.

17.0 RECOVERY METHODS

17.1 Introduction

The Schaft Creek deposit is described as a calc-alkalic, copper-molybdenum-gold-silver, porphyry deposit, with a low-sulphidation state, and overlapping mineralized zones that contain significant copper, molybdenum, and gold mineralization.

The deposit will be mined using conventional truck-shovel, open-pit equipment. The mineralization will be processed in a conventional flotation plant, producing a copper concentrate containing gold and silver, and a molybdenum concentrate.

This section outlines the metallurgical process, including major design criteria and process description.

17.2 Summary

The proposed processing plant is designed to process the Schaft Creek mineralization at a nominal throughput of 133,000 t/d to produce market-grade copper and molybdenum concentrates.

The LOM average mill feed grades are estimated to be 0.26% Cu, 0.16 g/t Au, 1.23 g/t Ag, and 0.017% Mo. The estimated metal recoveries are 83.1% for copper, 71.0% for gold, and 40.3% for silver in copper concentrate and 60.1% of molybdenum in molybdenum concentrate. The LOM average annual production is estimated to be approximately 385,000 t/a of copper concentrate, which contains 28% Cu, 14.1 g/t Au, 63.1 g/t Ag, and 9,780 t/a of molybdenum concentrate at 50% Mo.

A conventional flotation process is proposed for the Project. The processing plant will consist of the following:

- Primary crushing at the mine site
- A crushed mill feed stockpile
- A main processing plant, including
  - Two primary grinding circuits, consisting of two SABC circuits
  - Two copper-molybdenum bulk flotation circuits
  - One copper and molybdenum separation circuit, including molybdenum concentrate leaching to reduce copper and lead contents in the final molybdenum concentrate
  - Concentrate dewatering
  - Tailings disposal

Two gyratory crushers, operating as the primary crushing units, will reduce the mill feed particle size to approximately 80% passing 120 mm or finer. The crushed mill feed will be conveyed to a stockpile with a live capacity of 120,000 t. The mill feed will then be reclaimed in two parallel lines to two SABC grinding circuits to further reduce particle size down to 80% passing approximately 150 µm.

There will be two trains of copper-molybdenum rougher/scavenger flotation. The products from the primary grinding circuits will feed to the rougher/scavenger flotation circuits, which will produce a high-grade rougher concentrate and a lower-grade rougher/scavenger concentrate. The two concentrates will be separately reground, then upgraded in three stages of cleaner flotation to produce a copper-molybdenum bulk flotation concentrate. The bulk concentrate will be further treated by flotation to produce a molybdenum concentrate and a copper concentrate. The copper concentrate will contain approximately 28% Cu. The molybdenum concentrate will contain approximately 50% Mo after the flotation concentrate is leached using the chloride leaching procedure to reduce copper and lead contents.

The final flotation concentrates will be thickened and then pressure-filtered to a moisture content of approximately 9%, while the molybdenum concentrate will be further dewatered by drying to a moisture content of approximately 4% to 5%. The copper concentrate will be stockpiled and then bulk trucked via Highway 37 to the Port of Stewart for storage and loading for export of the concentrate to foreign markets. The dried molybdenum concentrate will be bagged prior to being trucked to the Fairview Terminals in Prince Rupert for shipment to various international destinations.

The final rougher/scavenger tailings containing mostly non-sulphide gangue minerals will be stored in the TSF. The simplified processing flowsheet is shown in Figure 17-1.

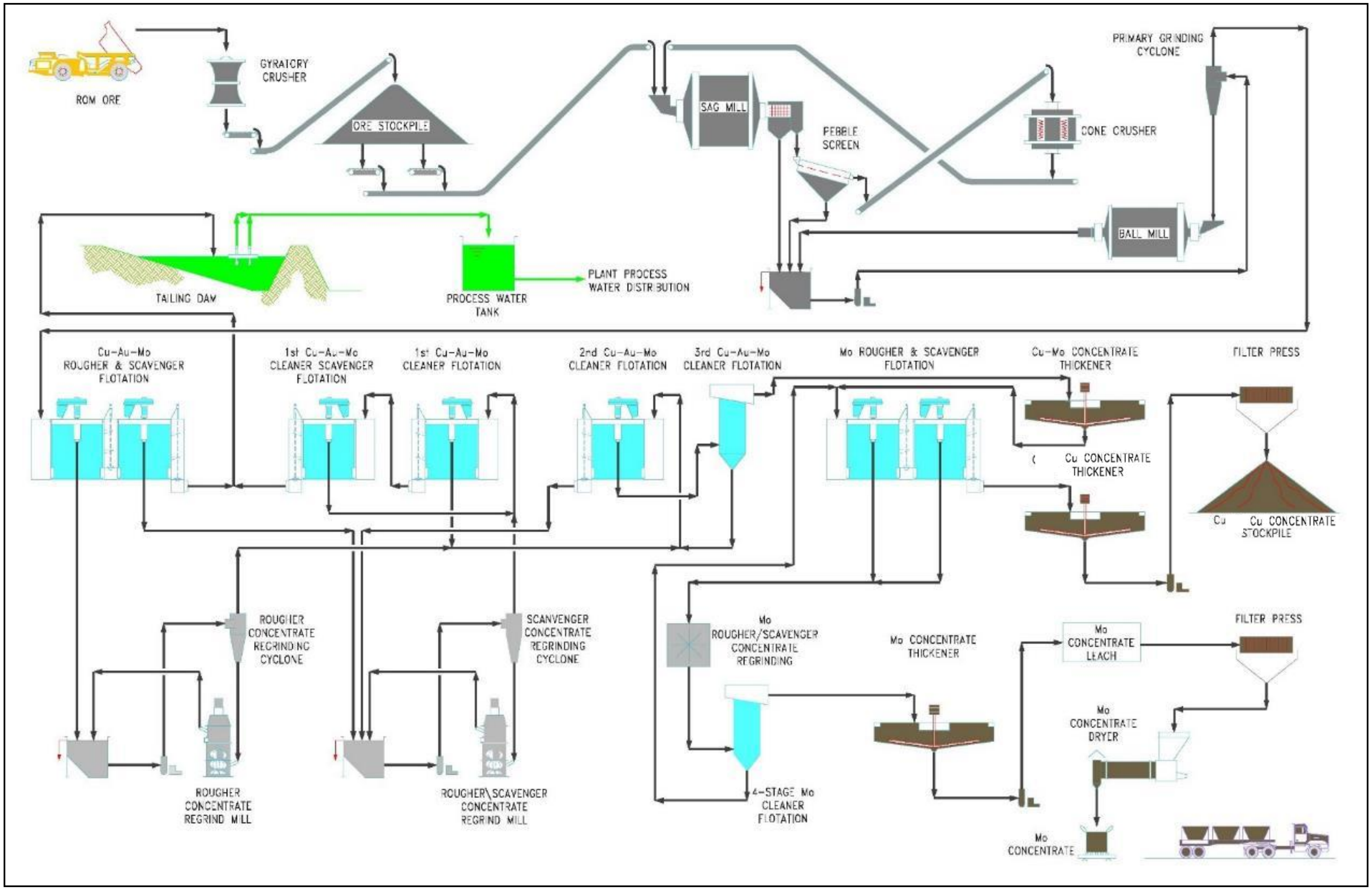


Figure 17-1: Simplified Processing Flowsheet

Reagents added for bulk flotation will be SEX, FO, MIBC, and lime. Sodium hydrosulphide (NaHS) will be used to suppress copper minerals in the copper and molybdenum separation circuit.

Chemical analytical and metallurgical laboratories will be provided to support the operation.

17.3 Major Plant Design Criteria

The processing plant is designed to process mill feed on a daily rate of 133,000 t/d. The major criteria used in the design are outlined in Table 17-1.

Table 17-1: Major Plant Design Criteria

Criteria	Unit	Value
Operating Days per Year	d	365
Plant Overall Availability	%	92
Crushing Rate	t/h	7,389
Milling and Flotation Process Rate	t/h	6,024
Crushing Product, Particle Size, P_{80}	mm	120 or finer
Primary Grinding Circuit Transfer Size, T_{80}	µm	2,900
SAG Mill Circulating Load	%	25
Ball Mill Circulating Load	%	250
Secondary Grinding Product Size, P_{80}	µm	150
Bond BWi (Volcanic Type)	kWh/t	22.4
JK SimMet Breakage Parameter A x b (Volcanic Type)	-	31.8
Bond Abrasion Index (Average)	g	0.23
Concentrates Regrind		
Rougher Concentrate Regrind Size, P_{80}	µm	35–40
Rougher/Scavenger Concentrate Regrind Size, P_{80}	µm	25–30
Molybdenum Concentrate Regrind Size, P_{80}	µm	18

The size selection of the crushers and grinding mills is based on the mineralization amenability to crushing and grinding, as determined through test programs performed by different laboratories, including the 2015 GeoMet test program. Tests were performed to determine the following parameters: BWi, RWi, CWi, JK SimMet Drop Weight breakage, SMC breakage, and HPGR-related hardness.

The 2015 simulation was based on dual-line circuit, each line equipped with one 40 ft. by 23 ft. (EGL) SAG mill with installed power of 22.5 MW, two 26 ft. by 44.5 ft. (EGL) ball mills with 18 MW per mill, and one MP1000 pebble crusher. The simulation results show that the maximum throughput is approximately 118 kt/d when processing 100% volcanic-type mineralization and approximately 153 kt/d when processing 100% breccia-type mineralization. It appears that the ball mill capacity is the circuit limit. The results may imply that when volcanic type material is the dominant mill feed, the primary grinding circuit may not be able to achieve the designed throughput of 133 kt/d at a mill availability of 92%. To meet the design capacity, the installed ball mill power has been increased to 20 MW per mill and the mill sizing to 27 ft. by 45 ft. (EGL), while the SAG mill remains the same. Table 17-2 shows the estimated nominal grind circuit capacities for different lithological types of the proposed mill feeds. Further circuit arrangement assessments should be conducted in conjunction with the potential mine plan, including further investigations into the effect of the feed/product particle size distributions.

Table 17-2: Projected Grinding Circuit Capacity for Mill Feed Lithological Type

Lithology	Nominal Grinding Circuit Throughput, tpd*
Breccia	>150,000
Intrusive	>150,000
Porphyry	149,000
Volcanic	133,000

* daily nominal throughput at an availability of 92%

Regrind mill sizing is based on the Bond work index equation for ball mills and the estimated tower mill-to-ball mill efficiency factor.

Flotation cell sizing is based on optimum flotation times determined by test work and using scale-up factors from similar operations.

17.4 Processing Plant Description

The processing of this porphyry copper-gold-molybdenum mineralization will include crushing, grinding, and flotation as main operations to obtain marketable copper and molybdenum concentrates.

17.4.1 Crushing

The mill feed will be transported from the proposed open pit mine site to the primary gyratory crushers by haul trucks. A total of two crushing facilities (one stationary and one semi-mobile) are proposed to crush the mill feed at an average rate of 7,389 t/h. Each of the crushers will process 3,694 t/h. The stationary crusher will be located to the south of the pit, while the semi-mobile crusher will be located inside the pit.

The mill feed will be crushed to 80% passing 120 mm or finer by the two crushers depending on its particle size. Each of the crushing stations will be equipped with one rock breaker to break oversize rocks. In addition, dust collectors will be provided at the crushing facilities to control fugitive dust generated during crushing and transporting.

The main equipment and equipment features of the crushing facilities will be as follows:

- A stationary crushing facility, including:
  - One 1.52 m by 2.79 m gyratory crusher
  - One hydraulic rock breaker
  - One 2.13 m wide by 10.0 m long apron feeder
  - One 1.52 m wide by 326 m long belt conveyor
- A semi-mobile crushing facility, including:
  - One 1.52 m by 2.26 m in-pit semi-mobile gyratory crusher
  - One hydraulic rock breaker
  - One 2.14 m wide by 25 m long in-pit apron feeder
  - One 1.83 m wide belt conveyor
  - One 1.52 m wide by 261 m long belt conveyor

The crushed mill feed from the two crushing facilities will be conveyed to a 120,000 t live capacity stockpile. Dust collection systems will be installed to control the spread of the fugitive dust generated during the crushing and the mill feed transport at the crushed mill feed re-handling areas.

17.4.2 Coarse Mill Feed Stockpile

The coarse mill feed stockpile will have a live capacity of 120,000 t. The received material will be reclaimed from the stockpile by eight 1.52 m wide by 7.60 m long apron feeders at a nominal rate of 1,004 t/h per feeder. The stockpile reclaim area will also be equipped with a dust collection system to minimize the spread of dust generated during mill feed transport.

The reclaimed mill feed from the apron feeders will be discharged onto two 1.52 m wide by 670 m long SAG mill feed conveyors.

17.4.3 Grinding and Classification

Two SABC circuits are proposed to grind the coarse mill feed to 80% passing 150 µm. On average, each of the SABC circuits will be capable of processing 3,012 t of the coarse mill feed per hour.

17.4.3.1 Primary Grinding and Classification

There will be two primary grinding circuits; each of the circuits will consist of one SAG mill and one vibrating screen, operating in a closed circuit with pebble (cone) crushers. The major pieces of the primary grinding and crushing equipment will be as follows:

- Two 12.20 m diameter by 7.01 m EGL SAG mills, each with one 22.5 MW gearless motor drive
- Two 3.66 m wide by 7.30 m long vibrating screens

- Three cone crushers, each powered by one 750 kW motor

The coarse mill feed from two conveyors will be separately fed to the two SAG mills. The SAG mills will be equipped with 76 mm pebble ports for the removal of undersize material, including critical-size pebbles. The mill discharge from each SAG mill will be screened by a SAG mill trommel and then a vibrating screen. The trommel and screen undersize will flow by gravity to a sump and be pumped to the secondary grinding circuits. The oversized material from the two screens will be separately conveyed to a common pebble crusher feed surge bin. An automatic ball charge device will add grinding balls into the SAG mills at a controlled rate.

The pebble surge bin will have a capacity of 1,500 t. The pebbles will be reclaimed by three retrievable belt feeders to three cone crushers, which will crush the pebbles to 80% passing 12.5 mm. The crushed pebbles will be directed onto the SAG mill feed conveyors, which will carry the crushed pebbles and coarse mill feed to the SAG mills.

To protect the cone crushers, each pebble surge bin feed conveying train will be equipped with two magnets and one metal detector for removal of any metals.

Lime slurry will be added to the SAG mills to adjust the flotation feed slurry pH.

17.4.3.2 Secondary Grinding and Classification

The undersize material from each primary grinding circuit will be pumped to the cyclone feed pump box in the secondary grinding circuit. The SAG mill product, together with the ball mill discharge, will be pumped to cyclone clusters. The cyclone underflow will flow by gravity to the ball mills, while the cyclone overflow with a solid density of approximately 35% w/w will be sent to downstream bulk flotation circuits.

The secondary grinding circuit will include four ball mills, each in closed circuit with one cyclone cluster. The grinding circuits will further grind the SAG mill products to 80% passing 150 µm. The major equipment in the secondary grinding circuits will include the following main features:

- Four 8.23 m diameter by 13.72 m EGL ball mills, each driven by two 10.0 MW dual-pinion motors
- Four cyclone clusters, each consisting of ten 840 mm diameter cyclones (seven operating, three in standby)

Two separate automatic ball charging systems will be provided to charge grinding media to the ball mills. Collectors (FO and SEX) will be added to each of the ball mill pump boxes.

17.4.4 Flotation

The ground mill feed slurry will be processed by conventional flotation to recover the valuable minerals. The flotation will consist of bulk flotation to produce a bulk copper-molybdenum concentrate and subsequent copper/molybdenum separation flotation to produce a copper concentrate and a molybdenum concentrate.

17.4.4.1 Bulk Flotation

Bulk (copper, gold, silver, and molybdenum) flotation will include rougher, scavenger, and cleaner flotation. Two bulk concentrate regrind circuits are proposed to further liberate the valuable minerals from the gangue material.

17.4.4.2 Bulk Rougher/Scavenger Flotation

Rougher and scavenger flotation will be carried out in 300 m^3 tank flotation cells. Two flotation banks are designed for the flotation, each bank having six 300 m^3 tank cells for rougher flotation and three 300 m^3 tank cells for scavenger flotation. The pulp from the four secondary grinding circuits will be separately fed to the two rougher/scavenger flotation banks.

Two concentrates, one high-grade and one low-grade, will be produced from rougher/scavenger flotation. The high-grade concentrate will be produced from the first flotation cell and the low-grade concentrate from the other rougher/scavenger flotation cells. The scavenger flotation tailings from the flotation banks constitute part of the final tailings and will be sent to the TSF.

The high-grade concentrates from the first cell of the two rougher flotation banks will be sent to one high-grade bulk concentrate regrinding circuit, and the remaining rougher concentrates, together with the scavenger concentrates, will be directed to low-grade concentrate regrinding circuits.

Rougher bulk flotation will be conducted at pH 9 while scavenger flotation will be between pH 8.5 and 9. The proposed reagents for rougher and scavenger flotation will include SEX and FO as collectors and MIBC as frother. Lime will be added as required.

17.4.4.3 Bulk Concentrate Regrinding

Two separate regrinding circuits will be provided for the concentrates produced from rougher and scavenger flotation.

The high-grade rougher concentrates will be reground by one tower mill in closed circuit with hydrocyclones to 80% passing approximately 35 µm to 40 µm. The low-grade rougher and scavenger concentrates will be ground to approximately 80% passing approximately 25 µm to 30 µm in two circuits, each circuit consisting of three tower mills in closed circuit with hydrocyclones.

The high-grade rougher concentrate regrinding circuit will include the following major equipment:

- One 2,237 kW tower mill for the two high-grade rougher concentrates
- One cyclone cluster with six 380 mm hydrocyclones, four in operating and two in standby

The low-grade rougher and scavenger concentrate regrinding circuits will include the following major equipment:

- Six 2,237 kW tower mills, three mills per circuit
- Two hydrocyclone clusters, each with twenty-two 250 mm hydrocyclones, eighteen operating and four in standby

Lime, SEX, and FO will be added to modify physical and chemical characteristics of the mineral surfaces to improve target mineral flotation.

17.4.4.4 Bulk Concentrate Cleaner Flotation

The reground low-grade concentrate will be upgraded by three stages of cleaner flotation: first, cleaner flotation and cleaner scavenger flotation; second, cleaner flotation; and third, cleaner flotation. There are two parallel cleaner flotation trains. Major equipment will consist of the following:

- Ten 100 m^3 first cleaner flotation tank cells, five cells per train
- Four 100 m^3 first cleaner scavenger flotation tank cells, two cells per train
- Six 50 m^3 second cleaner flotation tank cells, three cells per train
- Two 4.3 m diameter by 8.0 m high flotation columns, one per train for the third cleaner flotation

For each of the cleaner flotation trains, the reground low-grade concentrate from the rougher and scavenger concentrate regrinding circuit will flow by gravity to the head of the first cleaner flotation bank. The concentrate generated from the first cleaner flotation will be pumped to the second cleaner flotation cells, while the tailings will be floated in cleaner scavenger flotation cells. The resulting flotation tailings, which constitute part of the final tailings, will report to the bulk rougher/scavenger tailings pump box from where the tailings will be sent to the TSF. The cleaner scavenger concentrate will be recycled to the first cleaner flotation.

The first cleaner concentrate, together with the reground higher-grade concentrate from the high-grade rougher concentrate regrinding circuit, will be subjected to further cleaning in the second cleaning flotation cells. The tailings from the second cleaner flotation will be directed to the rougher and scavenger concentrate regrinding circuit. The second cleaner concentrate will feed the flotation column for further upgrading.

The concentrates from the flotation columns will be sent to the common copper/molybdenum bulk concentrate thickener, while the tailings will be recycled to the preceding second cleaner flotation.

The cleaner flotation will be carried out at a pulp pH between 11.0 and 11.5 and a pulp density lower than 20% solids (w/w). Reagents added are FO and SEX as collectors, MIBC as frother, and lime as pH modifier.

17.4.4.5 Bulk Concentrate Thickening

The copper-molybdenum bulk concentrates from the flotation columns will feed to a 20 m diameter, high-rate thickener at a solid feed rate of approximately 51 t/h. Flocculant will be added to the thickener feed well to assist settling of fine particles. The thickened concentrate (approximately 60% solids) will be pumped to the molybdenum flotation circuit to separate molybdenite from copper minerals. The thickener overflow will be recycled to the primary grinding circuits.

17.4.4.6 Copper-Molybdenum Separation

The thickened bulk copper-molybdenum concentrate will be pumped to the copper and molybdenum separation circuit, which is located in a separate area with its own ventilation system to separate the circuit from the main copper-molybdenum bulk flotation. The circuit will include molybdenum rougher and scavenger flotation, molybdenum rougher/scavenger concentrate regrinding, and molybdenum cleaner flotation.

17.4.4.7 Molybdenum Rougher/Scavenger Flotation

The thickened bulk concentrate will feed to the conditioning tank and then overflow to the head of the molybdenum rougher/scavenger flotation cell bank. The flotation will include the following:

- One 3.0 m diameter by 2.5 m high conditioning tank
- Four 30 m^3 molybdenum rougher flotation tank cells
- Two 30 m^3 molybdenum scavenger flotation tank cells

Molybdenum rougher and scavenger flotation will be carried out at a pH of 11 and a slurry density of approximately 28% solids. Sodium hydrosulphide will be added to suppress copper minerals, and FO will be used as a molybdenite collector. To reduce the potential oxidation and consumption of sodium hydrosulphide, nitrogen gas instead of air will be introduced into the flotation cells as aeration media.

Tailings from the scavenger flotation will constitute the copper concentrate. This copper concentrate will be fed to a copper concentrate dewatering circuit.

The concentrate from molybdenum rougher/scavenger cells will be directed to the molybdenum rougher/scavenger concentrate regrinding circuit.

17.4.4.8 Molybdenum Rougher Concentrate Regrinding

The molybdenum rougher/scavenger concentrate will be further reground in an opened regrind circuit consisting of the following:

- One 25 kW stirred mill
- One 50 mm diameter cyclone

The molybdenum rougher/scavenger concentrate will be pumped to the cyclone to separate the fines from the feed. The cyclone underflow will flow by gravity to the stirred mill and be further reground to 80% passing 18 µm. The reground concentrate, together with the cyclone overflow, will be sent to the first molybdenum cleaner flotation column.

17.4.4.9 Molybdenum Concentrate Cleaner Flotation

The four stages of cleaner flotation by flotation columns are designed to further upgrade the molybdenum rougher/scavenger concentrate to approximately 50% molybdenum.

The reground concentrate will be pumped to the first cleaner flotation column. The tailings from the first cleaner column will be recycled to the copper/molybdenum concentrate conditioning tank at the beginning of the molybdenum rougher flotation circuit. The resulting concentrate from the first column will be further cleaned by three successive stages of cleaner flotation in flotation columns. The cleaner tailings generated from each flotation column will be recirculated to the preceding cleaner flotation column.

The slurry pH at these cleaner flotation stages will be approximately 11. The reagents added into the molybdenum rougher/scavenger flotation, including nitrogen gas, will be used in the cleaner flotation as well.

The following flotation cells will be used for the cleaner flotation:

- One 3.5 m diameter by 8.5 m high flotation column for the first cleaner flotation
- One 2.5 m diameter by 8.5 m high flotation column for the second cleaner flotation
- One 1.8 m diameter by 6.0 m high flotation column for the third cleaner flotation
- One 1.5 m diameter by 6.0 m high flotation column for the fourth cleaner flotation

The cleaner concentrate from the fourth cleaner flotation column will be the final molybdenum concentrate and will be sent to the molybdenum leach circuit or to the molybdenum concentrate dewatering circuit depending on impurity levels in the flotation concentrate.

17.4.5 Molybdenum Concentrate Leach

A leach circuit using the chloride leaching procedure is proposed for removing impurities, such as copper and lead, from the final molybdenum flotation concentrate when copper and lead contents in the concentrate are higher than the levels set up by the smelters where the concentrate will be treated. The molybdenum concentrate will be leached with ferric chloride at an elevated temperature in a lined vessel. The leach residue will be washed and dewatered prior to shipping to the molybdenum smelters.

17.4.6 Concentrates Dewatering

Copper concentrate and molybdenum concentrate from the copper and molybdenum separation flotation circuit or molybdenum concentrate leach circuit will be dewatered to reach moisture levels established by market standards.

17.4.6.1 Copper Concentrate Dewatering

The copper concentrate coming from the molybdenum flotation circuit will be subjected to two stages of dewatering, thickening, and pressure filtration. The following major equipment will be used for the dewatering:

- One 18.3 m diameter high-rate thickener
- One 8.0 m diameter by 8.5 m high concentrate stock tank
- Two plate-and-frame pressure filters

The flotation concentrate will be pumped to the copper concentrate thickener. Flocculant will be added to assist the settling of fine copper minerals. The concentrate slurry will be thickened to 60% solids and stored in a stock tank, then pumped to the filters at a controlled rate. The filtered concentrate, containing approximately 9% moisture, will discharge directly by gravity to the concentrate stockpile (load out stockpile) from where it will be transported by trucks to the off-site concentrate storage facility prior to being shipped to the overseas copper smelters.

The filtrate from the filters will be sent to the copper concentrate thickener feed well. The thickener overflow will be used in the molybdenum flotation circuit as process water.

17.4.6.2 Molybdenum Concentrate Dewatering

The final molybdenum concentrate coming from the molybdenum flotation circuit or the leached molybdenum concentrate will be treated by three stages of dewatering, including thickening, filtration, and drying. The key dewatering equipment will include the following:

- One 4.6 m diameter high-rate thickener
- One pressure filter
- One 533 kW dryer

The molybdenum concentrate will be pumped to the thickener feed well where flocculant will be added. The thickener underflow will be filtered and then sent to the molybdenum concentrate dryer. The dewatered concentrate containing 4% to 5% moisture will be loaded into the molybdenum concentrate storage bin prior to bagging. The bagged molybdenum concentrate will be shipped to the molybdenum smelters.

17.4.7 Tailings Disposal

Two flotation tailings streams will be produced from the processing plant. Each of the flotation tailings streams consists of the bulk scavenger flotation tailings and the first cleaner scavenger tailings. The tailings will be sent to the TSF via two pipelines. In the initial operation, the tailings will flow by gravity to the TSF. With an increase in the dam level, two pumping systems will be installed to pump the tailings to the TSF.

The supernatant from the TSF will be reclaimed and used as process water. Anti-scalant will be added to the reclaimed water to protect against potential scaling.

Tailings storage is further detailed in Section 18.0.

17.4.8 Reagent Handling and Storage

To ensure containment in the event of an accidental spill, the reagent preparation and storage facility will be located within a containment area designed to accommodate 110% of the content of the largest tank. The storage tanks will be equipped with level indicators and instrumentation to ensure that spills do not occur during normal operation. Appropriate ventilation, fire and safety protection, and material safety data sheet (MSDS) 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.

17.4.8.1 Collectors

FO will be added in undiluted form to the process circuits via individual metering pumps. This collector will be delivered in bulk containers.

SEX will be shipped to the mine site in solid form in bags, diluted with water to 15% solution strength in a mixing tank, and stored in a 2.0 m diameter by 2.5 m high holding tank before being added to the various addition points by metering pumps.

17.4.8.2 Frother

MIBC will be shipped as liquid in bulk tankers, stored in a holding tank, and pumped in undiluted form to the various addition points.

17.4.8.3 Flocculant

Solid flocculant will be used in this project. It will be shipped in 25 kg bags, prepared in a wetting and mixing system, diluted to 0.5% strength, and stored in a holding tank. The solution flocculant will then be further diluted and fed to the thickener feed wells by metering pumps.

17.4.8.4 Lime

Pebble lime will be delivered by bulk tanker trucks and stored in a dedicated silo with a capacity of at least 300 t. It will be retrieved from the silo by a screw conveyor and slaked in a tower mill. The slaked lime slurry at 15% solids will be stored in a 5.5 m diameter by 5.5 m high agitated tank and distributed throughout the processing plant via a pressurized lime loop.

17.4.8.5 Sodium Hydrosulphide

The solid sodium hydrosulphide will be shipped to the mine site in drums or bags, diluted with water to 15% solution strength in a mixing tank, and stored in a 1.5 m diameter by 2.5 m high holding tank before being added to the various addition points by metering pumps.

17.4.8.6 Nitrogen Gas

Nitrogen gas generators will provide nitrogen gas at the site for the copper and molybdenum flotation.

17.4.8.7 Other Reagents

Anti-scale chemicals, as required, will be added to minimize scale build-up in the reclaim or recycle water lines. This reagent will be delivered in liquid form and metered directly into the intake of the reclaim water pumps.

The reagents that will be used for molybdenum concentrate leaching will be prepared as required. The solid reagents, ferric chloride, copper chloride, and calcium chloride will be shipped to the mine site in drums or bags, separately diluted with water to 15% solution strength in a mixing tank, and stored in a holding tank before being added to the leaching circuit by metering pumps. The liquid reagent, sulphuric acid, will be shipped as liquid in bulk tankers, stored in a holding tank, and pumped in undiluted form to the leach circuit.

New reagents will occasionally be tested to determine their effect on metal recovery and concentrate grading. These reagents will be handled in accordance with MSDS requirements, and any unused test reagents will be returned to the suppliers for disposal. A facility for mixing and dosing these reagents will be provided.

17.4.9 Assay and Metallurgical Laboratory

The assay laboratory will be equipped with necessary analytical instruments to provide all routine assays for the mine, the processing plant, and the environmental and geological departments. The main instruments will include the following:

- An atomic absorption spectrophotometer (AAS)
- An ICP-MS
- A Leco furnace
- Gold fire assay related furnace and devices

The metallurgical laboratory will undertake all necessary tests to monitor metallurgical performance, and more importantly, to improve process flowsheet and efficiency. The laboratory will be equipped with laboratory crushers, ball and stirred mills, particle size analysis devices, flotation cells, gravity concentration devices, filtering devices, balances, and pH meters.

17.4.10 Water Supply and Compressed Air

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

17.4.10.1 Fresh Water Supply System

Fresh water supply system will be installed to provide fresh water and potable water for the Project. Fresh water will be supplied to a 12 m diameter by 9 m high fresh water storage tank at the plant site. Fresh water will be distributed throughout the plant by pumping. All the fresh water pipelines outside heated buildings will be buried below the freezing level.

Fresh water will be used primarily for the following:

- Firewater for emergency use
- Reagent preparation
- Dust suppression
- Potable water supply

The fresh water tank will be full at all times and will be capable of providing at least two hours of firewater in an emergency.

The potable water will be treated via chlorination and ultraviolet lamps and stored in a 3 m diameter by 4 m high tank prior to delivery to various service points.

17.4.10.2 Process Water

Process water will consist primarily of reclaim water from the TSF and the copper-molybdenum bulk concentrate thickener, as well as fresh make-up water. Fresh and reclaimed water will be directed to a 25 m diameter by 15 m high process water storage tank, from where the water will flow by gravity to the distribution lines in the processing plant. The process water tank will be located at an elevation of 1,050 m, approximately 200 m north of the processing plant. As with fresh water, process water supply and distribution pipelines outside of the heated buildings will be buried below the freezing level.

17.4.10.3 Air Supply

Separate air service systems will supply air to the following areas:

- Flotation: Low-pressure air for flotation cells will be provided by four 34,400 m^3/h air blowers (three operating, one in standby).
- Filtering: High-pressure air for filter pressing and drying of concentrate will be provided by dedicated air compressors.
- Crushing: High-pressure air for the dust suppression (fogging) system and other services will also be provided by a separate air compressor.
- Stockpile: High-pressure air will be provided by a separate air compressor.
- Plant Services: High-pressure air will be provided by two separate air compressors.
- Instrumentation: Instrument air will be generated from two dedicated oil-free air compressors and will be dried and stored in a dedicated air receiver.

17.5 Process Control and Instrumentation

17.5.1 Overview

The plant control system will consist of a distributed control system (DCS) with personal computer-(PC) based operator interface stations (OIS) located in three separate control rooms. The DCS, in conjunction with the OIS, will perform all equipment and process interlocking, control, alarming, trending, event logging, and report generation. DCS input/output (I/O) cabinets will be located in electrical rooms throughout the plant and interconnected via a plant-wide fibre optic network.

Field instrumentation will consist of microprocessor-based “smart” type devices. Instruments will be grouped into process areas and wired to local field instrument junction boxes within those areas. Signal trunk cables will connect the field instrument junction boxes to DCS I/O cabinets.

Intelligent-type motor control centres (MCCs) will be located in the electrical rooms throughout the plant. The MCC remote operation and monitoring will be via DeviceNet (or other approved industrial communications protocol) interface to the DCS.

Programmable logic controllers (PLCs) or other third-party control systems supplied as part of mechanical packages shall be interfaced to the plant control system via ethernet network interfaces.

A supervisory expert control system is proposed to control product particle size and to optimize fresh mill feed tonnage in the grinding circuits. Expert supervisory control will be developed to optimize the set-points for controllers at the regulatory level. Mill solid concentration variable-ratio control, dilution water flow rate control, and level control will be carried out at the regulatory level to reach the control targets. The set-point modification by expert control for dilution water controller will provide good dynamic performance. The set-point adjustment for feed rate controller will ensure long-term stability in the particle size, even if mill feed hardness has changed.

Further expert system data from the DCS shall be provided to optimize pit blast patterns.

The plant control room will be manned by trained personnel 24 h/d.

17.5.1.1 Primary Crushing Facility

A control room in the primary crushing building will be provided with a single OIS. Control and monitoring of all primary crushing and conveying operations (including discharging onto the crushed mill feed stockpile) will be conducted from this location. Control and monitoring functions will include the following:

- Plugged chute detection at all transfer points
- Zero-speed switches, side-travel switches, emergency pull cords, and belt rip detection of all conveyors
- Weightometers on selected conveyors to monitor feed rates and quantities
- Equipment bearing temperatures and lubrication system status
- Vendors' instrumentation packages
- Monitoring of in-pit crushing

17.5.1.2 Process Plant

A central control room in the mill building will be provided with three OIS. Control and monitoring of all processes in the mill building, pebble crushing, reagent building, concentrate load out, and remote ancillary areas will be conducted from this location.

Control and monitoring functions will include the following:

- Grinding feed conveyors (zero-speed switches, side-travel switches, emergency pull cords, belt scales, metal detectors, and plugged chute detection)
- Grinding mills (mill speed, bearing temperatures, lubrication systems, motors, and feed rates)
- Grinding particle size monitoring and control by particle size analysers for primary grinding circuits and regrinding circuits
- Cone crushers (speed, bearing temperatures, lubrication systems, motors, and feed rates)
- Pump box, tank, and bin levels
- Variable speed pumps

- Cyclone feed density controls
- Thickeners (drives, slurry interface levels, underflow density, and flocculant addition)
- Flotation cells (level controls, reagent addition, and airflow rates)
- X-ray analysers and samplers
- Concentrate pressure filter and load out
- Reagent handling and distribution systems
- Tailings disposal
- Water storage, reclamation, and distribution, including tank-level automatic control (via ethernet remote I/O)
- Air compressors
- Vendors' instrumentation packages

17.5.1.3 Remote Monitoring

Closed circuit television (CCTV) cameras will be installed throughout the plant, with monitors in the two control rooms. The CCTV monitoring locations will include primary crushing facilities, the stockpile conveyor discharge point, the stockpile reclaim tunnel, the pebble crushing area, and the concentrate load out.

17.6 Annual Production Estimate

According to the mining production plan and the metallurgical performance outlined in Section 13.0, the annual metal productions are estimated and shown in Table 17-3. On average, the LOM average annual production is estimated to be approximately 385 kt of copper concentrate containing 28% Cu, 14.1 g/t Au, 63.1 g/t Ag, and 9,780 t molybdenum concentrate containing 50% Mo.

Table 17-3: Projected Metal Production

Year	Mill Feed	Head Grade				Copper Concentrate					Molybdenum Concentrate		
	Processed	Cu	Au	Mo	Ag	Tonnage	Grade	Recovery			Tonnage	Grade	Recovery
	(kt/a)	(%)	(g/t)	(%)	(g/t)	(t/a)	Cu (%)	Cu (%)	Au (%)	Ag (%)	(t/a)	Mo (%)	Mo (%)
1	30,095	0.293	0.151	0.012	1.00	263,882	28.0	83.8	70.3	35.4	4,138	50.0	58.8
2	48,557	0.299	0.226	0.014	1.25	434,759	28.0	84.0	74.3	40.6	7,909	50.0	59.3
3	48,586	0.296	0.237	0.013	1.32	430,879	28.0	83.9	74.8	41.9	7,633	50.0	59.2
4	48,579	0.300	0.208	0.015	1.29	437,243	28.0	84.0	73.5	41.4	8,907	50.0	59.8
5	48,705	0.293	0.174	0.015	1.17	426,589	28.0	83.8	71.7	39.2	8,826	50.0	59.7
6	48,587	0.251	0.170	0.014	1.09	360,860	28.0	82.7	71.5	37.4	7,966	50.0	59.4
7	49,096	0.287	0.157	0.015	1.22	421,089	28.0	83.7	70.7	40.1	8,981	50.0	59.7
8	48,917	0.260	0.187	0.015	1.17	376,069	28.0	83.0	72.5	39.2	8,698	50.0	59.6
9	48,939	0.281	0.190	0.015	1.30	410,086	28.0	83.5	72.6	41.6	8,810	50.0	59.7
10	48,912	0.258	0.157	0.016	1.12	374,079	28.0	82.9	70.8	38.0	9,633	50.0	60.0
11	49,916	0.218	0.145	0.014	1.18	318,048	28.0	81.7	70.0	39.4	8,599	50.0	59.5
12	50,789	0.249	0.149	0.015	1.33	373,152	28.0	82.6	70.2	42.1	8,909	50.0	59.6
13	51,224	0.254	0.152	0.018	1.31	385,180	28.0	82.8	70.4	41.8	11,367	50.0	60.4
14	51,191	0.244	0.135	0.018	1.28	367,964	28.0	82.5	69.3	41.3	10,869	50.0	60.3
15	51,487	0.228	0.108	0.017	1.19	344,235	28.0	82.0	67.0	39.5	10,788	50.0	60.2
16	51,067	0.233	0.125	0.016	1.19	349,762	28.0	82.2	68.5	39.5	10,097	50.0	60.0
17	51,357	0.236	0.122	0.015	1.15	356,280	28.0	82.3	68.3	38.7	9,424	50.0	59.8

table continues…

Year	Mill Feed	Head Grade				Copper Concentrate					Molybdenum Concentrate		
	Processed	Cu	Au	Mo	Ag	Tonnage	Grade	Recovery			Tonnage	Grade	Recovery
	(kt/a)	(%)	(g/t)	(%)	(g/t)	(t/a)	Cu (%)	Cu (%)	Au (%)	Ag (%)	(t/a)	Mo (%)	Mo (%)
18	51,406	0.278	0.124	0.020	1.29	425,658	28.0	83.5	68.4	41.5	12,257	50.0	60.7
19	50,841	0.288	0.139	0.021	1.38	437,837	28.0	83.7	69.6	43.0	12,959	50.0	60.9
20	51,228	0.268	0.129	0.026	1.23	407,995	28.0	83.2	68.8	40.3	16,166	50.0	61.6
21	50,728	0.259	0.126	0.020	1.23	389,505	28.0	82.9	68.6	40.2	12,505	50.0	60.8
Total	**1,030,207**	**0.265**	**0.157**	**0.017**	**1.23**	**8,091,151**	**28**	**83.1**	**71.0**	**40.3**	**205,439**	**50.0**	**60.1**

18.0 PROJECT INFRASTRUCTURE

18.1 Overview

The Project will require the development of a number of infrastructure items. The locations of project facilities and other infrastructure items were selected to take advantage of local topography, accommodate environmental considerations, and for efficient and convenient operation of the mine haul fleet.

Project infrastructure will include the following:

- Completion of the Galore Creek access road to the turn-off to Schaft Creek
- Installation of the More Canyon Bridge
- Access roads, including 40 km of new road, constructed north through the Mess Creek Valley to the mine site
- A TSF to safely manage the tailings and water associated with mill feed processing
- A network of site haul roads
- A complete water supply and distribution system
- A sewage disposal plant
- Process and ancillary facilities, including:
  - A primary crushing facility
  - A mine site crushed mill feed stockpile
  - A crushed mill feed stockpile
  - A pebble crushing building
  - A mill building
  - Reagent storage
  - A warehouse/truck shop/mine dry
  - A cold storage warehouse
  - Facilities for administration and an assay laboratory
  - Facilities for the storage of fuel, explosives, and concentrate
  - An emulsion plant
  - An operation and construction camp
  - A power supply and distribution network

  - Communications infrastructure
  - Diesel fuel storage and distribution

Offsite infrastructure include the following:

- Upgrade of BQLA, with installation of navigation/instrument landing system and a terminal
- A concentrate storage facility at Stewart Port (concentrate unloading, handling, and ship loading services will be provided by the port operator)

18.2 Major Layout Modifications since Feasibility Study

In comparison to the 2013 FS, numerous infrastructure design improvements have been applied to the 2021 Technical Report, including major facilities on and off site such as the WSF, TSF, process plant, ancillary buildings, and airstrip.

The new mine plan has reduced the waste, which allows elimination of the south WSF.

The improved TSF design presented in this study includes the following modifications:

- The number of embankments is reduced from three to two.
- The overall TSF footprint is smaller.
- The TSF has been relocated closer to the mine and process plant.
  - The north embankment has been relocated 2 km to the south of the 2013 FS location, and the south embankment has been relocated 750 m to the south, relative to the 2013 FS location.
- Two tailings delivery pipelines are used instead of three.

The updated TSF design reduces the starter dam construction volumes, material movement, tailings and reclaim water piping and pumping, construction schedule, and construction and operating costs.

The process plant, truck shop, camp, and most ancillary buildings have been relocated closer to the mine site and to flatter terrain to reduce the earthworks equipment, access roads, and building pad cut and fill quantities. The length of the overland crusher conveyor has been reduced significantly. Despite the process plant being located closer to the mine and further from the TSF, the total tailings slurry pipeline length remains about the same because of the relocation of the south tailings impoundment structure and removal of one tailings slurry pipeline.

An existing airfield near the Property will be upgraded and utilized for the Project. This eliminates the need of building a new airstrip and a passenger terminal at the Project site.

The 100-km fuel delivery pipeline between the plant site and Highway 37 junction in 2013 FS has been eliminated. Fuel will be delivered to site by inbound freight trucks similar to other operating mines in BC.

Inbound delivery of material and equipment will be directed from a marshalling yard near the highway 37 junction to Project site. The Tahltan Transfer Depot proposed in the 2013 FS has been eliminated.

The site preparation, pad and road cut and fill quantities have been substantially reduced due to infrastructure pad size reductions, flatter terrains, and shortening of roads. Consequently, the capital and operating costs have also been reduced. Table 1-2 shows the differences of selected project key metrics between 2013 FS and 2021 PEA.

Table 18-1 shows the differences of selected project key metrics between the 2013 FS and 2021 PEA.

Table 18-1: Key Infrastructure Metrics Summary

Project Metrics	Unit	2013 FS	2021 PEA	Difference Δ	Comment
Overland Conveyor	m	7,150	2,650	**-4,500**	Pumping slurry is more cost efficient than conveying materials. At about CAD$10,000 per metre, the estimated Capex reduction by shortening the overland conveyor is CAD$45M. The Opex reduction in power consumption is estimated CAD$1.8M per year.
Tailings Line (Total Length)	m	11,250	11,900	**+650**	The third tailings line in the 2013 FS has been eliminated as there were already two tailings lines so that one of them could stay in operation while the other line is temporarily shut down, with a reduced throughput. This practice is common among other mine operations such as Kemess Mine. Despite the increased distance between the process plant and TSF, the additional tailings pipe length is mostly offset by the elimination of the redundant pipe. The total tailings pipe length remains about the same.
Distance between Mine and Crushed Mill Feed Stockpile	m	5,600	1,400	**-4,200**	The shortened distances between the site facilities and the open pit have resulted in Capex reductions in site access road construction and Opex reductions in material haulage (e.g., fuel).
Distance between Mine and Process Plant	m	6,500	2,100	**-4,400**	
Distance between Mine and TSF	m	8,000	6,200	**-1,800**	
Distance between Mine and Truck Shop	m	4,400	1,000	**-3,400**	

table continues…

Project Metrics	Unit	2013 FS	2021 PEA	Difference Δ	Comment
Distance between Mine and Camp	m	5,000	3,200	**-1,800**	
Site Airstrip	m	1,800	0	**-1,800**	By eliminating the airstrip on site and upgrading/using the existing BQLA, the CAD$57.4M airstrip Capex has been eliminated, partially offset by the BQLA upgrade Capex and incremental staff transportation Opex. The estimated NPV of the savings is CAD$16.1M over LOM.
Fuel Delivery Pipeline	km	100	0	**-100**	By eliminating the 100-km fuel delivery pipeline from Highway 37 to site, the CAD$54.7M Capex has been eliminated. The incremental fuel haulage cost to cover the distance between Highway 37 and project site is estimated CAD$120K per year.
Site Preparation, Cut and Fill and Roads	Labour Hours/ Man-hrs	1,007,000	566,000	**-441,000**	By relocating the process plant and most ancillary facilities to flatter locations, the estimated site preparation, pads cut and fill and site access road Capex allowance has been reduced by approximately 43% in terms of construction manhours.
Capital Cost Before/After Process Plant Relocation[1]	CAD$M	127.6	72.1	**-55.5**	This is the estimated initial Capex savings from relocating the site facilities.
Operating Cost Before/After Process Plant Relocation	CAD$M/yr	15.7	8.3	**-7.4**	This is the estimated Opex savings per annum from relocating the site facilities.
TSF Initial Capital Cost	CAD$M	212.2	178.4	**-33.8**	This is the estimated initial Capex savings from relocating the TSF north embankment 2 km to the south.

[1] Excludes TSF and Mining.

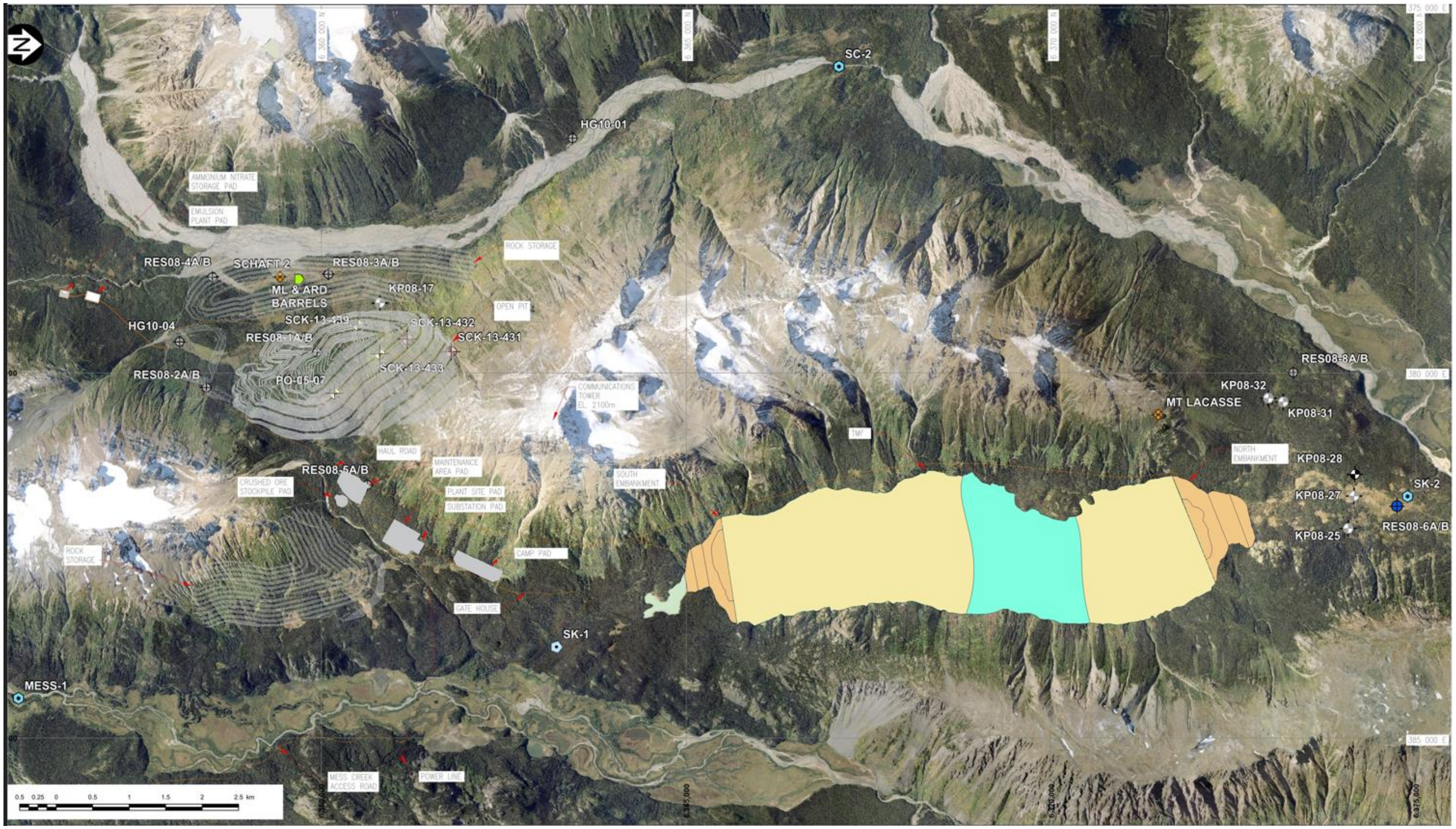
18.3 Site Layout

In comparison to 2013 FS (see Figure 18-1), the concentrator has been relocated closer to the open pit. The new TSF location shifted favourably toward the south, which is closer to the mine and concentrator. The new layout is presented in Figure 18-2 and Figure 18-3.

Figure 18-1: 2013 FS Overall Site Layout

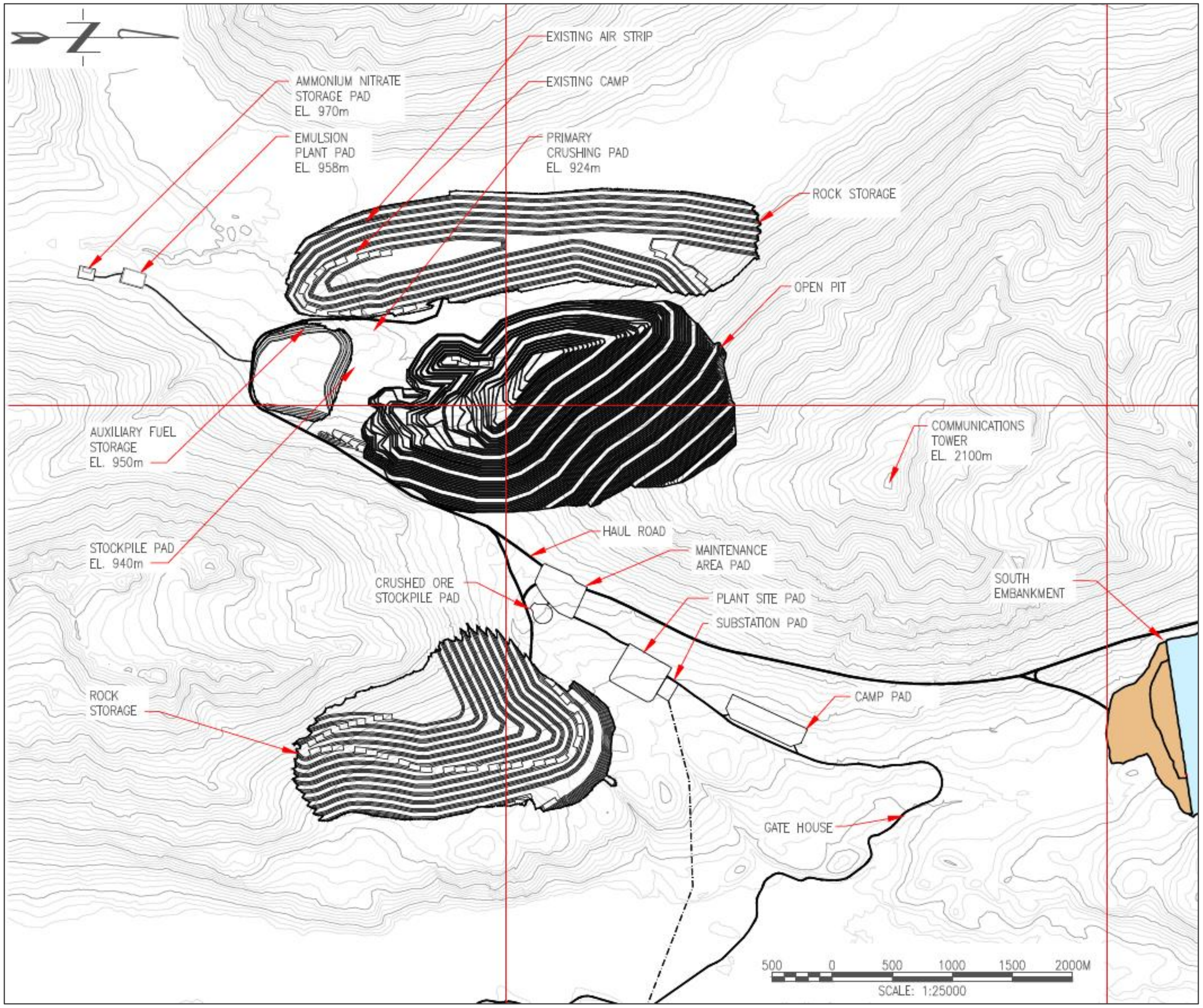
Source: Tetra Tech (2020)

Figure 18-2: 2021 PEA Overall Site Layout



Source: Tetra Tech (2020)

Figure 18-3: Site Layout – South



Source: Tetra Tech (2020)

18.4 Access Roads

For the 2013 FS, McElhanney analyzed the Project site access road requirements and developed a preliminary road design and associated construction cost estimate, which was updated for this report. A single-lane resource access road will be built to support the construction and operation of the Project. This route will include, and extend from, the first 65.2 km of the partially constructed Galore Creek Access Road, which originates at Highway 37 and heads west. Starting at kilometer 65.2 of the Galore Creek access road; 40 kms of new road will be constructed north through the Mess Creek Valley to the mine site. The entire access road will be named the Schaft Creek Access Route.

McElhanney's design established the road alignment, location, and design of stream crossings, construction recommendations and construction cost estimate for the Mess Creek Valley portion of the Schaft Creek Access Route. The access road design incorporates recommendations resulting from geotechnical and environmental investigations conducted by BGC and Rescan. McElhanney designed the road to avoid or mitigate the impact of geohazards avalanche areas, archeological sites, and environmentally sensitive sites, where possible.

As the Project is being considered as a standalone project, the completion of the Galore Creek access road and More Canyon Bridge (maximum allowable GVW 197,000 kg) have been reviewed by Ruskin/Allnorth and updated design and costs have been included in the overall scope of the Project.

18.4.1 Road Location

18.4.1.1 Route Selection

McElhanney evaluated three potential routes to the Schaft Creek site:

- The Raspberry Pass route would travel west from Highway 37 along the Willow Creek Forest Service Road, then through Mount Edziza Provincial Park through Raspberry Pass to Mess Creek, then south to the mine site.
- The Tahltan Highland route, which would branch off the Galore Creek Access Road, climb up to the Tahltan Highland Plateau, and travel along the west side of Mount Edziza Provincial Park, then drop down to and across the Mess Creek Valley.
- The Mess Creek route, which would travel north along the Mess Creek Valley to the site.

The Mess Creek route was chosen over the other potential routes, because use of this route will minimize the impact on the environment and provide safer access to the Project site.

18.4.1.2 Route Description

The Mess Creek Access Road starts at km 65.2 on the partially constructed Galore Creek Access Road and travels north from the headwaters of Mess Creek, along Mess Creek Valley. The road will then climb to the mine site located on Mount LaCasse.

Km 0+000 to 2+300

The road will travel in a northerly direction, along the western slopes of a saddle between the More Creek drainage to the south, and the headwaters of the Mess Valley to the north. Several small ponds and minor drainages are evident throughout this region. The first crossing of Little Mess Creek will be built in this area.

Km 2+300 to 3+000

At approximately Km 2+400, the road will cut into the base of a rock bluff, next to Little Mess Creek. This bluff is noted in the Terrain Stability Field Assessment (TSFA) as a significant obstacle and will require geotechnical input during design and construction. Full bench cuts of 5 to 10 m throughout this section should be expected. As a precautionary measure the right-of-way has also been widened significantly, to allow for possible construction of slope stability measures and to allow for access to the top of the cut slopes.

Km 3+000 to 4+300

In this region the route crosses several slide areas noted in the TSFA. Geotechnical input during design and construction will be necessary at these slides. Road gradients in this area are at the maximum limit of the design standards. Cut slopes are as high as 25 m. This section ends at the second crossing of Little Mess Creek.

Km 4+300 to 19+200

Traveling north along the east side of Mess Creek Valley the route crosses many transverse ridges and gullies. Short sections with high cuts and fills are present. To avoid sliver fills, the road is often in full bench cut. In some cases, it will be necessary to use mechanically stabilized earth (MSE) walls to avoid excessively large fill slopes, which could extend down to Mess Creek. Eleven crossings require engineered bridges or major culverts through this section.

Km 19+200 to 28+600

Topography in this region is generally flatter. The route crosses some large alluvial fans, which have been noted as carrying debris flows and unstable channels, and there is more overland type of construction in the design. Eight crossings require engineered bridges or major culverts in this section.

Km 28+600 to 32+100

The road will follow the base of a steeper slope, again with ridges and gullies. There is one crossing that requires an engineered culvert in this section. At Km 30+500, the road will pass the base of a region identified in the TSFA as an area of unstable (heavily dilated) bedrock. After discussion with BGC, the road alignment has been moved down onto the talus at the base of the slope to avoid disturbing the dilated bedrock. Some of the road fill will encroach into the riparian zone of Mess Creek. This will create a HADD to fisheries habitat, but has nonetheless been assessed as the better option to provide a safer route.

Km 32+100 to 33+200

In this section, the road will cross the Mess Creek Valley, on a raised causeway. This entire region is covered by flood waters regularly. A causeway will be constructed through this area with multiple culverts installed to allow relief of flood waters. The east portion of the road will allow overtopping of the road in a 100-year flood event. There are two small rocky islands that the road passes adjacent to that could be used as rock borrow sources. The two primary channels of the creek will be spanned by bridges.

Km 33+200 to 40+100

The route climbs up the alluvial fan of Shift Creek, and then turns to the north, crossing the creek and then climbing out of the Mess Creek Valley along the west side of a ravine that drains from Start Lake. The route then makes a large sweeping turn to the southwest, and climbs along the south slopes of Mount LaCasse to the terminus at a saddle between the Mess Creek and Schaft Creek Valleys. Topography in this region is generally uniform but very steep. The road will include extensive sections with grades at the upper limit of the design standards. Three crossings require engineered bridges or major culverts in this section.

18.4.1.3 Road Design

The Mess Creek Access Road is classified as a resource development road. The design criteria specified for the road required a single-lane, radio-assisted road with pull-outs, capable of carrying the legal axle loading for trucks on BC highways on a year-round basis. The road must provide vehicle access for development of the mine site, and provide year-round road access for supplies, equipment, and personnel transport. Once operations commence, the road will be used for continuous concentrate and supply haulage. The road design criteria are detailed in Table 18-2.

Table 18-2: Road Design Criteria

Item	Ministry of Forest Road Guidebook June 2002	McElhanney Proposed November 2019	Comments/Notes
Classification	Single Lane Radio Assisted	Single Lane Radio Assisted	All weather use road. 100% of gross combined vehicle weight year-round.
Average Daily Traffic	N/A	≤ 30	Estimated about 30 outbound concentrate hauls and 32 inbound supply hauls.
Design Speed (km/h)	≥30	≥30, ≤60	See Ministry of Forest Table 2.0
Minimum SSD (m)	65	65	See Ministry of Forest Table 2.0
Minimum Radius (m)	35	35	See Ministry of Forest Table 2.0 (simple curves)
Switchbacks - Minimum Radius (m) - Maximum Grade	 15 8%	 15 8%	Site-specific engineering required for switchbacks with radius less than 35 m or grades above 8%. No switchbacks are anticipated along the main access road.

table continues…

Item	Ministry of Forest Road Guidebook June 2002	McElhanney Proposed November 2019	Comments/Notes
Minimum K Factor Sag Minimum K Factor Crest	5.1 4.5	5.1 3.1 *	See Ministry of Forest Tables A2.4 & A2.5 for pre-calculated k values. *Double lane crest values used assuming "restricted" road and strict radio control.
Maximum Short-pitch Grades - Favourable (≤150 m) - Adverse (≤100 m)	 14% 10%	 12% 12%	See Ministry of Forest Table 2.0 for standards. The designer should attempt not to exceed 10% for short pitches, unless required otherwise due to difficult terrain.
Maximum Sustained Grades - Favourable - Adverse	 12% 8%	 10%** 10%**	See Ministry of Forest Table 2.0 for standards. "Adverse" grades are for loaded trucks on an uphill grade. "Favourable" grades are for loaded trucks on a downhill grade. **The designer should attempt to maintain nominal 10% maximum sustained grades, unless required otherwise due to terrain restraints.
Road Width (m)	≥4.0, ≤6.0	≥5, ≤6.0	The primary road finished surface width shall be 6.0 m (single lane). Double lane 8.0 m. Subgrade widths are 1.2 m wider.
Pull-out width (m)	Add 4.0 m	Add 4.0 m	With pull-outs "intervisible" (nominal 3 to 4 km). Standard turnout length 30 m including tapers.
Right-of-way (m)	Variable	≥30	Variable width as required for road prism, construction, snow removal, maintenance, powerline, etc.
Design Vehicle	Lowbed Vehicles	-	GCVW B-Train Configurations (BTD) Maximum 63,500 kg. Structures BCFS L-100 Maximum 90,680 kg.
Structure Designs	-	L100 Loading	Structure designs to meet to L-100 off-highway logging truck requirements, consistent with structure designs on Galore Creek Access Road.

18.4.1.4 Stream Crossings and Bridge Design

The More Canyon Bridge, located at km 48 on the Galore Creek access road, is the only major bridge to be constructed. The bridge has a span of approximately 200 m and this study carries the Ruskin/Allnorth design and costing.

The stream crossings were designed according to the following criteria:

- All bridges will withstand 1-in-100-year flood conditions and maintain 1.5 m debris clearance.

- All fish bearing streams will be bridged or will be outfitted with open-bottomed arches to protect the channel.
- Culverts will be sized for 1-in-100-year flood conditions, and cross-drain culverts will be installed as appropriate to maintain natural drainage. The maximum amount of spacing between culverts will be governed by corresponding road gradients and soil types outlined in the Forest Road Engineering Guidebook, published by the BC Ministry of Forests.
- Where required, crossings will meet the requirements of the federal *Navigable Waters Protection Act*.

18.4.1.5 Hydrology

McElhanney conducted a preliminary hydrology assessment of the streams, which utilized Government of BC isoline mapping, the “Beaumont” formula, channel capacity, and regional analysis. BGC conducted an independent hydrological assessment—including estimates of the 1-in-100-year peak flows for culvert and bridge design—for 12 stream crossings along the Mess Creek Access Route. BGC used two methods to estimate peak flows for the select stream crossings on the Mess Creek Access Route:

- a regional hydrological analysis
- the BC peak flow isoline method, as a check of the flood flows

The 1-in-100-year flows utilized for the stream crossings were determined from the two assessments and the actual catchment area. The flows utilized are shown on the general arrangement drawings for each crossing.

18.4.1.6 Drainage Structures

Drainage structures were designed in general accordance with standard industry practices consisting of CAN/CSA-S6-2006 Design Specifications modified to BCFS L100 Loading (90,680 kg gross vehicle weight) and the BC Ministry of Forests and Range Bridge Design Manual. A general arrangement drawing has been prepared for each of the bridge and major culvert crossings. Each general arrangement drawing shows:

- the watercourse
- the bridge alignment
- the abutment configuration
- the bridge length and clearance
- the direct gradient
- the bridge approaches and turnouts

Typical bridge detail specifications will supplement the general arrangement drawing, for cost estimating and permitting purposes. Bridge structures will typically consist of a steel girder superstructure with treated timber deck panels. Treated timber deck panels should, with minor maintenance, meet the 21-year life expectancy of the mine and will be relatively simple to replace if necessary.

Standard debris clearance of 1.5 m from the 1-in-100-year flood elevation to the underside of the girders is specified for all bridges except for those requiring additional clearance to meet the navigable waters and/or geotechnical requirements.

18.4.2 Operation and Maintenance

This access road will undergo scheduled maintenance such as snow removal, sanding, ditch clearing, rock scaling, and spot gravelling and grading. All structures, including bridges, culverts, and retaining walls, will also undergo scheduled inspection and maintenance for items such as:

- riprap replacement
- clearing of log jams
- semi-annual inspections
- scour damage repair
- curbs, deck and delineators replacement
- appropriate signage and sign maintenance
- corrosion
- traffic barriers

18.4.2.1 Construction Requirements

The road alignment has been modified in response to recommendations by BGC to improve the safety of the road. All areas of major rock cuts will require the review of a qualified professional during construction to ensure stability of the areas.

18.4.2.2 Avalanche Forecasting and Control

Numerous areas have been identified (see Table 18-3), as posing an avalanche risk as shown in BGC's Access Route Terrain and Geohazards Mapping report. The road has been located to minimize the impact from an avalanche. During construction a qualified avalanche professional should review the possible impact on the road to determine if structures are required to protect the road or if monitoring and control will be sufficient to ensure continuous road access.

During winter operation of the mine, an avalanche safety and mitigation program will be required to ensure the safe operation of the road.

Table 18-3: Major Avalanche Chute Locations

From (km)	To (km)
1.8	2.6
2.8	3.8
8.3	8.4
10.6	10.9
11.6	12.1
22.0	22.7
23.1	23.5
38.2	38.3

Special Structures

The Mess Creek Access Road will cross the Mess Creek floodplain. A causeway has been designed that will allow the relief of flood water through culverts and act as a spillway, by overtopping during flood events. The causeway design and location is shown in Figure 18-4.

At km 30.5, the road will pass the base of an unstable region. To avoid this area, the route was relocated onto the talus slope at the base of the rock with fill slopes extending onto the riparian zone of Mess Creek. The movement will result in a HADD. The realignment is shown in Figure 18-5.

Authorized Access

The road has been designed as a single lane radio-assisted resource road and it will be necessary to ensure that only authorized users are allowed on this road. A gate house will be installed near Highway 37 junction to control access.

Figure 18-4: Potential HADD Site – km 32.1 to 33

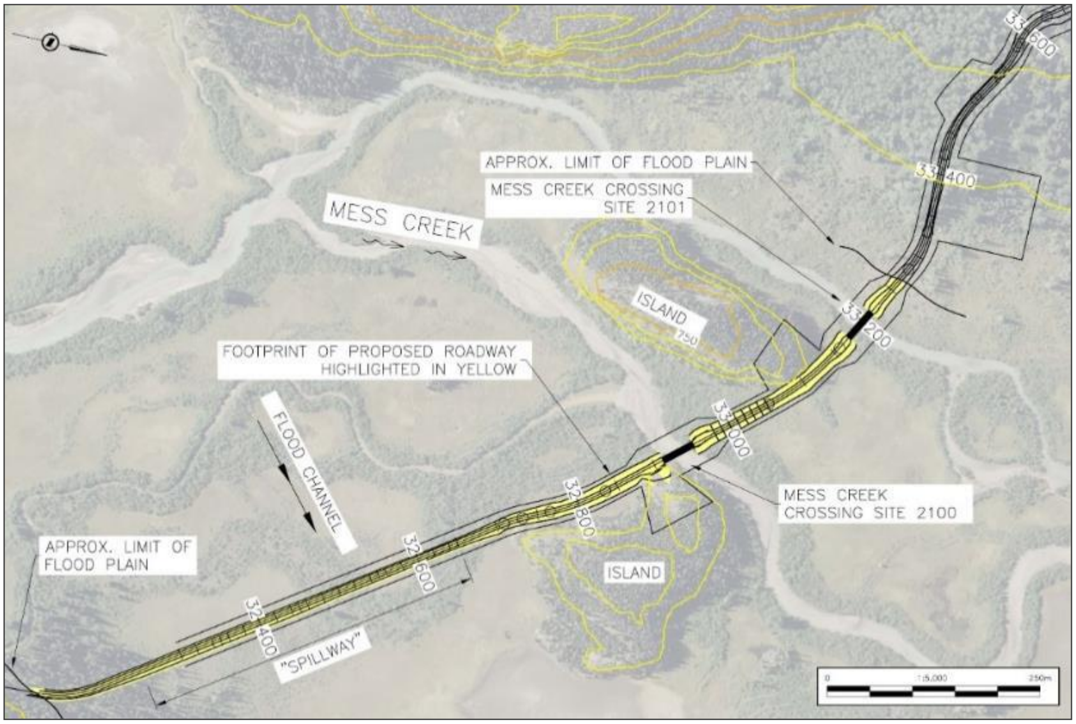
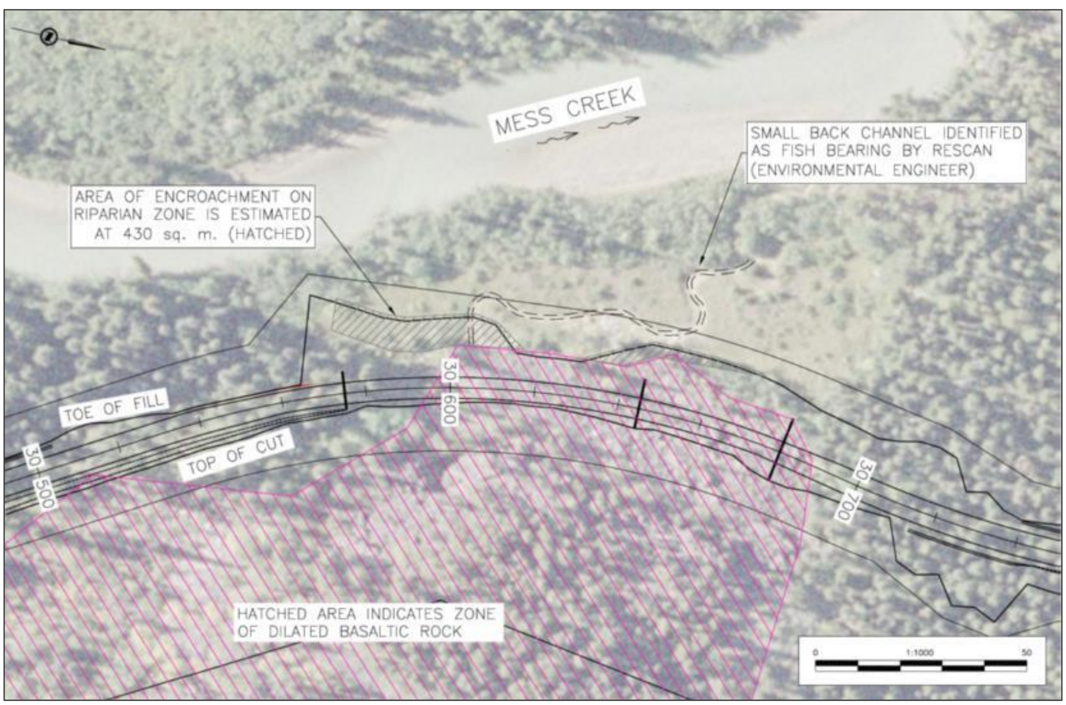


Figure 18-5: Potential HADD Site – km 30.54 to 30.7



18.5 Bob Quinn Lake Airport Upgrade

Mine personnel will be transported to the site by regular daily charter flights to BQLA, located approximately 120 km by road from site, and the scheduled bus service between BQLA and site. The BQLA will require upgrades to support the transportation of mine personnel during construction and operation.

18.5.1 Runway and Navigation Instrument Upgrade

The upgraded runway will be 30 m wide and 1,400 m long, with a gravel surface and navigational lights capable of receiving chartered aircraft with a capacity of up to 19 passengers. An appropriate level of navigation instruments will be installed to provide safe take-offs and landings during poor weather conditions.

18.5.2 Airport Terminal

A modular-type airport office building will be constructed. The control room will be equipped with a navigation system and other communication and control equipment required for safe airport operations. The office building will include a waiting room with washroom facilities. The airport will include facilities for emergency jet fuel storage.

18.6 Water Supply and Distribution

The permanent operation and construction camp will require approximately 320 L/person/day or 640 m^3/d of potable water, supplying a peak of about 1,900 direct workers, staff and indirect operations personnel during the construction phase of the Project. When construction is complete, water requirements will decrease to an expected 192 m^3/d, supplying 600 permanent operation staff during the LOM. A potable water package treatment unit will be located by the permanent camp, which will have a bottling facility for the distribution of drinking water. The potable water will undergo chlorination and ultraviolet light treatment.

Fresh water will be supplied by groundwater wells located close to the individual locations equipped with storage tanks in order to avoid long pipe runs.

Fresh water will be pumped from the local well to fresh and firewater storage tanks, located near each major facility. To ensure adequate firewater levels, two individual reservoirs inside the tank will be separated by an internal standpipe to draw fresh water from the top of the fresh water standpipe. The fresh water tank will be equipped with level detection control of the fresh water supply pumps located nearby.

The fire/fresh water tank for the process building/pebble crusher/reagent building will be located inside the process building in order to prevent freezing during the winter months. The storage tank will have a total storage capacity of 785 m^3, which includes 685 m^3 for firewater storage and the remaining 100 m^3 for fresh water storage. The pressure will be maintained by fresh water booster pumps, jockey pumps, and diesel back-up pumps. A plant site alarm will signal a low system pressure condition. The hydrant system will be wall mounted with the pipe ring installed inside the process building.

The fire/fresh water tank for the truck shop/administration/camp area will be located inside the truck shop building in order to prevent freezing during the winter months. The emulsion plant and the primary crusher area will have their own storage tank, well, and pumping system for fire water and fresh water. Due to the long distances between facility areas and the hard rock terrain, it is more economical to have dedicated systems at each facility rather than run buried lines in blasted trenches from a central system.

Hose cabinets will be installed in the concentrator and ancillary facilities, supplemented by portable fire extinguishers. Fire alarm panels, flow devices, pressure switches, alarm valves, manual pull stations, detectors, and audible alarms will be installed in individually protected areas. Activation of any sprinkler system or initiation device will trigger an alarm at the central fire alarm panel, located in the main central control room. An automatic sprinkler system will protect all hydraulic equipment with reservoirs over 378 L. Mechanical equipment will be equipped with interlocks to automatically shut down equipment when sprinklers are triggered. Heat and smoke detectors will be installed in electrical rooms, process instrumentation control rooms, and all other occupied areas that do not have sprinklers. Hose stations installed at 50 m intervals and automatic sprinklers over the belt will protect the underground conveyor. Automatic wet sprinkler systems will be installed throughout ancillary buildings.

Emergency showers and eyewash stations will be installed throughout the process building.

18.7 Waste Disposal

18.7.1 Sewage Disposal

During the construction phase, 120 L/person/day or 240 m^3/d of sewage will be collected and treated. The proposed treatment plant will be a membrane bioreactor system and chemical phosphorus removal to meet 1 mg/L total phosphorus. Each train will include fine pre-screening, equalization (surge capacity of seven hours at average flow or three hours at the peak hourly flow), aeration, membrane filtration, chemical phosphorus removal, ultraviolet disinfection to 200 CFU/100 mL fecal coliforms. Aluminum tank, membrane and sludge tank will be coated for protection for chemical cleaning. The building will have 12 air exchanges per hour.

Solid and liquid material will be separated at the sewage treatment plant via a series of modules; liquid streams will flow to the tailings pond, and a specialized licenced contractor will pump out and truck solid material away twice each year.

During the first year of construction, the effluent from the sewage treatment plant will flow to a holding lagoon located within the tailing basin. A design flow of 400 m^3/d has been recommended for 1,900-person fly-in/fly-out workforce. This conservative estimate accounts for the possibility of temporary spikes in employee numbers during short maintenance outages.

18.7.2 Domestic Waste Disposal

Construction and industrial waste will be landfilled within the WSB laydown area. Non-hazardous solid construction industrial waste will also be buried in the WSB laydown area. Domestic waste from the construction camp and operating areas will be incinerated with the ash buried in the WSB laydown area.

Scrap metal will be collected in bins and recycled by a qualified local contractor.

Several forms of domestic and industrial solid waste will be generated over the life cycle of the mine. All avenues of re-use, reduction and recycling of materials will be examined and implemented prior to disposal of any waste.

18.8 Tailings Storage Facility and Tailings Management

18.8.1 Tailings Storage Facility Alternatives Assessments

Alternatives assessments for tailings management were completed for the Schaft Creek project that considered the following:

- TSF Location Alternatives
- Tailings Technology Alternatives
- TSF Embankment Siting Alternatives
- TSF Construction Material Alternatives

Conventional tailings storage in the Start Creek / Skeeter Creek Valley is the preferred tailings management concept for the Project.

18.8.2 Tailings Storage Facility Design Basis

The process plant will comprise two parallel trains, each operating at 66,500 t/d. Initial production from the mill is expected to be 66,500 t/d, which will increase to 133,000 t/d when the second train is commissioned. The TSF will store approximately 1 Bt of tailings during an operating mine life of approximately 21 years. The capacity of the TSF can be increased by raising the embankments to accommodate a maximum of 2 Bt with minimal changes to the overall footprint of the TSF. The design flood and earthquake events are the Probable Maximum Flood (PMF) and the Maximum Credible Earthquake (MCE), respectively. These criteria are based on the Dam Classification described in subsequent sections. The TSF will be designed to eliminate credible failure modes to the extent practical by selecting conservative design criteria.

18.8.3 Dam Classification

A dam classification was carried out to enable appropriate design earthquake and flood events to be determined for the TSF. The selection of the design flood and earthquake events is based on the classification criteria provided by the Canadian Dam Association (CDA) Dam Safety Guidelines (2013, 2014). These guidelines are the design standard for major impoundments, water management facilities, and dams as specified in Section 10 of the Health, Safety and Reclamation Code for Mines in British Columbia (2021). The hazard classification of the TSF embankments has been determined to be “Extreme” due to the potential consequence of environmental loss in the event of a hypothetical dam breach. It is recognized that this hazard classification will need to be reviewed at the next level of study.

18.8.3.1 Design Earthquakes

The CDA guidelines (CDA, 2013; 2014) recommend that a dam classified as “Extreme” be designed for a probabilistically derived event (known as the Earthquake Design Ground Motion) having an annual exceedance probability (AEP) equivalent to the 1 in 10,000 (or the Maximum Credible Earthquake [MCE] return period events. The maximum ground acceleration for the 1 in 10,000-year earthquake (approximately 0.108 g) has been adopted as the Maximum Design Earthquake (MDE).

18.8.3.2 Design Storm Events and Inflow Design Flood

Based on the “Extreme” hazard classification assigned to the TSF, an appropriate inflow design flood (IDF) is a probabilistically derived event equivalent to the probable maximum flood (PMF) during operations and post-closure.

An IDF equivalent to two concurrent PMF events has been selected for design of the TSF for this study. This large storm event was selected recognizing the evolving guidance with respect to the design of tailings facilities and changing industry expectations for the management of tailings facilities, particularly with respect to water management and stormwater management.

The TSF is designed with sufficient capacity and freeboard to store the IDF at all times during operations. The storm storage volume required during operations is approximately 44 Mm^3.

18.8.4 Tailings Storage Facility Design

18.8.4.1 General

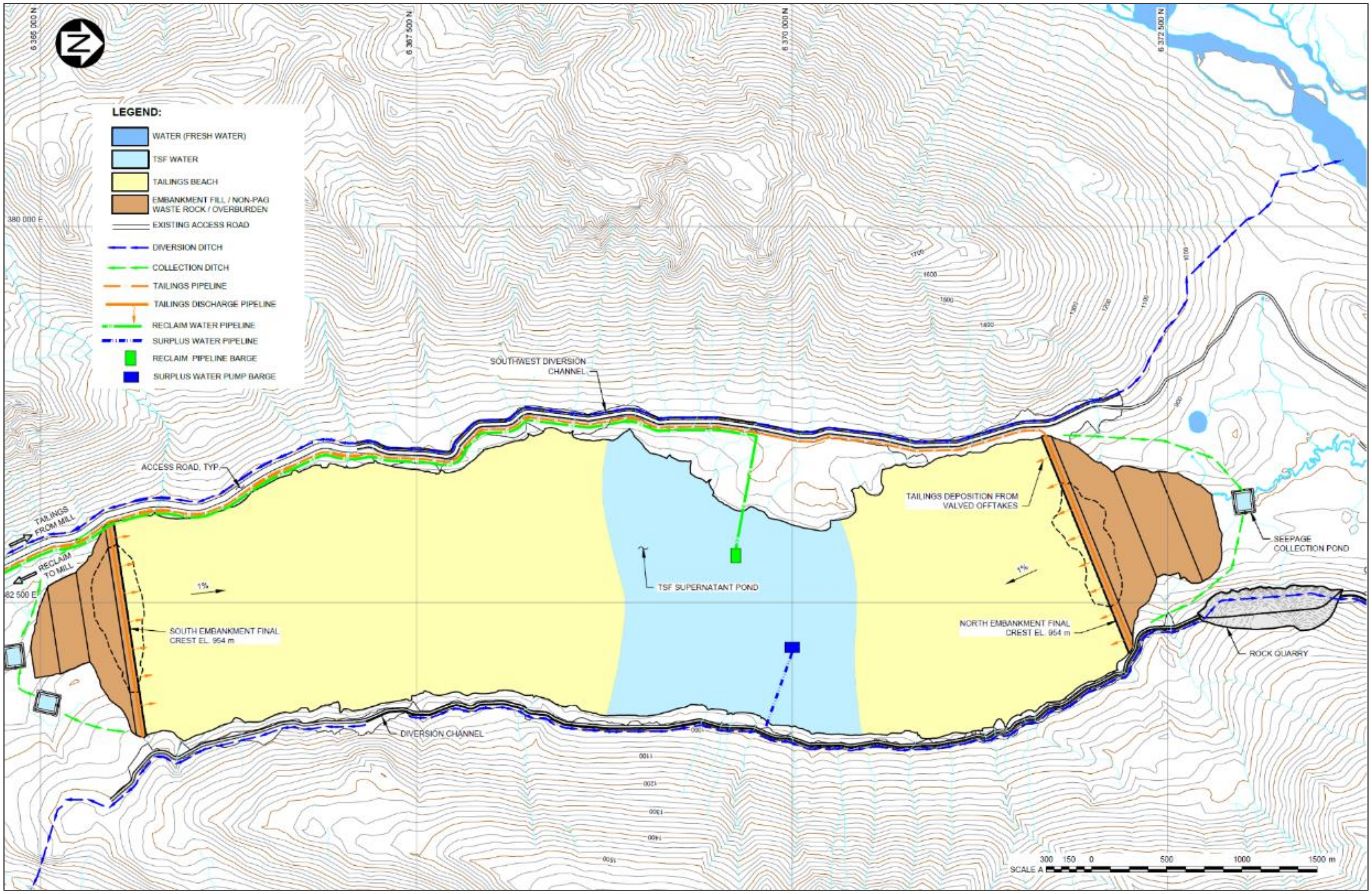
Tailings will be impounded in a TSF in the Start Creek and Skeeter Creek Valley, located to the northeast of the open pit and due north of the plant site. The overall mine site layout is shown on Figure 18-2 and the TSF is illustrated on Figure 18-6.

The TSF concept includes the following:

- Two zoned embankments constructed using cyclone sand
- Upslope surface water diversion channels
- Seepage and embankment runoff collection systems
- Tailings transport and deposition system
- Reclaim water system
- Surplus water removal system
- Extensive tailings beaches
- Supernatant water pond

The depth-area-capacity relationship for the TSF is shown on Figure 18-7. The TSF embankments would be expanded in stages as the elevation of the stored tailings increases with time.

Figure 18-6: TSF General Arrangement



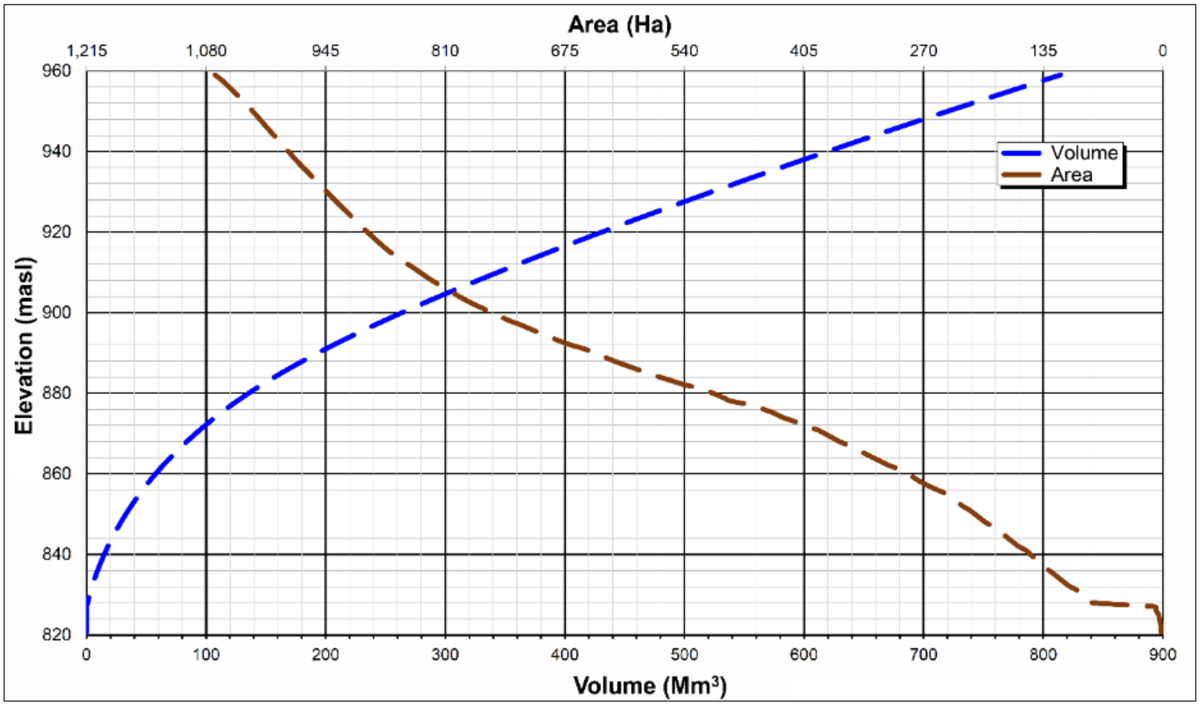
Source: Knight Piésold (2020)

Note: Coordinate grid is UTM (NAD83) Zone 9V.

20 m interval contours provided by Eagle Mapping (2007).

Open pit, waste rock dumps, and other project infrastructure by others.

Figure 18-7: TSF Depth-Area-Capacity Curve



Source: Knight Piésold (2019)

18.8.4.2 Embankments

The north embankment has been relocated approximately 2 km south of the location previously proposed in the 2013 FS (CFM, 2013) to reduce initial construction requirements and starter embankment construction volumes. The south embankment has been relocated by 750 m to the south, relative to the 2013 FS location. This more efficient arrangement also allows for future expansion of the Project beyond the current 21 year LOM for up to 2 Bt without substantially changing the footprint.

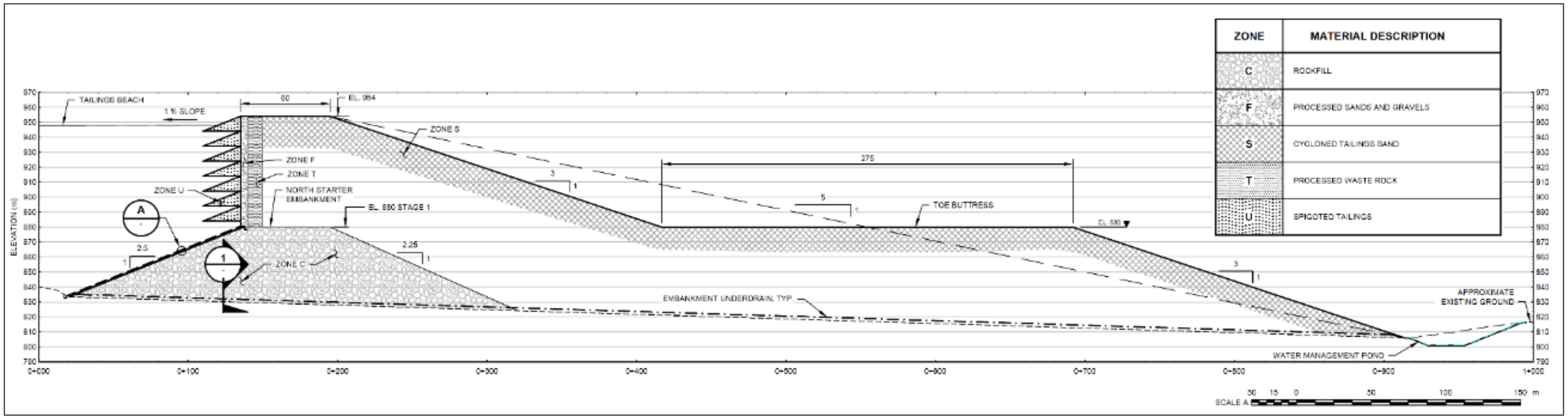
The north and south embankments will be constructed using cyclone sand. The north starter embankment will be constructed using rockfill from a local quarry and the south starter embankment will use non-acid generating waste rock material from open pit mining activities. The starter embankments will include a low-permeability geomembrane liner on the upstream embankment face; the raised portions will include a lower permeability central zone, selective tailings deposition, and downstream filter and transition zones.

The TSF embankments will be progressively raised with tailings sand generated from the underflow of a cyclone sand plant.

The ongoing embankment raises will have filter and transition zones that will be supported by the upstream and downstream shell zones. The tailings beach, which will largely comprise cyclone overflow, will have a relatively low permeability and will provide confinement for the supernatant pond and mitigate drainage towards the embankment. An underdrainage system will be constructed at the base of the embankments to reduce porewater pressure in the cyclone sand shell zone. The underdrainage system will collect seepage through the embankment and direct it to the water management ponds.

The embankments will be constructed with a 3H:1V downstream slope and a toe buttress to provide an effective downstream slope of approximately 5H:1V. A schematic typical cross-section for the TSF embankments is shown on Figure 18-8.

Figure 18-8: TSF Embankment Schematic Cross-section



Knight Piésold (2020)

18.8.4.3 Access Roads

A temporary construction road will be built along the base of the Skeeter Creek Valley to provide access for equipment to the north embankment area for construction of the starter dam. An access road will be constructed along the west side of Skeeter Valley to provide a route for mine trucks to deliver waste rock to the south embankment for starter embankment construction and to provide access to the north embankment. A service access road and utility corridor will be constructed along the east side of the TSF for the east diversion channel and surplus water discharge pipeline.

18.8.4.4 Surface Water Diversion Channels

Major diversion structures include the east and west TSF diversion channels shown on Figure 18-6. The west diversion channel is split into two reaches: the northwest and southwest diversion channels. The northwest channel will divert a portion of the TSF upslope catchment northwest to Schaft Creek and southwest divert catchment runoff south to Start Creek.

18.8.4.5 Tailings Deposition Strategy

Tailings will be discharged through a series of offtakes (spigots) located along the north and south embankments.

Tailings deposition will be prioritized to develop extensive tailings beaches to maintain the supernatant pond remote from the embankments. Successful management of tailings deposition and beach development will reduce seepage through the embankments and ensure that water is accessible and suitable for reuse in the process.

Tailings from the underflow of the cyclone sand plant will be selectively deposited and compacted in sand cells during the summer months to develop a sandy shell zone for ongoing embankment crest raises.

18.8.4.6 Seepage and Impacted Surface Water Management

Seepage from the TSF will be largely controlled by the tailings beach, zoned embankment, and the embankment drains that will discharge into the water management ponds downgradient of the embankments. Surface water runoff from the embankment face, seepage through the embankment fill material, or other runoff and seepage from impacted areas in the vicinity of the TSF will be collected and directed to the water management ponds located at topographic low points along the downstream toe of the embankments.

Water collected in the downstream water management ponds will be continuously monitored and pumped back into the TSF if required.

18.8.5 TSF Construction Methodology

18.8.5.1 General

The TSF will be constructed annually to the crest elevations shown in Table 18-4. Where crest elevations differ between north and south embankments, the elevations have been denoted with an "N" or an "S".

Table 18-4: TSF Construction Sequencing – Annual Crest Raise Elevation

Year	Crest Elevation (masl)	Embankment Construction
-2	880	North
-1	880	North
1	880	North
2	887(N), 905(S)	North, South
3	892(N), 905(S)	North, South
4	897(N), 905(S)	North, South
5	901(N), 905(S)	North, South
6	905(N), 905(S)	North, South
7	910	North, South
8	914	North, South
9	918	North, South
10	921	North, South
11	925	North, South
12	928	North, South
13	932	North, South
14	935	North, South
15	938	North, South
16	942	North, South
17	945	North, South
18	949	North, South
19	952	North, South
20	954	North, South
21	954	North, South
22	954	North, South

Source: Knight Piésold (2020)

Stage 1 of the north embankment will be constructed by a contractor, while ongoing embankment raises will be built by a combination of a contractor and mine-owned equipment.

18.8.5.2 Scheduling Considerations

The primary scheduling constraints are seasonal changes in temperature, precipitation, snowpack, and stream flow. These include the following:

- Environmental
  - Maintaining the instream flow requirement for aquatic life once the construction of the embankment isolates the creeks from the upstream catchments. The instream flow requirements will vary seasonally.
  - Water quality downstream of the north embankment in Skeeter Creek (erosion and sediment control from impacted areas) and downstream of the south embankment in Start Creek.
- Precipitation and Temperature
  - Access restrictions due to snowfall and heavy rain in late fall and winter months.
  - Lower construction productivities due to winter conditions (low temperatures, ice hazards on access roads, and frozen construction materials).
  - Cold weather limitations on excavating, placing and compacting cohesive materials, and installing a geosynthetic liner (starter embankments only).
  - Avalanche hazard and snow clearing.
  - Spring freshet, storm events and flood flows (dewatering and flooding).

18.8.5.3 Winter Construction Considerations

Winter conditions are anticipated to make construction of the TSF embankment using the cyclone sand material challenging. The cyclone plant will operate during non-freezing months (assumed to be five months of the year) and annual TSF raise construction is anticipated to take place over this period.

Whole tailings will be deposited in the TSF during freezing months (the remaining seven months of the year).

18.8.6 TSF Water Management Plan

Construction water management at the TSF will commence approximately two years prior to mill start-up and coincide with initial construction of the facility. This phase is characterized by extensive clearing, grubbing, and stripping; the development of access roads and haul roads; and establishing water management and erosion and sediment control systems.

Non-contact water will be diverted around the TSF during operations to the extent practical and contact runoff will be stored within the TSF. Process water will be discharged into the TSF with the tailings slurry and supernatant water, and will be reclaimed back to the mill for use in mill feed processing. Surplus water will be removed from the facility to prevent the accumulation of water and discharged to

Schaft Creek. Seepage from the TSF will be collected in the water management ponds downstream of the embankments and recycled to the TSF supernatant pond.

18.8.7 Water Balance

18.8.7.1 General

A stochastic monthly operational mine site water balance was developed using the GoldSim© software package. The intent of the modelling is to estimate the potential magnitude and extent of water surplus and/or deficit conditions in the TSF based on a range of climatic conditions. The modelling timeline included one pre-production year (Year -1), and 21 years of operation (Years 1 to 21) at an average mill throughput of 133,000 t/d. The model is shown schematically on Figure 18-9 and incorporates the following major project components:

- Open pit
- Mill
- TSF
- Waste rock storage facilities

18.8.7.2 Model Assumptions

Average Climatic Conditions

The mean annual precipitation (MAP) used in the model was 850 mm, with 37% of the annual precipitation falling as rain and the remainder as snow. The annual average long-term potential evapotranspiration for the TSF site was estimated to be 433 mm.

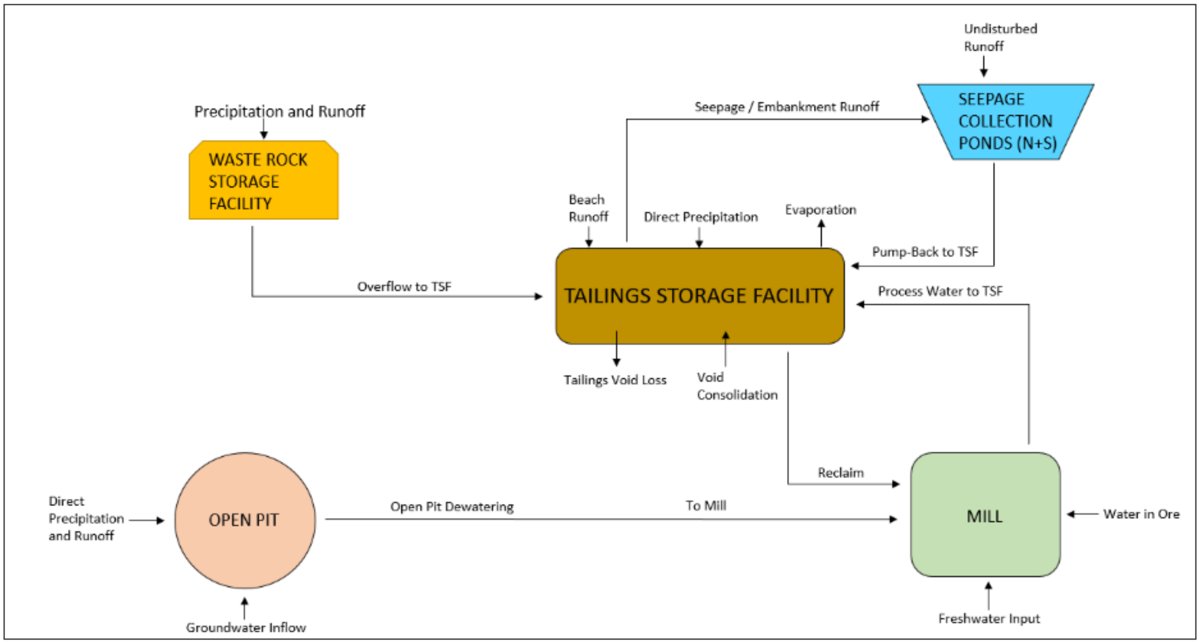
Stochastic Inputs

The variability of climatic conditions was addressed using a stochastic version of the water balance model that included Monte Carlo type simulation techniques. The monthly climatic parameters were modelled as probability distributions rather than simply as mean values. The model was cycled through 1,000 realizations to simulate a range of possible wet and dry conditions.

18.8.7.3 Water Balance Results

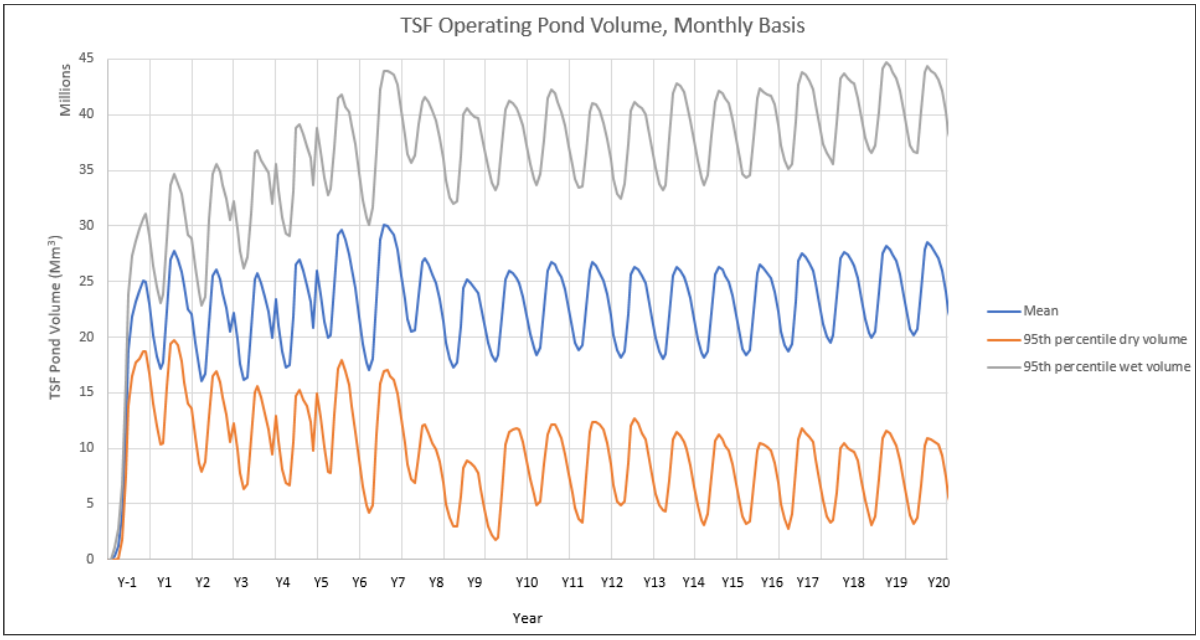
Model results were used to estimate the amount of surplus water reporting to the TSF on a monthly basis. Excess water from the TSF will be removed on a year-round basis, and will fluctuate between 1,500 m^3/hr and 3,000 m^3/hr. This maintains the mean operating pond volume within the target range. Figure 18-10 illustrates the TSF operating pond volume on a monthly basis over the LOM, with the upper and lower bound pond volume estimates defined as the 95th percentile wet and dry scenarios. The 95th percentile wet and dry conditions exceed the maximum operating pond volume target. In these situations, surplus water removal rates will be adjusted to keep the pond volume within the target operating range.

Figure 18-9: Mine Site Water Balance Schematic



Source: Knight Piésold (2020)

Figure 18-10: TSF Monthly Operating Pond Volume



Source: Knight Piésold (2020)

18.8.8 Tailing Management Systems

18.8.8.1 General

Two interconnected 42″ HDPE pipelines will transport the tailings between the mill and the TSF. Both lines will serve both mill trains during normal operating conditions. One line may be shut down for pipeline inspection, maintenance, and rotation while the other line continues operating, with controlled flow curtailment. The tailing lines will follow the TSF access roads and maintain a slope from the mill to the TSF. The tailings system is divided into two components: the transport pipelines from the mill to the TSF and the valved discharge or deposition pipelines around the perimeter of the TSF.

18.8.8.2 Tailings Transport

It is expected that the tailings will flow to the embankments by gravity under normal operating conditions until the end of approximately Year 2. Tailings pumping is required to reach the TSF from Year 3 onwards.

Tailings pumps will be installed at the plant site to provide the required pressure for tailings delivery to all areas of the TSF under flow conditions up to 30% greater than the nominal design flow.

Knife gate valves with hydraulic actuators will be used for flow control in the tailings system. Crossovers between tailings lines will be provided at the mill, at the southwest, and the northwest corners of the TSF. Drainage outlets for the tailings lines, as well as flushing connections to the reclaim line, will be provided at the southwest corner of the TSF.

18.8.8.3 Tailings Deposition

The tailings deposition pipelines will carry tailings from the delivery lines to each of the deposition points along the TSF embankments. The tailings line will be adjusted and relocated with each embankment raise.

Discharge points will initially be established at each end and at the centre of the embankment crests, with large diameter single point offtakes. Additional discharge points will be installed along the length of the embankment crests, as required for managing tailings beach development. It has been assumed that spigot offtakes will be installed at a nominal 150 m spacing along the embankment crests.

18.8.9 Reclaim Water Systems

18.8.9.1 General

Reclaim water will be pumped from the TSF surface water (supernatant) pond by two floating pump stations to a process water head tank at the mill for reuse in the process. The average elevation of the tailings supernatant pond will increase over the Project life, with seasonal variations due to snowmelt, runoff, precipitation, and evaporation. The rising elevation of the tailings supernatant pond will reduce the elevation difference between the mill and the TSF, therefore, reducing the pumping head requirements over the life of the Project and allowing modifications to the pumps, or their mode of operation, to minimize power consumption.

18.8.9.2 Reclaim Barges

The nominal pumping capacity of each of the two barges will match 50% of the peak process demand (i.e., one mill line). Each pump station will include an extra standby pump. Each barge will be linked to the shore by mooring lines, a hinged access walkway, and a pontoon supported reclaim discharge line with connecting ball joints. The pump stations will be periodically relocated closer to shore as the elevation of the supernatant pond rises.

18.8.9.3 Reclaim Pipelines

The discharge line from each pump station will be a 36″ diameter steel pipe and will include an isolation valve and a check valve to facilitate barge relocation and maintenance. The two discharge lines will join into a single 42″ diameter steel reclaim pipeline at an elevation above the ultimate embankment crest.

This section of the pipeline will be permanent for the life of the Project.

18.8.10 Surplus Water System

The surplus water system will be used to remove excess water (greater than the maximum operating pond volume) from the TSF supernatant pond to prevent excessive accumulation of water within the facility and to maintain appropriate beach lengths. Surplus water will be pumped from the TSF supernatant pond by a floating pump station and discharged into the east diversion channel, which discharges into Schaft Creek to the north of the TSF. The average base elevation of the tailings supernatant pond will increase steadily over the Project life as tailings are progressively deposited in the TSF. The pond volume will vary seasonally due to snowmelt, runoff, precipitation, and evaporation. The rising elevation will substantially reduce the pumping head requirements over the life of the Project and allow modifications to the pumps, or their mode of operation, to minimize power consumption.

18.8.11 Instrumentation and Monitoring

Instrumentation will be installed in the TSF embankments, foundations, and ancillary structures during construction and throughout the life of the Project. The instrumentation will be monitored during the construction and operation of the TSF to assess performance against Quantifiable Performance Objectives (QPOs) and to identify any conditions different to those assumed during design. Trigger-Action Response Plans (TARPs) will inform ongoing designs and/or remediation work that can be implemented to respond to changing conditions, should the need arise. QPOs and TARPs will be described and outlined in an Operation, Maintenance, and Surveillance (OMS) Manual, which will be prepared as part of the *Mines Act* Permit Application (MAPA) process, and amended and updated throughout the life of the Project.

Geotechnical instrumentation, including vibrating wire piezometers, slope inclinometers, and survey monuments, will be installed along instrumentation planes within each of the embankments. Groundwater monitoring wells will be installed at suitable locations downstream of each embankment. Flow and level monitoring instrumentation will be installed in the embankment foundation drainage system and seepage collection ponds. Other specialized monitoring instruments will be implemented as required. Surveys will be conducted regularly of the TSF embankments, impoundment, and surrounding areas.

Instrumentation monitoring will be routinely completed during construction, operations, and active closure. Measurements will be taken and analyzed regularly during construction to monitor the response of the foundations and embankments to fill loading. In addition to the routine inspections carried out by mine personnel on a shift/daily/weekly and monthly basis, a Dam Safety Inspection will be conducted on an annual basis to ensure it is operating in a safe and efficient manner, according to the *Health, Safety and Reclamation Code for Mines in British Columbia* (EMLI, 2021). A dam safety review will be conducted every five years by a qualified geotechnical engineer.

Criteria for upset conditions for monitoring and mitigation and response measures will be outlined in an Emergency Preparedness and Response Plan (EPRP), which will also be developed as part of the MAPA process and updated periodically throughout the life of the Project.

18.8.12 Foundation Conditions

Knight Piésold completed a site investigation program in the summer of 2008 to collect geotechnical and hydrogeological information to support the 2008 PFS. The 2008 site investigation program supplemented geotechnical and hydrogeological information obtained by DST Consulting Engineers Inc. during the 2007 investigation. The TSF concept has evolved since the historical site investigations and no drilling has therefore been completed at the current proposed embankment locations. Additional site investigation programs will be required to collect geotechnical and hydrogeological information, and to characterize foundation conditions for the current embankment locations to support advancement of the Project.

The 2008 field program, conducted between June and September, comprised 12 geotechnical drill holes (10 vertical, 2 inclined), 12 paired hydrogeological drill holes at 6 sites, 88 test pits, and 15 Dutch auger sites within the TSF area. In situ testing, piezometer installation, and groundwater level monitoring were completed in selected drill holes. Soil and rock samples were collected for laboratory test work. Additional surface geological mapping was also conducted to characterize surficial materials and to define potential borrow sources for construction materials.

18.9 Plant Ancillary Facilities

18.9.1 Architectural Design Basis – Process and Ancillary Facilities

The Architectural Design Basis outlines cost estimation parameters of structures and facilities for the Project. The ancillary facilities have been designed using pre-engineered and modular construction where possible to minimize cost and site construction. Local climate and site conditions have been considered in the preliminary design of the buildings. More complex buildings with multiple interior platforms are more suited to conventional engineered post and beam design.

18.9.1.1 Pre-engineered Buildings

These buildings will be constructed with a structural steel frame, steel girts and purlins, and intermediate structural members. Walls will be constructed of metal wall panels and the roof will be metal standing seam roof system. The insulation will be blanket insulation with vapor barrier; insulation will be protected by 10 ft. high protective metal panels on each floor. The envelope package comes complete with doors and all other envelope-related items. Capital cost pricing is determined by using recent budget pricing for buildings of similar size and function.

The cold storage warehouse is also a pre-engineered building and will be a galvanized steel support structure with an un-insulated fabric envelope. Capital cost pricing is determined by using recent budget pricing for buildings of similar size and function.

18.9.1.2 Modular Buildings

Each building will be outfitted with heating, ventilation, and air conditioning (HVAC), electrical, piping, fire detection, and suppression systems ready to be connected to the site utilities. The modules will be constructed of wood framing with insulated metal clad walls and a seamless membrane roofing system on plywood substrate, suitable for local climate, conditions, and codes. Once the modules are in place and connected together, the complex will be weather-tight.

18.9.1.3 Engineered Post and Beam Buildings

These buildings will be constructed with a structural steel frame, steel girts and purlins, and intermediate structural members. Walls will be constructed of metal wall panels and the roof will be a metal standing seam roof system, preferably to match wall and roof assemblies for the pre-engineered buildings. The insulation will be blanket insulation with vapor barrier; insulation will be protected by 10 ft. high protective metal panels on each floor. Internal platforms can be supported by the exterior columns with this type of construction.

18.9.1.4 Internal Architectural Items

Many of the pre-engineered buildings and buildings that are constructed with post and beam design will have certain amounts of architectural materials within the structure, and these items were quantified and priced separately on a building-by-building basis.

18.9.2 Building Descriptions

18.9.2.1 Primary Crushing Building

The primary crushing structure will be of concrete construction with multiple levels housing the gyratory (primary) crusher, the primary apron feeder, the primary discharge conveyor, rock breaker, and overhead bridge crane.

The access ramp structure will be earth-retaining on two sides of the primary crushing building. ROM mill feed will be discharged into the dump pocket from two sides at the top level. Interior steel platforms and roof will be provided to support equipment for ongoing operation and maintenance needs. The control room adjacent to the dump pocket will be a modular prefabricated unit.

18.9.2.2 Mine Site Crushed Mill Feed Stockpile

The mine site crushed mill feed stockpile will be fed by the primary crusher discharge conveyor. The concrete reclaim tunnel, complete with three apron feeders discharging onto a transfer conveyor, will transfer the coarse mill feed onto an overland conveyor that will feed the crushed stockpile.

18.9.2.3 Crushed Mill Feed Stockpile

The crushed mill feed stockpile will be fed by the overland conveyor. The two concrete reclaim tunnels, complete with eight apron feeders discharging onto two SAG mills feed conveyors, will feed the SAG mills.

18.9.2.4 Pebble Crushing Building

The pebble crushing building will be a pre-engineered steel structure with insulated steel roof and walls. The building foundation will comprise concrete spread footings, grade walls along the building perimeters, and slab-on-grade. The building will be serviced with 35 t and 5 t overhead cranes.

The pebble crusher feed bin will be fed by two pebble crusher feed conveyors. The three pebble crushers will be fed by three belt feeders from the feed bin. The three pebble crushers will discharge the fine mill feed onto two SAG mill feed conveyors.

18.9.2.5 Mill Building

The mill building will be an engineered post and beam steel structure with insulated steel roof and walls. The building foundation will comprise of concrete spread footings, grade walls along the building perimeters, and slab-on-grade floor. The floor surfaces will have localized areas sloped toward sumps for cleanup operations.

Interior steel platforms on multiple levels will be provided for ongoing operation and maintenance needs. The building will house such processes as milling, media separation, refining, flotation, concentrate dewatering, process water tank and fresh/fire water tank, and some offices for mill staff. The building will be serviced by a number of bridge cranes of different capacities suited for the specific functions and areas.

18.9.2.6 Conveying

Conveyors will be vendor-supplied systems and will include all structural support frames, trusses, bents, and take-up structures. Overland conveyors will be supported on concrete precast strip panels/sleepers spaced at regular intervals. Elevated conveyor systems will be supported on vendor-supplied steel trusses spanning between steel bents on concrete spread footings.

18.9.3 Reagent Storage and Handling

The reagent storage and handling building will be a pre-engineered steel structure with insulated steel roof and walls. The building foundation will comprise of concrete spread footings, grade walls along the building perimeters, and slab-on-grade. The building will be serviced with 35 t and 5 t overhead cranes. The internal floors and platforms will be structural steel, and elevated floors, where required, will be concrete slab. The majority of equipment will be small tanks, pumps, and mixing equipment.

18.9.4 Warehouse / Truck Shop / Mine Dry

The warehouse / truck shop / mine dry building will be a pre-engineered building to accommodate maintenance and repair, warehouse storage, offices, dry areas, and general storage facilities. This building will comprise of:

- eight maintenance bays
- two light vehicle repair bays
- a machine shop
- a welding shop
- an electrical shop
- an 840 m^2 storage warehouse with an upper level mezzanine area
- a dry area including 240 lockers
- first aid
- emergency vehicle storage

18.9.5 Cold Storage Warehouse

The 600 m^2 cold storage warehouse will be located adjacent to the warehouse / truck shop / mine dry building. It will be a pre-engineered building with single-skin metal cladding, and metal roll-up doors facing north and south.

18.9.6 Administration Building

The 2,400 m^2 administration building will be a slab on-grade modular two-storey structure to accommodate administrative, engineering, mining, geology, and environmental personnel.

18.9.7 Fuel Storage

The main diesel fuel storage will be located at the Mine Site with the capacity of two 500,000 L tanks housed in an HDPE-lined tank farm with berms. Two 8 m (diameter) by 8 m (high) diesel tanks will be provided for a mining haulage truck fleet consumption of 25,000,000 L/a, including a 10-day storage capacity allowance for additional mobile mine support and auxiliary equipment. Fuel dispensing facilities, including light vehicle as well as fast fill facilities for mining equipment, will be provided. It is assumed that diesel will fuel all vehicles at site.

18.9.8 On Site Explosive Storage

The explosive storage facility will be a 120 m by 120 m compound located south of the open pit, approximately 1,000 m from the nearest road or mine building. The compound will include one ammonium nitrate storage structure.

18.9.9 Emulsion Plant

The emulsion plant will be a 100 m by 80 m compound located south of the open pit, approximately 1,000 m from the nearest road or mine building. The compound will include the following items:

- emulsion plant
- truck storage and shop
- emulsion storage
- fuel storage
- office trailer
- parts storage

18.9.10 Detonator and Explosive Storage Magazines

Three 9 m^2 Type 4 magazines will be fabricated from a 6 mm metal plate. The walls will contain at least 7.6 cm of bullet-resistant material and the roof will be a 4.7 mm (or heavier) metal plate. The magazines will be mounted on large metal I-beam skids giving a minimum ground clearance of 10 cm or more for structural rigidity and portability. Fabrication will be as per current storage standards for industrial explosives.

18.9.11 Assay Laboratory

The assay laboratory will be a 600 m^2 single-storey modular structure located adjacent to the mill building. The laboratory will be equipped to perform daily atomic adsorption spectrophotometry and fire assaying analyses on mine blasthole and process samples. A metallurgical investigation section to support operations and test future mill feed will also be housed within the laboratory.

18.9.12 On-site Concentrate Storage Facility

The on-site concentrate storage facility will have an approximate seven-day storage capacity for approximately 8,000 t of concentrate. Filtered concentrate will be loaded by front-end loaders into 40 t capacity, side-dump B-train trucks. The trucks will move through a wheel-wash spray to remove road ice and dirt before entering the loading bay. Inside the loading bay, the vehicles will move through a spray bar to remove concentrate spillage, which will be periodically collected from the settling sump and pumped to the concentrate thickener.

18.9.13 Operation and Construction Camp Accommodations

On-site camp accommodations for construction personnel will be provided during the construction phase. The current design includes:

- 1,200 single-occupancy rooms (construction and operation)
- 700 single-occupancy rooms (construction only)
- an existing 80-person exploration camp that can be used for site pioneering and during construction
- a dormitory building
- a kitchen
- a recreation complex
- an administration complex
- a sewage treatment facility
- potable water facilities

After construction is completed, the camp will be updated for the operations staff.

18.9.14 Operations Personnel Transport

All construction and operation personnel will be flown in on a regular basis and transported to site. No private vehicles will be allowed on site.

18.10 Power Supply and Distribution

Knight Piésold completed the detailed engineering design for the 95 km 287 kV transmission line used in the 2013 FS. This design was updated for this study and is summarized in the following sections.

18.10.1 Power Supply

The site will obtain its power supply from BC Hydro by constructing a private 95 km, 287 kV transmission line from the mine site to the existing BC Hydro Bob Quinn Substation. The Bob Quinn substation is located 97 km south of Iskut, British Columbia, and 335 km north of Terrace, British Columbia. It is connected to the BC Hydro grid via a 287 kV line to Terrace, British Columbia. The Bob Quinn substation forms part of the NTL.

The electrical load at the mine site will average 180 MW, with a power factor of approximately 97%. The transmission line loss is expected to be approximately 1.25%.

The proposed 287 kV transmission line runs in an approximate west orientation and mostly follows the Galore Creek road and Mess Creek road. The design of the line route is more direct than the access road, which reduces line length. Trees will be removed where required to create a right-of-way at least 63 m wide. The 287 kV transmission line will utilize two-pole wooden H-frame structures for the majority of its length. Three-pole wooden structures will be used for angled installations and dead-end structures within the line. Transmission structures will be located to ensure that all required design clearances will be met, including open ground, river, highway, resource road crossings, pipelines, and mine site structures and facilities. The line and its structures will be located to minimize exposure to 200-year return period flood elevations and terrain hazards. Environmental considerations, such as riparian vegetation buffers and riparian setbacks, archaeological features, forest recreational areas, and wet ground were considered in the route selection.

18.10.2 Power Distribution

18.10.2.1 Main Substation

The transmission line will terminate at a substation located on site. The substation will contain the following:

- A main incoming disconnect switch and main circuit breaker (SF6)
- Four circuit breakers (SF6) with line side disconnects
- Four main step down transformers each rated 60/80 MVA, ONAN/ONAF, 287 kV to 35.0 kV, nominal 10% impedance, high voltage and low voltage delta with zigzag transformers with resistance grounding

The transformers will be equipped with on-load tap changers on the 287 kV side. The main transformers will be sized to allow full plant operation, even if one transformer is down for maintenance or has failed.

All four main transformers will be equipped with station class surge arrestors. All current transformers will be bushing type on the circuit breakers or the transformers. One set of potential transformers will be installed on the incoming bus feeding the three main transformers.

The main transformers will feed a line-up of 38 kV metal clad or gas insulated switchgear, located inside a prefabricated building, within the substation area. The building will also contain protection and control for the 287 kV equipment, HVAC equipment, station service, battery charger, and batteries.

18.10.2.2 Site Power Distribution

The 38 kV switchgear will contain 33 feeder breakers (including 5 spares) that will distribute power throughout the site. Six of the 35 kV feeders will directly connect to the SAG/Ball mill motors via cycloconverters. The remaining plant loads will be served at the 4.16 kV or 600 V level. Resistance grounding will be used at all distribution voltage levels.

Transformers used to convert from 35 kV to 4.16 kV will be outdoor type, oil filled, and located adjacent to the electrical rooms on the side of the concentrator. Tie breakers on the 4.16 kV level will be used to provide redundancy. Transformers used to convert from 35 kV to 600 V will be indoor, dry type, and located within the electrical rooms. Generally, these will be included as part of double ended unit substations.

18.10.2.3 Emergency Power Distribution

There will be two emergency generators at the plant site, retained and refurbished after construction use. The emergency generators will be distributed around the plant to pick up critical loads via transfer switches. The generators will be located near the critical loads.

18.10.2.4 Remote Loads

Remote loads will be served at 35 kV via overhead power line. The first line will service the explosives, water, truck, fuel, and the TSF area. The second line will service the pit shovels and drills.

The line servicing the open pit will feed two electric drills and two electric shovels. The drills will be a fairly constant load of 400 kW. The shovels will be a nominal 40 second cyclic load varying between 4 MW load and 2.5 MW regeneration. The shovels and drills will be powered at 7.2 kV.

Portable substations served by overhead line will be located at the top edge of the pit. The portable substations will contain the step-down transformers and main distribution switchgear. From this point, an on-ground cable will extend down the longwall to a portable building containing a splitter. The splitter will serve up to two loads in the immediate area in the pit.

An overhead line will extend around the TSF embankment to provide power to the seepage pump stations and lights.

18.11 Communications

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

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

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

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

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

The requirements for communications, particularly satellite bandwidth, are a function of the voice and data requirements of the active participants in the Project. The expectation is that the need for satellite bandwidth will build to a peak during the plant construction phase, and then taper off slightly as the initial construction crew yields to plant operations.

Technologies and services to be provided include the following:

- Construction Phase:
  - Local VoIP wireless network
  - Satellite link for voice, data and video services
  - PC Local/Wide Area Network (LAN/WAN)
  - Trunked mobile radio system
  - Internet service
  - Private telephone system for voice and fax
  - Video conferencing to minimize travel during design and construction

  - Ground-go-air communications system (VHF radio)
  - Cable television on independent satellite system
- Operations Phase (includes selected services above):
  - Process monitoring and control for efficient operation and maintenance
  - Fibre optic cabling for plant wide communications
  - Security access control

19.0 MARKET STUDIES AND CONTRACTS

19.1 Copper Concentrate Sales

The key aspects of Schaft Creek's copper concentrate production are highlighted below:

- The annual copper concentrate production will be approximately 385,000 dmt/a (LOM average). Peak concentrate production will be approximately 440,000 dmt/a.
- The copper concentrate grade is estimated to be 28%.
- The copper concentrate contains significant by-product gold and silver credits.
- The copper concentrate is not expected to have any notable levels of arsenic, zinc, lead, or any other impurity elements that would result in penalty charges or marketability issues.

From a logistics perspective, given the location of Schaft Creek and the Port of Stewart, copper concentrates produced from this location will be competitive selling into the Asian concentrate market.

Schaft Creek's copper concentrate is expected to be of good quality in the market. The good copper grade with gold and silver values and low impurity levels will likely allow Schaft Creek to establish its brand as premium; currently, no major issues are envisaged in regards to placing this product in the marketplace.

19.1.1 Forecast Market Distribution

Due to its geographic location, Asian smelters are the natural home for Schaft Creek's copper concentrates. Asian smelters represent over 80% of the global custom concentrate market and Schaft Creek's copper concentrate can be delivered, directly or indirectly, to the largest and most well-established smelters in China, Japan, Korea, and India.

Schaft Creek's copper concentrate sales would likely be split between smelter customers and international concentrate trading companies.

19.1.2 Copper Concentrate Contract Terms

There has been no study conducted on smelting terms. The smelter terms used for the analysis are generic based on the in-house data as below. The treatment charge, copper refining charge, and price participation, if any, are negotiated periodically.

Table 16-1 summarizes the anticipated smelting terms and conditions for Schaft Creek's copper concentrates.

Table 19-1: Typical Smelter Contract Terms

Parameter	Metal Content	Term
Treatment Charge	N/A	USD 90/t concentrate
Copper Payable	<22% Cu	Grade less an absolute deduction 1.1
	≥22% and <32% Cu	Net payable 96.50%
	≥32% and <38% Cu	Net payable 96.65%
	≥38% Cu	Net payable 96.75%
Copper Refining Charge	N/A	USD 0.09/lb payable copper
Price Participation	N/A	None
Gold Payable	≤ 1 g/t Au	0%
	> 1 g/t and ≤ 3 g/t Au	90%
	> 3 g/t and ≤ 5 g/t Au	92%
	> 5 g/t and ≤ 8 g/t Au	95%
	> 8 g/t and ≤ 10 g/t Au	96%
	> 10 g/t and ≤ 15 g/t Au	97%
	> 15 g/t and ≤ 50 g/t Au	97.5%
	> 50 g/t Au	98%
Gold Refining Charge	N/A	USD 5.00/troy oz payable gold
Silver Payable	≤ 30 g/t Ag	0%
	> 30 g/t Ag	90%
Ag Refining Charge	N/A	USD 0.50/troy oz payable silver

19.1.3 Copper Concentrate Freight

Copper concentrate produced and stored on site will be loaded into concentrate freight trucks by a front-end loader, then trucked from the site to the deep-water port in Stewart, British Columbia. At the Stewart Port, the bulk copper concentrate will be unloaded from trucks and conveyed to the storage facility prior to being shipped to Asia by ocean vessels. The estimated copper concentrate transportation cost is USD 76.67 per tonne of wet concentrate.

19.1.4 Molybdenum Concentrate Sales

The annual molybdenum concentrate production will be approximately 9,780 dmt/a (LOM average). Peak concentrate production will be approximately 16,200 dmt/a. The molybdenum grade is estimated to be 50%. Molybdenum payable is 99.0% of the molybdenum concentrate grade (per dmt). The treatment and refining costs net of payables is assumed to be USD$2.50/dmt concentrate and USD$1.52/lb molybdenum, respectively.

The molybdenum concentrate is anticipated be sold directly to processors in North America, South America, and Europe, or to international trading companies. The estimated molybdenum concentrate transportation cost is USD$154.05 per t of wet concentrate.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The following presents the environmental, social, and permitting components of the Schaft Creek Project as defined within this report. The proposed project design was reviewed in the context of current legislation within British Columbia and Canada. Based on the current understanding of the Project, regulatory approval requirements and socio-cultural understanding/interpretation, the Schaft Creek Project as currently defined does not present significant risk to successful completion of the regulatory approval process.

20.1 Environmental Setting and Studies

20.1.1 Overview

A number of project-specific baseline studies were completed for the Schaft Creek Project between 2006 and 2012, in support for a potential Environmental Assessment (EA) Application. This includes baseline collection of dust, noise, meteorological, groundwater, and surface water monitoring studies; aquatic and fisheries studies; collection of physical, chemical, and biological marine data; sediment quality, wetland, flora, and fauna surveys; species at risk surveys; site metal analysis; archaeological assessments; land use reviews; and cultural and socio-economic studies.

Collection of environmental data has continued as part of a long-term data collection effort for hydrology, climate, and hydrogeology intermittently since 2013.

The following sections provide a brief summary of the environmental setting of biophysical aspects of the Project area, including a summary of baseline study results of the following key topics:

- climate and atmospheric conditions
- topography and glacial history
- geology, surficial geology, and soils
- geohazards
- ML and ARD
- hydrology and watershed characterizations
- surface water
- groundwater
- fisheries and aquatic habitat
- terrestrial ecosystems
- wildlife and wildlife habitat

A summary of the baseline social, economic, and cultural studies and human health studies are provided in Section 20.2 and Section 20.3, respectively.

Additional details, along with reference to source documents of baseline studies and programs, are presented in Schaft Creek Project Summary of Environmental Studies and Permitting Report (Greenwood, 2019).

20.1.2 Climate and Atmospheric Conditions

The Project is located on the eastern edge of the Boundary Range of the Coast Mountains in northwestern British Columbia. Climate in the Schaft Creek area is characterized as transitional, between coastal and interior conditions. The regional hydroclimate of northwestern British Columbia is dominated by weather systems generated over the Pacific Ocean and is strongly influenced by orographic effects caused by mountainous topography. On the coast, annual precipitation is very high, often exceeding 3,000 mm in the Coast Mountains; temperatures are relatively mild due to the moderating effect of the Pacific Ocean. Climate of the interior is continental, characterized by warm, short summers, cold winters, and an annual precipitation typically between 400 mm and 800 mm.

The climate field program was initiated in 2005, discontinued in 2009, and re-initiated in October 2013 with site visits conducted to present. Between October 2005 and September 2009, four climate stations and six snow course or snow depth observation stations were operated. However, several of these sites were challenging to maintain and data was unreliable. Since October 2013, the climate program has included data collection at two climate stations, with a focus on critical sites and locations where good quality data can be collected. Although more recent data has been collected, the following summarizes results from the last comprehensive review and reporting of climate baseline information (Knight Piésold, 2010).

Based on meteorological data collected from October 2005 to September 2009 from the Schaft Creek Saddle climate station (situated at elevation 977 m), mean annual temperature is 1.2°C, with minimum and maximum monthly temperatures of -8.2°C and 12.3°C occurring in January and August, respectively. Mean annual wind speed is approximately 2.5 m/s, with gusts reaching 6 m/s, with the dominant measured wind direction from the south. Mean annual relative humidity is approximately 72% and mean annual potential evapotranspiration for the Project area is estimated to be 433 mm. MAP for the Project area is about 850 mm, with 30% falling as rain and 70% as snow.

Baseline air quality in the area is considered good, since measured parameters fell within applicable project objectives and guidelines. This includes dustfall results recorded below the 1979 British Columbia Pollution Control Objectives for the mining, smelting, and related industries of British Columbia, and sulphate and nitrate concentrations below critical load estimates for similar regions in Canada.

When the climate program is reinstated with a higher level-of-effort, all instruments should be calibrated or replaced, per the manufacturer's recommendation. As described in the Water and Air Baseline Monitoring Guidance Document for Mine Proponents and Operators (British Columbia Ministry of Environment, 2016), a winter snow survey field program may be required to collect winter snowpack data to reduce uncertainty in the total annual precipitation. Winter precipitation data are challenging to collect, and snow course surveys provide valuable snow density information for validating and/or correcting the automated record to snow water equivalent (SWE). This program should be commenced at least two years prior to EA submissions and can be paired with winter hydrology data collection activities.

20.1.3 Topography and Glacial History

The Coast Mountains are characterized by steep, rugged topography, high relief, and extensive alpine glaciers and snowfields, with deeply entrenched, broad U-shaped valleys that trend north-south. Ground elevations typically range from around 700 m to 900 m on the valley floors, to an excess of 2,500 m at the mountain peaks. Mount Edziza Provincial Park, an area of recent extensive lava flows, is located approximately 5 km to the east of the Project.

Topography in the area varies widely, with slopes that range from level, to very gently sloping, to steeply sloping. Immediately north of the deposit, topography is dominated by Mount LaCasse, which rises to an elevation of more than 2,000 m. The east-facing slope drops into the Mess Creek Valley. Topography within the valley floor is very subdued and largely covered by glaciofluvial gravels with very scarce bedrock exposures in the lower elevations of the valley floor.

The topography within the Project area reflects burial beneath the Cordilleran ice sheet during the Late Wisconsinan Fraser Glaciation (ca. 25 ka – 10 ka), followed by Holocene alpine glaciation and erosion due to fluvial and landslide processes. In earlier to middle stages of glaciation, ice extent and thickness increased to form the continental scale, Cordilleran Ice Sheet, with south-westward flow towards the Pacific Ocean. During the later stages of glaciation, ice flow became confined to major valleys and fjords, and retreated primarily by frontal retreat and downwasting. The broad U-shape of Mess Creek reflects preferential glacial scour of the weaker, fractured rocks along the Mess Creek fault zone.

During Holocene time, several episodes of glacial advance and retreat have occurred since about 7,700 calendar years before present, most recently during the “Little Ice Age” from the 12th to 19th centuries. In most areas, the largest Holocene glacial advance culminated in the early to mid-1800s. Since then, many glaciated areas have decreased by over 30% with over 200 m of associated loss in ice thickness in some basins (refer to the options study by BGC (2008b) for detailed reference citations).

20.1.4 Geology, Surficial Geology, and Soils

The Project area is located near the border of the Tahltan Highlands in the Boundary Ranges. These ranges are comprised of steep granite mountains and the highlands form transitional belts between these granite mountains and the 5,000 ft. Yukon and Stikine Plateaus. The Coast Mountains constitute a large anticline of sedimentary and volcanic rocks with a central composite base of batholithic intrusions. Volcanic, sedimentary, and metamorphosed sedimentary rocks, including shales, siltstones, sandstones, greywacke, conglomerates, and limestone, along with plutonic rocks, occur in the region surrounding the project site.

The Schaft Creek deposit borders the Hickman Batholith to the west and the volcanic rocks of the Mess Lake facies to the east with the valley floor exposing the Stuhini group volcanics. The deposit is hosted by north striking, steep, easterly dipping volcanic rocks, mafic flows, subvolcanic intrusions, and epiclastics of the Stuhini Group, with only 10% of the mineralization in felsic dikes and quartz feldspar porphyry.

The complexity of the surficial geology in the region is indicated by the range of surficial materials. These include colluvium, glacial till (morainal), glaciofuvial, fluvial, and volcanic deposits. They occur in varying thickness, depending on the topography in which they were deposited and the process by which they developed. Bedrock outcrops are commonly found on upper crest meso slope positions within the high altitude areas.

The lowlands and valleys within the Project area consist of subdued relief, with forested, glacially rounded and elongate outcrops, and thick intervening Pleistocene and recent glacial, glacial fluvial, and fluvial deposits. The region has been the site of more than one glaciation and this process has influenced the present topography. Since the last glacial period, the land surface has continued to be modified by the action of gravity, wind, water, and ice resulting in large areas of colluvial and fluvial deposits along the creeks. Wetlands and soils with organic caps have developed in depressional and seepage areas.

Soils in the mine site area are primarily developed on colluvial and morainal materials, generally well drained and predominantly classified as Podzols and Brunisols with small areas of Regosols, Organics, and Gleysols.

Podzolization is the dominate soil forming process throughout the study area due to the region's geology and cold, moist climate in the lower- to mid-elevation range. These soils are found in the Interior Cedar-Hemlock, Engelmann Spruce, Spruce-Willow-Birch, and Boreal White and Black Spruce biogeoclimatic zones. This process involves a downward translocation of iron, aluminum, and organic matter. Orthic Humo-Ferric Podzols dominate and Orthic Dystric Brunisols, Eluviated Dystric Brunisols, and Orthic Sombric Brunisols occur also indicating less developed, less weathered soils with the latter having higher organic matter in the surface layer. Orthic Melanic Brunisols occur on basic parent materials and on younger fluvial soils. Soils classified as Orthic Gleysols have developed in wetter areas and many of these have a thick organic capping. In these, a permanent water table occurs within a metre of the surface.

In the high elevation alpine and subalpine areas, the high precipitation results in high rates of leaching and acidic soils. Cold temperatures in these areas slows the rate of plant decomposition; therefore, the rate of nutrient cycling is slow. The A horizons of soils in this area have a high proportion of undecomposed plant material causing them to have a dark colour and spongy texture. Alpine soils are shallow and are thus fragile to disturbances. Organic soils occur in depressional areas, generally in valley bottoms and are associated with wetlands (bogs, swamps, and fens). These soils' organic matter is moderately to well decomposed (Mesisols to Humisols). Organic soils that have developed in wetter areas generally have poor structural stability and generally a water table at or near the surface.

20.1.5 Geohazards

An overview assessment of geohazards was conducted in 2008 for the Mess Creek Access route and for part of the Tahltan Highland route (BGC, 2008b). Geohazards with the potential to impact the Mess Creek access route include debris flows, debris floods, rock fall, snow avalanches, and flooding. Debris flow hazard was identified at channel crossings, including locations where the road crosses a major debris fan. Rock fall hazard exists where the road traverses below steep rock slopes; however, exposure to natural rock fall hazard is uncommon along the access route and of lower frequency than would likely occur on rock cuts. Snow avalanche hazard exists where the access road crosses avalanche paths below the treeline in the lower part of the runout zone. Flood hazard exists at creek crossings and where the access route crosses the Mess Creek floodplain.

A geomorphic channel and erosion hazard assessment was completed during baseline studies in 2008 with the intention to present results that can be used to locate the most stable portions of the river and floodplain with respect to expected patterns of future river erosion (Fluvial Systems Research Inc. 2010).

An overview assessment of landslide and snow avalanche geohazards at three tailings options was also conducted in 2008 (BGC, 2008a), including the selected location of the TSF detailed in Section 18.0 of this report. Geohazards primarily include runout zones of snow avalanches and debris flows around the TSF footprint perimeter. Gentle terrain in the middle of the tailings footprint is considered to have low levels of geohazards.

An approximate 2,000 m wide linear bedrock feature exists on the east side of the valley above the TSF. Based on the position of the feature with respect to adjacent slopes, it is interpreted to have sagged several tens of metres relative to slopes above. No signs of recent movement are visible on the photos (BGC, 2008a), and it is possible that the feature is very old (on the order of thousands of years and possibly associated with glacial debuttressing). Implications of rapid failure would likely include wave triggering and could be factored into tailings dam design. However, rapid failure of the block is considered very unlikely (less than 1/1,000).

20.1.6 Metal Leaching and Acid Rock Drainage

An important part of environmental studies for the Project was the prediction of ARD, ML, and inorganic water chemistry from various major mine site components. These included the pit, waste rock storage piles, tailings, overburden, and the access road. From 2007 through 2012, numerous ML-ARD reports were developed, including the following:

- Schaft Creek Project – ML-ARD Geochemical Shake-Flask Testing of Overburden in the Proposed Pit Area, August 2010 (Minesite Drainage Assessment Group, 2010c).
- Schaft Creek Project – ML-ARD Report on 3D Modelling of ABA Data, May 2010 (Minesite Drainage Assessment Group, 2010b).
- Schaft Creek Project – ML-ARD Report on Acid-Base Accounting and Total Solid-Phase Elements for Rock, April 2010 (Minesite Drainage Assessment Group, 2010a).
- Morin, K.A., and N.M. Hutt. 2007. Schaft Creek Project – Prediction of Metal Leaching and Acid Rock Drainage, Phase 1. Report for Rescan Environmental Services.
- Morin, K.A., and N.M. Hutt. 2008. Schaft Creek Project – Prediction of Metal Leaching and Acid Rock Drainage, Phase 2. Report for Rescan Environmental Services.
- Morin, K.A., and N.M. Hutt. 2009. Schaft Creek Project – Prediction of Metal Leaching and Acid Rock Drainage for Overburden in the Proposed Pit Area. Report for Copper Fox Metals Inc.
- Morin, K.A., and N.M. Hutt. 2010a. Schaft Creek Project – ML-ARD Assessment of Surficial Samples from the Proposed Access Road. Report for Copper Fox Metals Inc.

- Morin, K.A., and N.M. Hutt. 2010b. Schaft Creek Project – Mineralogical Studies and Geochemical Kinetic Tests for Metal Leaching and Acid Rock Drainage from Rock and Tailings. Report for Copper Fox Metals Inc.
- Morin, K.A., and N.M. Hutt. 2010c. Schaft Creek Project – Acid-Base Accounting and Solid-Phase Total-Element Contents for Rock. Report for Copper Fox Metals Inc.

The summary below focuses on the potential of rock and tailings to release ML-ARD and affect drainage chemistry. It is not clear at this time whether the earlier predictions for overburden and the access road remain valid, further test work may be required.

20.1.6.1 ML-ARD Predictions for Rock

Rock Sample Collection and Analysis for ML-ARD

In total, 926 samples of rock from Schaft Creek core were collected for ML-ARD testing. These samples reflected ranges of rock units (lithology), three-dimensional locations within the deposit, and geochemical concentrations. Note that a full three-dimensional geochemical model was not developed for the current pit shell. All samples were submitted for single-step geochemical static tests, primarily ABA and solid-phase total-elemental contents. Selected samples were also submitted for repetitive geochemical kinetic testing under controlled laboratory conditions and uncontrolled on-site conditions.

Mineralogy of the ML-ARD Rock Samples

The ML-ARD mineralogical studies showed plagioclase (typically AN30-70) was often the dominant mineral, with quartz, orthoclase, sericite/muscovite, and chlorite/clinochlore often at significant levels. Other silicate minerals reaching significant levels in some samples included illite, epidote/clinozoisite, biotite, pyroxene/augite, potassium (K) feldspar, and amphibole/hornblende/actinolite. From a ML-ARD perspective, some samples contained percentage levels of pyrite, chalcopyrite, and gypsum. Moreover, some contained even higher levels, above 10%, of calcite, dolomite, and ankerite.

Based on data available in MDAG 2012, the mineralogy of sulphide minerals, including pyrite and chalcopyrite, could only be confirmed at Mineral Resource grade levels above roughly 0.5%S, and sulphide mineralogy could not be confirmed in waste rock. Thus, all measured total sulphur in waste rock was assumed to be potentially acid-generating pyrite.

Mineralogy of rock was generally similar to tailings mineralogy. The major exception was no reported K-feldspar in the tailings (MDAG, 2012).

ABA of ML-ARD Rock Samples

Paste pH in the 926 rock samples for Schaft Creek ranged from 7.4 to 9.9 (MDAG, 2012). Thus, no Schaft Creek rock was acidic at the time of analysis, although most samples had been exposed to weathering and oxidation for years to decades.

Total sulphur in the 926 Schaft Creek samples of cored rock ranged from <0.01 (detection limit) to 13.5%S, with a mean of 0.46%S and a median of 0.22%S (MDAG, 2012). Statistically, sulphide represented 79% of total sulphur on average, with a median of 84%. Thus, the two parameters were typically interchangeable, but not identical. To avoid the uncertainties and inaccuracies in sulphur species and calculations involving sulphur species at low levels, conservative ML-ARD estimates of acid potential for Schaft Creek rock used total sulphur and associated Total-Sulphur-Based Acid Potentials (TAP), assuming all total sulphur occurred as pyrite.

Sobek (U.S. EPA 600) NP ranged from 4 kg $CaCO_3$ equivalent/tonne to a maximum of 243 kg/t, with mean and median values of 75 kg/t and 68 kg/t. Some portion of this measured NP is typically unavailable for neutralization and should be subtracted from measured NP to obtain Effective NP. However, the lack of acidic paste pH and the lack of acidic kinetic tests (see below) meant that the worldwide general value of 10 kg/t was adopted for Schaft Creek rock.

Correlations with inorganic-carbon-based NP and Effective NP suggested most of the carbonate was calcite and dolomite, consistent with mineralogical studies. Also, correlations of NP with LOI represented a surrogate measurement of NP at the Project.

The acid-generating and acid-neutralizing capacities of the 926 rock samples were combined as Adjusted Total-Sulphur-Based Net Potential Ratios (Adj TNPR), including the subtraction of 10 kg/t of unavailable NP from measured NP. Adj TNPR ranged from 0.001 (the default value where NP ≤ 10 kg/t and thus the net-acid-generating sample has no Available NP) to 554 (net neutralizing). The arithmetic mean and median were 32.5 and 8.9, respectively, indicating most samples were net neutralizing.

Of the 926 rock samples, 137 (14.8%) had Adj TNPR values below 2.0 (net acid generating), and 85.2% was net neutralizing. A sensitivity analysis (1) replacing total sulphur with sulphide plus unaccounted-for sulphur and (2) assuming all measured NP was available (Unavailable NP = 0 kg/t) had a minor effect on the percentages, with 85% to 92% remaining net neutralizing. By rock unit, Breccia had the highest percentage of net-acid-generating samples. However, these percentages were "unweighted" in that they are based only on sample numbers and did not necessarily reflect tonnages and volumes within the deposit.

Based on simple correlations, a rock sample at Schaft Creek with more than 2% total sulphur would likely be net acid generating no matter the NP level. Conversely, one with less than 0.2%S would likely be net neutralizing. Also, rock samples with an NP above 140 kg/t were consistently net neutralizing.

Combined information from ABA and kinetic testing (see below) provided estimates of "lag times" until the small percentage of net-acid-generating rock samples (~15% of all samples) became acidic. A small number (roughly 13%) of the net-acid-generating samples would become acidic within 13 years after initial exposure. Also, half (median value) would be acidic after 26 years of active sulphide oxidation, and all would be acidic 70 years after initial exposure. This may explain why acidic pH levels were not readily detected in rock samples from weathered core at the Project.

Three-Dimensional Distribution of ABA Results in the 2012 Pit

As explained above, percentages of net-acid-generating rock were "unweighted" in that they were simply based on sample numbers. To correct this, the volume and three-dimensional distribution of the total net-acid-generating rock were estimated, within the 2012 proposed ultimate pit shell and on its walls, in a conservative way.

This three-dimensional modelling showed about 6% of the pit rock (both Mineral Resource and waste combined) would be net acid generating. The great majority (~94% of Mineral Resource and waste combined) would be net neutralizing. However, preliminary block modelling by Moose Mountain in 2013 suggested about 12% of pit rock (both Mineral Resource and waste combined) would be net acid generating, but this was not finalized or confirmed.

Most net-acid-generating rock would be mined only in later years of operation, providing years to prepare for its excavation and management. Also, a significant portion of the net-acid-generating rock would be Mineral Resource grade, where it would be milled into net-neutralizing tailings.

In 2012, options for management of net-acid-generating waste rock included sending the low percentage of net-acid-generating waste rock to the mill, submerging the net-acid-generating waste rock in the tailings impoundment, disposing of net-acid-generating rock separately for separate management and mitigation, and mixing the rock together within the waste-rock pile.

Most net-acid-generating areas on the 2012 pit walls (~125,000 m^2) lay deep in the pit. As a result, most net-acid-generating areas on the pit walls were expected to be submerged after closure and pit-lake formation. The exception was an area about 100 m by 100 m higher on the northwest pit wall, which could require alternative management or mitigation, such as pit-wall pushback, if it adversely affects pit-water chemistry.

Total-Element Analyses of ML-ARD Rock Samples

The dominant solid-phase elements in the 926 rock samples were mostly silicon and aluminum. This reflected the dominance of aluminosilicate minerals in Schaft Creek rock.

Compared with average crustal abundances, the samples were frequently elevated in silver, bismuth, copper, molybdenum, sulphur, antimony, selenium, and tungsten. The samples were also occasionally to rarely elevated in arsenic, cadmium, cesium, lead, and zinc. These elevated levels did not automatically mean these elements would leach into water at high concentrations. Their presence could instead indicate low leaching as was generally observed for many elements in the Schaft Creek kinetic tests (see below). The elements that showed some correlation with sulphide, suggesting they were at least partly occurring in/as sulphide minerals, included silver, copper, and selenium.

Laboratory Kinetic Testing of Rock

The laboratory-based kinetic test chosen for the nominal 1 kg samples of rock and tailings was a “standard-Sobek humidity cell”. Nine samples of rock and four samples of tailings were tested in cells for up to 160 weeks (3 years). These samples represented the major rock units and spanned ranges of total sulphur, NP, and other elements. Based on initial solid-phase analyses, two rock samples were predicted eventually to become acidic, after some lag time.

All weekly effluents from the cells remained near neutral pH. Most cells converged on the pH range of 8.0 to 8.2, which is typical of carbonate neutralization. The paste is an initial check on the characteristics of the waste rock and the humidity cells refine the characteristics of the waste rock.

Correlations among kinetic rates and initial solid-phase levels were examined so that predictions of one rate could be made from another rate or from a solid-phase analysis. For example, the rate of sulphide oxidation in rock and tailings could be roughly estimated from the initial sulphur analyses. For the two rock cells initially predicted to become acidic, these equations predicted this would happen after 14 to 19 years of active sulphide oxidation.

The Schaft Creek cell rates were compared to compiled rates in the International Kinetic Database, to determine if they were unusually high or low, or generally typical. For sulphate production, the Schaft Creek rock cells were generally typical of other rates within the pH 8 range, except for two Schaft Creek rock cells that were among the lowest rates in the database.

On-Site Kinetic Testing of Rock

To evaluate upscaling for Schaft Creek rock, on-site kinetic tests were set up using nine plastic barrels, nearly 1 m high and holding up to approximately 400 kg of rock. Up to six rounds of drainage analyses were available from the nine on-site barrel kinetic tests, from October 2008 to September 2011. Over this period, drainage pH remained consistently near neutral, mostly between pH 7.5 and 8.5. Thus, no acidic drainage or ARD occurred from these barrels, even from the barrel expected to eventually release ARD.

The concentrations from the nine on-site barrels were compared to the nine 1-kg laboratory-based humidity cells. Simple upscaling of cell rates indicated maximum concentrations of many elements from the barrels were overpredicted (overestimated) based on the cells. Thus, the barrel concentrations for these elements had apparently reached large-scale equilibrium, and their concentrations could not increase as high as predicted from the smaller scale.

The above was further supported by the U.S. EPA speciation-solubility model, Minteq. Minteq indicated some maximum aqueous concentrations in the on-site kinetic tests (and even some in the small-scale cells) were likely in generic or local equilibrium with various minerals, such as cobalt and zinc. Thus, the maximum measured concentrations in the kinetic tests would likely not increase as scale increased.

Prediction of Full-Scale Drainage Chemistry from Mine Rock

Based on the preceding work, predicted full-scale drainage concentrations from rock at the Project were compiled in Table 20-1. For example, aqueous sulphate is predicted to be 1400 mg/L to 2000 mg/L, reflecting gypsum saturation. At early times in mining and at monitoring locations where significant dilution occurs, the predicted concentrations of Table 20-1 may be unrealistically high.

Table 20-1: Predicted Full-scale Equilibrium Drainage Chemistry for Mined Rock at Schaft Creek

Parameter* (mg/L unless noted)	Range of Full-Scale Predictions for Dissolved Parameters		Parameter* (mg/L unless noted)	Range of Full-Scale Predictions for Dissolved Parameters	
	Minimum	Maximum		Minimum	Maximum
pH (units)	7.73	8.35	Co	0.0080	0.017
Conductivity (µS/cm)	2400	3390	Cu	0.11	0.48
Acidity§	5.3	9.8	Fe§	<0.03	0.34
Alkalinity	158	204	Pb	<0.0025	0.0025
Sulphate	1460	2030	Li	0.04	0.08
Hardness	1350	1850	Mg	24	43
Bromide	<0.5	<0.5	Mn	0.26	0.46
Chloride	26	49	Hg	0.000016	0.000034

table continues…

Parameter* (mg/L unless noted)	Range of Full-Scale Predictions for Dissolved Parameters		Parameter* (mg/L unless noted)	Range of Full-Scale Predictions for Dissolved Parameters	
	Minimum	Maximum		Minimum	Maximum
Fluoride	0.55	1.0	Mo	5.0	7.2
Nitrate†	0.92	1.59	Ni§	0.0057	0.031
Nitrite†	0.039	0.074	P	<0.3	<0.3
Ammonia†	0.097	0.21	K§	13	32
Phosphate (P)	0.041	0.16	Se	0.029	0.14
Al§	<0.025	0.78	Si§	2.3	4.7
Sb§	0.0099	0.20	Ag§	0.000051	0.0030
As	0.0030	0.0098	Na‡	170	350
Ba§	0.066	0.23	Sr	4.3	11
Be	<0.005	<0.005	Tl	<0.001	<0.001
Bi	<0.2	<0.2	Sn§	0.00087	0.0015
B	0.10	0.16	Ti§	0.016	0.020
Cd	<0.0025	<0.006	U‡	0.012	0.074
Ca	494	719	V§	<0.005	0.081
Cr	<0.005	<0.005	Zn	2.0	2.6

*Concentrations of metals and other elements are dissolved (filtered).
†Concentrations of nitrogen species predicted here are not necessarily representative of those that will be derived from blasting residues upon mining.
‡Mineral-solubility calculations could not determine if these elements were limited by or close to equilibrium, so increasing scale may increase their concentrations.
§The maximum value was the smaller-scale humidity-cell maximum, rather than larger-scale barrel data.

In 2013, preliminary work suggested that predicted aqueous concentrations of several potential contaminants, like those in Table 20-1, varied with the solid-phase level of molybdenum (Mo).

In other words, the solid-phase level of molybdenum in a mass of waste rock became the predictor of its aqueous drainage chemistry, rather than the minimum-maximum values of Table 20-1.

20.1.6.2 ML-ARD Predictions for Tailings

ML-ARD Test Work on Tailings

Metallurgical test work was conducted in 2007, 2009, 2011, and reported in 2013. This work produced solid-phase tailings samples for static analyses, like ABA, and for kinetic testing, namely humidity cells. This work also produced aqueous "supernatants" that were analyzed directly for many aqueous parameters and dissolved elements.

Based on levels of total sulphur and NP from ABA, all samples except 2011 cleaner/scavenger tailings were net neutralizing. Thus, no ARD is predicted from Schaft Creek tailings. However, segregated 2011 (not 2009) cleaner/scavenger tailings would release ARD after some lag time of oxidation.

Supernatant waters can change geochemically upon exposure and aging in air. To examine this effect, supernatants including suspended fine-grained tailings solids were aged in air for up to 34 days. The suspended solids allowed solid-liquid interactions during this aging.

Initial pH at Day 0 was typically around pH 10 to 11. The pH then decreased during aging to values around 8, which was consistent with long-term, stabilized pH around 8 for humidity cells containing rock and tailings.

This decreasing pH was accompanied by a general trend of increasing alkalinity, likely attributable to the ingassing of carbon dioxide into the alkaline supernatants. Decreasing pH was also accompanied by, for example, relatively steady to somewhat increasing conductivity and sulphate, and decreasing dissolved aluminum. Thus, long-term aging of the Schaft Creek tailings pond could lead to changes in tailings-pond chemistry.

Prediction of Full-Scale Tailings-Pond Chemistry

To obtain full-scale prediction of maximum values (Table 20-2), the highest concentration was chosen for each element or parameter by comparing the maximum concentration from 2011 supernatant testing (fresh and aged samples), maximum concentration from the tailings humidity cells, and the maximum value from full-scale mined-rock predictions (Table 20-1) predominantly reflecting mineral equilibrium. For many elements and parameters, maximum concentrations from Table 20-1 (rock predictions) were higher than 2011 supernatant testing and thus were used in Table 20-2. The exceptions were total organic carbon, dissolved aluminum, dissolved iron, dissolved magnesium, dissolved phosphorus, dissolved potassium, and dissolved titanium.

Tailings-pond chemistry can be more dynamic and variable than mined-rock drainage for several reasons. For example, variable pH from the mill process can be significant, and the solubilities of several metals and other elements can be strongly dependent on pH. For the rock and aged tailings water, the pH in the test work has been around 8, reflecting equilibrium effects of the abundant carbonate minerals in Schaft Creek rock and tailings. However, during active milling and tailings generation, pH can be maintained at higher levels due to lime addition, such as pH 9 or 10, which is common for such tailings impoundments. In this case, the higher pH can substantially affect the aqueous concentrations of several metals and other elements listed in Table 20-2.

Table 20-2: Predicted Full-scale Equilibrium Drainage Chemistry for Tailings at Schaft Creek

Parameter[*] (mg/L unless noted)	Full-Scale Predictions for Dissolved Parameters in Tailings Drainage[*]	Parameter[*] (mg/L unless noted)	Full-Scale Predictions for Dissolved Parameters in Tailings Drainage[*]
pH (units)	8.35	Fe	0.39
Conductivity (µS/cm)	3390	Pb	0.0025
Acidity	9.8	Li	0.08
Alkalinity	204	Mg	49
Sulphate	2030	Mn	0.46
Hardness	1850	Hg	0.000034
Bromide	0.5	Mo	7.2
Chloride	49	Ni	0.031
Fluoride	1	P	0.79
Nitrate[†]	1.59	K	61
Nitrite[†]	0.074	Se	0.14
Ammonia[†]	0.070	Si	4.7
Phosphate (P)	0.14	Ag	0.003
Al	2.58	Na[‡]	350
Sb[§]	0.2	Sr	11
As	0.0098	Tl	< 0.001
Ba	0.23	Sn	0.0015
Be	< 0.005	Ti	0.03
Bi	< 0.2	U[‡]	0.074
B	0.16	V	0.081
Cd	< 0.006	Zn[§]	2.6
Ca	719	Thiosalts[‖]	< 20
Cr	< 0.005	Total Organic Carbon[#]	16.2
Co	0.017	Chemical Oxygen Demand (COD)[#]	550
Cu[§]	0.48		

*Many of these full-scale concentrations are those of rock (Table RS-1), meaning the full-scale rock maximums were mostly higher than the tailings-test-work maximums. Also, the tailings impoundment may be operated at a different pH due to lime addition, such as pH 9 or 10, which could substantially change the dissolved concentrations from those predicted here. Concentrations of metals are dissolved (filtered).

†Concentrations of nitrogen species predicted here are not necessarily representative of those that will be derived from blasting residues upon mining.

‡Mineral-solubility calculations could not determine if these elements were limited by or close to equilibrium, so increasing scale may increase their concentrations.

§Tailings test work suggests these elements might have substantially lower concentrations in the tailings impoundment, but this cannot be confirmed at this time.

‖Based on analyses of thiosulphate, trithionate, and tetrathionate of the 2007 supernatants.

#Based on analyses of the 2011 supernatant; COD from 2007 supernatants only.

20.1.7 Hydrology and Watershed Characterization

The Project area lies within tributary catchments of the Stikine River (51,600 km^2). The site is bounded to the east by the northerly flowing Mess Creek and to the west by Hickman and Schaft creeks, which also flow northwards and merge with Mess Creek downstream of the Project area. Mess Creek continues northwards and discharges to the Stikine River some 60 km to the north near Telegraph Creek. After its confluence with Mess Creek, the Stikine River flows to the southwest discharging to the Pacific Ocean near Wrangell, Alaska.

Rivers typically flow in deeply entrenched north-south valleys. Headwater areas of the main Project area watershed can be substantially glacierized (i.e., Hickman, Schaft Creek), although lower elevation sub-watersheds do exist that lack glaciers (i.e., Skeeter Lake watershed).

Based on the mine plan (Section 16.0), proposed sites of main project components are dispersed over a number of watersheds and sub-watersheds including Schaft Creek (702 km^2), Hickman Creek (87 km^2), Skeeter Lake (38.6 km^2), and Mess Creek (2,330 km^2). The majority of the mine site infrastructure including the open pit will be located in the Schaft Creek watershed.

Hydrometric field programs were initiated in 2006, discontinued in 2009, and re-initiated in October 2013 with site visits conducted to present. Between 2006 and 2008, up to 10 hydrology stations were operated. However, several of these sites were challenging to maintain and data was unreliable. Since October 2013, the hydrology program has included data collection at four hydrology stations, with a focus on critical sites and locations where good quality data can be collected. Although more recent data has been collected and presented in rating curves, the following summarizes results from the last comprehensive review and reporting of hydrology baseline information from 2006 to 2008 (Knight Piésold, 2010).

The annual hydrograph typically has a unimodal shape, with high flows resulting from snowmelt in the spring and early fall, and low flows throughout the winter. Streamflow in the region is typically highest through June and July due to melting of the winter snowpack, and in August in heavily glaciated watersheds due to glacial melt. Peak instantaneous flows commonly occur during the freshet period on larger rivers, but they may also occur in late summer or early autumn due to intense rain or rain on snow events on smaller streams. Annual runoff was observed to range from 1,870 mm to 400 mm across the Project area, with the majority of the annual runoff occurring in June, July, and August. Flows decrease throughout the winter and minimum flows typically occur in March and early April when the majority of available water was stored within the snowpack. Most watercourses within the Project area are perennial with some level of continuous flow year-round.

The mean annual streamflow for the Project area is highly variable as a result of the range in watershed characteristics within the Project area. Mean annual unit runoff for Schaft Creek, downstream of the proposed waste dump location, is 46 l/s/km^2. Mean annual runoff from the Skeeter Lake valley, for the northern and southern outlet creeks, is 28 l/s/km^2 and 21 l/s/km^2, respectively. The greatest monthly streamflow variability at the project site, as a percentage of the monthly mean, typically occurs in April and in October and November as a result of variations in freshet timing and in storm event precipitation phase and magnitude.

The British Columbia hydrometric standards (British Columbia Ministry of Environment, 2018) were issued in December 2018 as an update from the 2009 Standard. The new standard specifies both the number of stage-discharge measurements required to define a rating curve and the number of measurements required each year to confirm rating curve stability. *Water and Air Baseline Monitoring Guidance Document for Mine Proponents and Operators* (British Columbia Ministry of Environment, 2016) recommends at least two complete years (open water season and winter monitoring) of baseline data collection. The current program does not meet the number of visits per year required by the 2018 Standard (British Columbia Ministry of Environment, 2018). Several years of open-water season monitoring are available at each site, although all sites require more rigorous winter monitoring to meet the monitoring duration guidelines.

To meet current guidelines (British Columbia Ministry of Environment, 2016; 2018) the program will need to be reinstated with a higher level-of-effort. As noted previously, the lack of winter low flow data is a known gap and a winter hydrology field program is recommended. Winter is typically a critical period for mine effects (impacts of water quality and fisheries) and having a robust dataset for this period is necessary for meaningful assessments. This program should be commenced at least two years prior to EA submissions.

20.1.8 Surface Water

Baseline studies on surface water quality have been conducted and reported in various aquatic resources reports from 2006 through 2008 (RTEC, 2010a; 2008a; 2007a). Stream data has been collected from Schaft Creek, Mess Creek, Skeeter Creek, and Yehenico Creek (reference location) watersheds. A total of two lakes (Skeeter Lake and Start Lake) and seven wetland sites were assessed for water quality.

Stream water hardness levels ($CaCO_3$) were greatest at each site during winter (January to March) and spring and decreased during the summer months. Hardness ranged from below detection at most sites during the summer to 123 mg/L in March. All pH values were slightly alkaline and with most readings falling between pH 7.70 and 8.0. No seasonal or spatial trends were observed.

Nutrients peaked from May to July and the Skeeter Creek watershed generally had the highest concentrations. The highly mineralized watersheds in the Project area contribute to the metal concentrations observed in the surface waters. Concentrations of several metals exceed the available guidelines for the protection of aquatic life; metals that most frequently exceeded guidelines include aluminum, cadmium, chromium, copper, and iron. The highest concentrations of some metals coincide with the elevated total suspended solid concentrations that occur during peak flows (i.e., freshet), while other metal concentrations are highest during low flows. The sites that most often exceeded guidelines were in the Schaft Creek watershed.

20.1.9 Groundwater

Groundwater monitoring began at the Project in 2008 with the installation of monitoring wells and was expanded in 2010 (RTEC, 2008; Knight Piésold, 2010). Groundwater monitoring and sampling was conducted several times a year through to 2013. Water-level monitoring resumed on an annual basis in 2015. The following summarizes results from the baseline hydrogeology report, which incorporated data up to 2011 (Knight Piésold, 2012).

20.1.9.1 Groundwater Hydrology

A conceptual understanding of the groundwater hydrology of the Project area has been developed with consideration of the site geology, specifically the surficial geology, groundwater levels, and results from hydrogeologic testing.

The source of groundwater is ultimately from precipitation that falls on the ground surface, where a portion of the precipitation infiltrates through the soil or fractured bedrock into deeper settings below the water table. This occurs in recharge zones. Groundwater typically flows from recharge areas at higher elevations through fractures and permeable zones in bedrock and unconsolidated to semi-consolidated sediments towards discharge areas located at lower elevations. In the Project area, groundwater comes from the infiltrating run-off from snowmelt, ice-melt from glaciers, and from rain. The groundwater surface or zone of saturation in the subsurface is likely a muted replication of the topography.

Groundwater flow is typically related to the geology and structure of the area, which includes the lithology of the geologic units and the geologic structures, such as faults and folds that influence fracturing in the geologic units. Faults can act as conduits and/or barriers to groundwater flow, which often depends upon the type of fault. Thrust faults and reverse faults form from compressional forces, which can fracture the rock, but the fractures may not be open due to compressional stress. Normal faults and transform faults form from extensional forces and fractures formed from these types of faults are more likely to be open. Fractures that form around an extensional fault typically increase the permeability along the fault, but the fault core can be filled with fine-grained gouge that has low permeability, which creates a barrier to flow across the fault. As a result, faults are often conduits to groundwater flow parallel to the fault, but are barriers to flow across the fault. In the Schaft Creek area, many of the fractures are likely created by extensional forces as indicated by the presence of normal faults in the area.

The major aquifers within the study area are located within the alluvial deposits of Schaft Creek Valley and Skeeter Creek Valley. Major bedrock aquifers include the andesite from the Stuhini Group and the granodiorite from the Hickman Batholith.

Groundwater flows from recharge zones in the upper elevations towards lower elevation discharge zones in Schaft Creek, Mess Creek, and Skeeter Creek Valleys. Groundwater in the bedrock flows through fractures and is under confined conditions. No dissolution or karst features have been identified in the Project area. In unconsolidated valley-fill and overburden deposits, groundwater flows through voids and pores between sediment grains. The Schaft Creek Valley-fill aquifer is very permeable and groundwater is under water table conditions. In the Skeeter Creek Valley overburden deposits, groundwater is under localized, confined, and water table conditions. Artesian conditions are common in the Skeeter Creek Valley.

There are three implied groundwater divides to note within the Project area:

1. In the saddle area near the eastern boundary of the deposit. Groundwater east of the divide flows towards Mess Creek and groundwater west of the divide flows towards Schaft Creek.
2. At the northern end of Start Creek drainage. Groundwater under the footprint of the TSF flows south towards Mess Creek along Start Creek.

3. At the southern end of the Skeeter Creek Valley. Groundwater under the footprint of the TSF flows from the southern end of Skeeter drainage northwest towards Schaft Creek.

20.1.9.2 Groundwater Quality

Groundwater quality in the Project area is generally slightly basic to basic, moderately hard to hard, and alkaline. Groundwater generally had total dissolved solids concentrations less than 1,000 mg/L. All monitoring locations have high buffering capacity with alkalinity values consistently greater than 49 mg/L $CaCO_3$, ranging upwards to 201 mg/L $CaCO_3$. Sulphate concentrations exceed the 100 mg/L British Columbia Water Quality Guidelines (BCWQG) limit in at least one well at monitoring locations in the vicinity of the proposed TSF and one other well located within the Schaft Creek floodplain.

Groundwater in the deposit area, Schaft Creek Valley, and Start Lake area are predominantly calcium to calcium-magnesium bicarbonate type. Groundwater in Skeeter Creek Valley ranges from magnesium type to no dominant cation type, and bicarbonate to sulphate type water.

Baseline water quality analytical and in situ data are examined and compared to relevant guidelines with respect to the most sensitive receptors in the downstream environment. Guidelines relevant to water quality data for the project are the British Columbia Ministry of Environment Approved and Working BCWQG for Fresh Water Aquatic Life (BCWQG) – Maximum and the Canadian Council of Ministers for the Environment (CCME), Canadian Environmental Quality Guidelines (CCME) – Water Quality Guidelines for the Protection of Aquatic Life (Freshwater).

Aquatic life guideline exceedances occur for aluminum, arsenic, cadmium, iron, molybdenum, uranium, and vanadium, but these issues are not pervasive. Guideline exceedances vary with location within Schaft Creek Valley. In the southern portion of the valley vanadium exceed guidelines. In the central part of the valley, sulphate, cadmium, molybdenum, and uranium exceed guideline limits, and further north, arsenic and iron are the only parameters to exceed guidelines. In the deposit and saddle areas, arsenic, cadmium, and iron exceed guideline limits. In Skeeter Creek Valley, sulphate, iron, and uranium exceed guidelines. It should be noted that arsenic concentrations in all of the samples that exceed guidelines are near the guideline limit and that uranium concentrations only exceed the more stringent CCME limit and are well below the BCWQG limit. Cadmium exceedances are rare following the redevelopment of the monitoring wells, with only 3 out of 61 samples exceeding the guideline limit for the Project area.

20.1.10 Fisheries and Aquatic Habitat

Fisheries and aquatic baseline studies were undertaken in the Schaft Creek, Skeeter Creek, Mess Creek, and Hickman Creek watersheds as part of the baseline program in the Project area from 2006 to 2008 (RTEC, 2010a; 2010b; 2008a; 2008b; 2007a; 2007b). In addition to surface water quality (Section 20.1.8), other aquatic components assessed in streams, rivers, lakes, and wetlands in these watersheds included sediment quality, primary producers (periphyton, phytoplankton, zooplankton, benthic invertebrates), and secondary producers (fish).

20.1.10.1 Sediment Quality

In Project area streams, concentrations of several metals (i.e., antimony, beryllium, bismuth, cadmium, lead, molybdenum, selenium, silver, thallium, and tin) are consistently near or below detection limits. Metal concentrations are generally lowest in the Schaft Creek sites and highest in Skeeter Creek and

Mess Creek. The concentrations of some metals (e.g., copper and zinc) in Skeeter Creek are considerably higher than those of other sites. The concentrations of several metals, including arsenic, chromium, copper, iron, nickel, and zinc, exceed the available guidelines for aquatic life.

Most lake and wetland sediments in the Project area generally comprise fine particles, particularly silt and clay. Larger-grained sediments can be found where running water can remove many of the finer particles.

Generally, no lakes or wetlands have collectively high metal concentrations. However, Start Lake sediment has the highest copper and zinc concentrations; arsenic, chromium, copper, selenium, and zinc concentrations at various lake and wetland sites exceed CCME interim sediment quality guidelines. In 2008, the probable effect level was exceeded for arsenic and copper, and the lowest effect level and the severe effect level guidelines were exceeded for both iron and nickel.

20.1.10.2 Periphyton, Phytoplankton, and Zooplankton

Average periphyton species richness in streams range from 3 to 15 taxa; with the greatest richness reported in Skeeter Lake Watershed. The average community (Simpson) diversity ranges from 0.3 to 0.9 and does not vary considerably between streams from year to year. The Skeeter Lake Watershed generally has the highest diversity values, while the Schaft Creek Watershed has the lowest periphyton community diversity.

Wetland and lake communities are often composed of large proportions of Bacillariophyceae, Chlorophyta, and Cyanophyta, with wetlands generally having greater taxa diversity than lakes. Average phytoplankton richness varied annually during the baseline program, from 5 taxa to 15 taxa in 2006, 3 taxa to 9 taxa in 2007, and 10 taxa to 51 taxa in 2008.

Relatively large numbers of immature copepods are common in most area lakes. Cyclopoid and calanoid copepods are always the most abundant zooplankton group. Rotifers are the next most abundant group. The remaining community is composed of organisms from the Amphipoda, Bosminidae, Insecta, and Daphnidae groups. Within the surveyed water bodies, zooplankton richness ranges from 2 taxa to 8 taxa. Mess Lake and a previously unnamed lake called L2 are the most diverse lakes.

20.1.10.3 Benthic Invertebrates

The average density of benthic invertebrates across all streams can vary substantially from year to year, with observed values increased from below 1,000 organisms/m^2 in 2006 to 13,717 organisms/m^2 in 2008. The Skeeter Lake watershed has the highest average benthic invertebrate density and average richness (26 taxa) while Schaft Creek watershed possesses the lowest richness (10 taxa).

Stoneflies (Plecoptera), true flies (Diptera), and mayflies (Ephemeroptera) are the dominant taxonomic groups, accounting for 90% of all organisms collected. Ephemeroptera, Plectoptera, and Trichoptera (EPT) taxa are known to be sensitive to environmental stress; thus, a high proportion of these groups indicate high-quality environmental conditions. EPT consistently composed between 50% and 70% of the community throughout most of the Project area streams, with the exception of two sampling sites. The remaining proportions of the benthic invertebrate communities include 11 other taxa: platyhelminthes, amphipoda, hemiptera, trichoptera, mullusca, arachnida, copepoda, nematoda, oligochaeta, cladocera, and ostracoda.

In lakes and wetlands, benthic invertebrate richness was found to range between 5 taxa and 11 taxa throughout the baseline survey period. Generally, wetland sites have higher species richness compared to lake sites; however, the Schaft Creek, Skeeter Lake, and Mess Creek watersheds all have similar average richness values.

20.1.10.4 Fish

The Stikine River hosts a wide assortment of fish species. The provincial Habitat Wizard database lists 20 fish species that are present in the Stikine River watershed, including five species of Pacific salmon. Coho (O. kisutch) and sockeye salmon (O. nerka) are the most abundant of the Pacific salmon species in the river and are known to spawn in the main stream and in tributaries as far north as the Tahltan, Tuya, and Klastline rivers. Only rainbow trout and Kokanee salmon were captured in the Mess and Schaft Creek watersheds. Other species were captured in Yehiniko Creek, a tributary of the Stikine River that was initially sampled as a reference site, and then dropped in later years because of its dissimilarity with other Project area streams.

Rainbow trout are the most widespread species in the watershed, occurring in most of the streams and rivers where habitat is suitable. Rainbow trout were captured extensively during baseline studies, including in the Start Lake watershed. Kokanee salmon was not listed in FishWizard or any other sampling records for the watershed (i.e., FISS database, Habitat Wizard, EcoCat, Mapster); however, they were captured in 2007 and 2008 in Mess Lake during baseline studies. Habitat Wizard does list Chinook salmon (Oncorhynchus tsawytscha), steelhead trout (O. mykiss), and mountain whitefish (Prosopium williamsoni) as species present in Mess Creek; however, these species are only believed to be present downstream of a waterfall barrier located 11 km from the Stikine River confluence, which is approximately 70 km from the Project site.

There are several barriers to fish migration within the Project area that limit fish distribution and community composition. The Stikine River hosts a wide variety of species, including five species of Pacific salmon (Oncorhynchus spp.), trout, char, and coarse fish. In contrast, most of the Mess Creek and Schaft Creek watersheds are inhabited only by rainbow trout (O. mykiss), with the exception of Kokanee salmon (O. nerka) in Mess Lake. No anadromous salmon or coarse species are present in the Mess Creek system, which suggest that barriers have prevented the migration of these species into this watershed.

An approximately 6 m high waterfall 11 km upstream from the Stikine River confluence prevents fish migration into the upper Mess Creek Watershed where the Project is located. The waterfall is located at the downstream end of a short, narrow canyon where it drops approximately 6 m into a large pool. Upstream of the waterfall, several cascades and smaller falls are present and flow through the canyon; it is extremely turbulent, making it unlikely that any fish are able to pass through to upstream areas. No anadromous species have been captured upstream of this waterfall.

A narrow canyon featuring cascading flow and steep chutes is also present on Schaft Creek approximately 10 km north of the mine site. The canyon features several drops of 1 m to 2 m, as well as turbulent flow. No fish were captured upstream of this barrier during sampling from 2006 to 2008, including at sites immediately upstream of the barrier.

A large waterfall is present on Skeeter Creek approximately 75 m upstream of its confluence with Schaft Creek. The waterfall on this stream measures at least 30 m in height and flows directly into a steep cascade that features several 1 m to 2 m drops. Flow through this section is very turbulent, even at low flows. No fish have been captured upstream of this barrier, despite numerous sampling attempts in both stream and lake habitats. Habitat in the upper Skeeter watershed is of excellent quality for salmons, so if fish could access the habitat, they should have been in sufficient numbers to have been captured easily with the applied sampling effort.

20.1.11 Terrestrial Ecosystems

The Project area is located within the steep, rugged terrain of the Boundary Ranges, with a portion of it in the more subdued terrain of the Tahltan Highlands. Ecologically, the area is diverse, with moist coastal ecosystems transitioning to drier interior ecosystems. The eastern portion is characterized by expansive mid-elevation plateaus, while the west is more representative of rugged coastal mountainous terrain, with Mess Creek forming the effective border between these two geomorphologies.

Ecosystems and vegetation for the Project were characterized at the regional and local scales using the Biogeoclimatic Ecosystem Classification system and two mapping methodologies: predictive ecosystem mapping and terrestrial ecosystem mapping (RTEC, 2010c; 2008c; 2008d).

Locally and regionally, 11 biogeoclimatic units are present, 6 of which are forested and 5 of which are associated with parkland/scrub and alpine environments. The Boreal Altai Fescue Alpine undifferentiated (BAFAun) subzone covers the largest extent of the regional area above 1,400 masl and the Engelmann Spruce Subalpine Fir moist cold (ESSFmc) subzone covers the largest extent of the local Project area, for both the proposed mine site and road corridor. The regional area is characterized predominantly by sparse/barren ecosystems, while the local Project area is predominantly mesic forests with dominant young and mature forest structural stages.

Six listed ecosystems of conservation concern (including five blue-listed wetland ecosystems) were identified during baseline studies; however, these would need to be checked against current status, as the red and blue listings change on an annual basis. Sensitive ecosystems identified include riparian, wetland, alpine, and plateau ecosystems. No plant species of conservation concern were identified within the Project area. One “nuisance weed”, common horsetail (Equisetum arvense), was identified during field surveys. This species is not regulated by the British Columbia *Weed Control Act* and is not considered a significant concern in any area of the province.

During the 2007 wetland-specific study, all five federally recognized wetland classes (bog, fen, marsh, swamp, and shallow open water) encompassing 23 provincial wetland ecosystem associations (to Mackenzie and Moran 2004 classifications) covering a total of 844.2 ha were mapped in the Project area. This ecosystem data was combined with the hydrological and aquatic biological survey data to support the descriptions of wetland function, including hydrological, biochemical, ecological, and habitat functions.

Eight plant species identified as being of cultural and/or traditional significance to the Tahltan Nation were identified during vegetation field studies. The majority are berry-producing species (e.g., blueberries, huckleberry, gooseberry) and are found throughout the Project area.

20.1.12 Wildlife and Wildlife Habitat

Baseline field studies were undertaken from 2006 to 2012 to characterize the terrestrial wildlife communities and identify important wildlife habitats in the Project area (RTEC, 2012a; 2010d; 2010e; 2010f; 2008e; 2008f; 2007c; 2007d). Several wildlife species and groups have been identified by communities, interested individuals, government agencies, and the Tahltan Nation as requiring study in relation to Project activities, including grizzly bears, moose, mountain ungulates (mountain goats, Stone sheep, northern caribou), hoary marmots, bats, amphibians, waterfowl, riverine birds, songbirds, and raptors.

Wildlife baseline studies included literature review of management plans specific to the region, identification of species at risk or of interest potentially occurring within the area, and field surveys. Surveys conducted focused on mammal, avian, reptiles, and amphibian (with a focus on western toad) communities. Surveys for mammals included mountain ungulates (mountain goat, stone's sheep, and caribou), moose, and bats. Surveys for birds included waterfowl and riverine birds, raptors, and breeding songbirds.

20.1.12.1 Moose

Moose are one of British Columbia's ungulates with the widest distribution and are abundant in northern British Columbia, with an estimated average population of 170,000 (Kuzyk, 2016). Moose are an important economic and cultural resource in the Skeena Region surrounding the Schaft Creek and Mess Creek watersheds. Moose are traditionally harvested by the Tahltan Nation and recreationally by resident and non-resident hunters.

The results from aerial surveys suggest that there has been a decline in the moose population in the regional Project area from 314 to 112 moose from 2006 to 2012. Productivity of moose in the regional Project area was very low compared to other areas in the region. As there was no evidence of human activity and habitat conditions appeared similar amongst years, wolf predation appears to be a primary contributor to the observed decline in moose numbers in the Project area over that period.

Mineral licks used by moose are generally characterized by well-worn trails leading to wet, muddy springs or seepage areas that contain dense track concentrations. Mineral licks can either be wet or dry. While dry licks are most often associated with use by mountain goat and Stone's sheep, wet licks are used by moose. A mineral spring has been located on the west bank of the Mess River that appears to be a thermal spring, noted as a wildlife habitat feature.

Wallows are a feature associated with moose use and may be attributed to rutting behaviour in the fall by bulls (e.g., a rutting pit or scrape where a moose urinates, paws, and creates a muddy pit). These features are typically produced new each season. Wallows may also indicate annual activity where moose of any sex exploit a wet area by pawing or manipulating the ground to access a resource such as the elements available from a mineral spring. A series of wallows were also observed on the east side of Mess Creek with some evidence of use, including tracks, melted snow, and muddied appearance.

20.1.12.2 Mountain Ungulate

Mountain ungulates receive particular conservation focus from government, Indigenous nations, and public and private stakeholders. Mountain ungulates tend to be important economic and social resources for traditional harvest by Indigenous nations and recreational harvest for resident and non-resident hunters, in addition to having important biological roles within ecosystems.

Surveys in both 2006 and 2008 confirmed the presence of three mountain ungulate species, mountain goat (Oreamus americanus), Stone's sheep (Ovis dalli stonei), and northern caribou (Rangifer tarandus population 15) within the Project area. Mountain goats (154 sightings) are far more abundant in the area than are sheep or caribou; very few sheep (35 sightings) were recorded on surveys and caribou (3 sightings) were very rare.

20.1.12.3 Bats

Bats were detected in the Project area. Little brown myotis (M. lucifugus) and long-eared bats, including western long-eared myotis (M. evotis) or the long-eared myotis (M. septentrionalis), were detected. Bats are suspected of exploiting the site during the growing season; however, there is little likelihood of hibernacula (i.e., winter hibernating habitat) being supported in the Project area.

20.1.12.4 Bird

A total of 21 species of waterfowl were observed within the study area, including 6 dabbling species, 9 species of diver, 2 species of geese, 2 species of loons, trumpeter swans (Cygnus buccinator), and eared grebes (Podiceps nigricollis). During spring, the distribution of waterfowl was relatively uniform among reaches, although higher concentrations existed, particularly within the larger wetland complexes associated with Schaft Creek and upper Mess Creek.

Nine raptor species were identified from the call playback surveys and stand watches as well as from incidental observations during other surveys in 2006. Of these, breeding was confirmed for two species in 2008: osprey and bald eagle.

There was a moderately high diversity of breeding songbirds (61 species) identified with high variation in species numbers and diversity between variable radius point count plots, including the provincially blue-listed and COSEWIC ranked species of concern rusty blackbird. The highest species diversity and abundance of birds was found within low-elevation forest and riparian marshland transitional habitat along upper Mess Creek adjacent to the proposed access corridor. Higher elevation sites above the treeline with predominantly rocky substrata hosted a low abundance and diversity of breeding songbirds.

20.1.12.5 Amphibians

Federally, western toad is a species of special concern under the Canadian *Species at Risk Act*. In British Columbia, western toad is yellow listed (secure but with conservation concern). Western toads undertake yearly migrations from hibernating sites to communal breeding ponds, and then to summer foraging sites, making them very susceptible to habitat fragmentation by roads and other barriers. Migrations occur over distances ranging from 1 km to 5 km during a relatively short period of several days. During these migration periods, adult toads and juvenile toadlets can experience heavy mortality on roads or may avoid roads altogether, thus being unable to reach breeding ponds.

Five confirmed western toad breeding ponds were found during the 2007 field surveys and no additional ponds were predicted to be breeding sites in alternate years.

20.2 Socio-Economic and Cultural Setting

20.2.1 Governance

Federal, provincial, regional, municipal, and Indigenous community governance occurs in the Project area and surrounding vicinity. The Project area lies within the Regional District of Kitimat-Stikine. District of Stewart, the City of Terrace, and the Town of Smithers are the closest communities with municipal governance. Terrace and Smithers are the main population centres closest to the Project, both of which are more than 400 km south. Dease Lake, an unincorporated community, is the largest settlement on Highway 37 and is approximately 300 km to the north-northeast of the Project area.

The non-Indigenous bodies that are unincorporated, and are governed by the regional district in which they are situated, include Dease Lake, South Hazelton, Bell II, Meziadin Junction, and BQL.

Tahltan governance is administered through the band system under the *Federal Indian Act* (1985), with an elected chief and council who oversee the daily socio-economic affairs of the community. The Tahltan Nation is comprised of two bands, the Iskut Band and the Tahltan Band. The Tahltan Central Government is the administrative governing body for the Tahltan Nation. The Tahltan Central Government is governed by an Executive Committee and a Board of Directors comprised of family representatives, with an Elders Advisory Council providing guidance. The Tahltan Central Government Governance Policy 2020 guides the Board of Directors for all governance-related issues, decisions, and actions. THREAT supports the Tahltan Central Government Lands Department and the Tahltan Leadership on matters related to consultation and engagement with resource development, land use, social, cultural, heritage, and socio-economic interests in Tahltan territory.

There is no current treaty between the Tahltan Nation and the governments of either Canada or British Columbia. The Tahltan Central Government manages negotiations with federal and provincial governments, as well as industry

Additionally, the Tahltan Central Government and industry has directly signed agreements related to Tahltan participation, communication, and economic benefits from projects within their territory. Some examples include the Galore Creek Mining Corp Participation Agreement, Crystal Lake Mining Corporation Communications Agreement, Garibaldi Resources Corp Communications Agreement, GT Gold Communications Agreement, Hudbay Minerals Inc.

20.2.2 Socio-Economic

Northwestern British Columbia is characteristically remote, with communities that are widely dispersed and isolated from each other. The area in general holds a greater dependence on primary resource industries such as mining, forestry, and fishing, than the rest of the province. Many smaller communities in the area have a predominantly Indigenous population and the major centres that provide goods and services to the region include Smithers, Terrace, and Prince Rupert.

Smithers and Terrace provide supplies and services to the majority of the remote region where the Project area is located. Communities are scattered and communication and transportation are somewhat limited. The region's economic and social growths have been limited due to poor infrastructure and access, as well as the cold climate hosting long winters. According to British Columbia Ministry of Environment (2017), regional population sizes have decreased in northwestern British Columbia, of which the Stikine has had the largest recent decrease in population size, i.e., half the 2015 population from what it was in 1986.

Northwestern British Columbia has a history of depending on natural resources, particularly forestry, mining, and fishing, as its main economic driver. Currently, and in the foreseeable future, British Columbia relies on these three resources, in addition to liquefied natural gas, tourism, and transportation (e.g., Highway 16 and Highways 37 and 37A are the primary transportation corridors in the northwest). The regional mining industry of the Project area is a key source of employment for communities along Highway 37, including Indigenous communities, and also employs a significant number of Smithers and Terrace residents. Currently, the Red Chris Mine is in operation in the region and located approximately 250 km from the Project area. Three mines in the region recently received their EA certificates and are in the permitting stages: Kerr-Sulphurets-Mitchell Mine, Red Mountain, and Kemess Underground. Mineral exploration is active throughout the region.

The economy in northwestern British Columbia is becoming more diversified over time, and newer industries to the region include energy production (including hydroelectric power generation). In addition, employment levels have increased in the public service, sales and service, tourism, transportation, and mineral exploration sectors. The mining industry provides approximate 30% of jobs for communities along Highway 37 in recent years (Tetra Tech, 2016).

The Project is expected to provide economic benefits to the local communities through direct and indirect employment opportunities. The overall local economic impacts are likely to be beneficial, as well as those to the northern region and province. Project development and employee and company expenditures are important economic benefits to the region, as well as contributing to potential annual revenue contributions through property tax, royalties, licensing fees, and income tax for most or all levels of government.

Highways in the region are paved, except for small sections of Highway 37 north of Iskut and Highway 51 to Telegraph Creek. Terrace and Smithers have major airports handling jets, and smaller airstrips are present closer to the Project area, including Bob Quinn, Dease Lake, Iskut, and Telegraph. The Canadian National Railway rail line joins from Prince Rupert to Prince George, running parallel to the Highway 16 corridor through Terrace, Hazelton, New Hazelton, and Smithers. Cellular phone coverage is limited to the large communities along Highway 16. The Port of Stewart is a deep-water facility that handles ocean-going vessels, and export of mineral products, including concentrate from Red Chris mine.

Socio-economic issues typically observed in the region communities are characteristic of a region in which the economy is strongly tied to fluctuations of the goods sector, specifically in regard to the forestry and mineral exploration industries. Currently, this translates to a lack of employment, job training, or education opportunities, as well as strains on community infrastructure. However, an economic focus on providing support for the mining and resource industry means that all primary and secondary communities in the region are well situated to see economic benefits by the proposed Project in the form of providing construction and operation support.

20.2.3 Tahltan Nation

The Project area lies in the territory of the Tahltan Nation. The Tahltan territory is approximately 95,933 km^2 and overlaps the Stikine, Nass, and Skeena River watershed, and includes part of the Yukon. The Tahltan Nation includes approximately 5,000 members living on and off-reserve. About 600 members live in Tahltan territory, though not all are living on reserve lands in Telegraph Creek, Dease Lake, and Iskut, and more than 5,000 people live elsewhere across the country.

Dease Lake is the farthest north of the three Tahltan communities, and is on Highway 37. It is a regional centre for services and consists both of an off-reserve community and a Tahltan reserve. Telegraph Creek is 120 km southwest of Dease Lake and 700 km northwest of Terrace. It is adjacent to the Stikine River Canyon. Iskut is on Highway 37, 80 km south of Dease Lake and 500 km north of Terrace.

Tahltan people have a historically strong connection with and respect for the land and landscape. The relationship between the people and the land is one marked by a deep respect for the land as provider and a strongly held belief that the people are keepers of the land. The traditional Tahltan idea of wellness is a balance of mental, physical, and spiritual health.

Primarily a hunting and trapping people, the Tahltan fostered inter-tribal trade with neighbouring tribes exchanging items such as fish, furs, and obsidian, useful for making tools and weapons. The Tahltan people held a significant position as middlemen in the pre- and post-contact trading industry of northern British Columbia. The Stikine River supported trade that took place between coastal nations and interior nations. The first contact with Europeans came in 1838 when Robert Campbell of the Hudson’s Bay Company arrived with intentions on setting up operations in the territory.

Fishing is an important land use activity for the Tahltan who have numerous fish-bearing river systems running through their territory. Wildlife species important to Tahltan culture include moose, black bear, grizzly bear, and mountain goat, though traditionally caribou may have been highly valued in some areas of Tahltan territory. Generally, wild game provides the bulk of the diet for Tahltan families and hunting and fishing continues to be the most important subsistence activity.

Tahltan territory is rich in natural resources, including gold, silver, and copper, which has earned this area of the province the reputation of The Golden Triangle, as well as salmon, forests, and wildlife. The abundance of these natural resources has resulted in overwhelming interest by industry in developing projects to extract the resources.

The Tahltan Nation Development Corporation (TNDC) and their established group of companies are significant employers in the area and involved in development projects within Tahltan territory. TNDC is a business corporation owned by the people of the Tahltan Nation through the Tahltan Band, Iskut Band, and the Tahltan Central Government. TNDC pursues sustainable and responsible business and economic development opportunities in the region that lead to employment, training, and business opportunities for Tahltan members. Mining, construction, hydroelectric power, and forestry are some examples of the sectors in which the company is active.

The Schaft Creek Project team has committed to work closely with the TCG, and with TNDC, their associated group of companies, and Tahltan-owned and -partnered businesses to identify employment and contracting opportunities arising from its project development activities.

20.2.4 Archaeology and Heritage

The *Heritage Conservation Act* protects all archaeological sites, whether on provincial or private land, that predate AD 1846 ("pre-contact"), while burial sites and rock art sites are protected regardless of age. This includes as-yet unrecorded sites and archaeological materials from disturbed contexts.

Archaeological investigations of the Project have been undertaken during 2006, 2007, and 2008 (RTEC, 2010g; 2008g). This included an Archaeological Impact Assessment (AIA) for the Project conducted in accordance with the *Heritage Conservation Act* Heritage Inspection Permit 2006-223, issued by the Archaeology Branch. The primary objectives of the AIA were to (1) identify and evaluate any archaeological sites located within and adjacent to the impact zone of the proposed developments, (2) identify and assess possible impacts of the proposed developments on any identified archaeological sites, (3) provide recommendations regarding the need and appropriate scope of further archaeological studies prior to the initiation of any proposed developments, and (4) recommend viable alternatives for managing adverse impacts.

Overall and to date, 51 archaeological sites and 43 historic sites have been identified throughout the Project area. The sites were found at elevations ranging from valley bottoms to the high alpine. All of the sites are obsidian lithic scatters, ranging from single artifact finds to larger sites with numerous artifacts and debitage (waste chips created when making stone tools). Nine artifacts were sent for XRF analysis, which confirmed that nearby Mount Edziza was the source of the obsidian. At the request of the Tahltan Nation, the location of these archaeological sites is kept confidential and is not shown on any maps intended for public dissemination.

Once sites have been identified as archaeological sites, the area, with a 50 m buffer zone, is marked as No Work Areas on relevant plans. Should further work or ground altering activities be planned in these areas, a Site Alteration Permit under Section 12 of the *Heritage Conservation Act* would be applied. Site alteration permits detail the management, mitigation, and monitoring steps that are deemed appropriate for the site.

A total of 43 historic and recent land use features (e.g., cabins, trails, culturally modified trees, mining, camp) were identified during the baseline study. All of these features date to the 20th century and are related to mining exploration during the 1960s to 1980s primarily by Hecla and Teck, and trapping activity. These sites are not protected by the *Heritage Conservation Act*. Fifteen of the historic features are associated with or in close proximity to archaeological sites and many of these sites exhibit some evidence of disturbance from historic activity. In cases where sites had excellent natural exposures, the absence of any formed tools is notable

The Project has adopted and subsequently developed the Tahltan Archaeological Chance Find Procedure to address the possibility of archaeological materials being encountered during surface development activities.

To the extent possible (and limited only by available data), consideration of heritage and archaeology formed part of the Project area infrastructure planning process described in Section 18.0. If areas identified for planned infrastructure have not had AIAs carried out, they will be done at the appropriate time(s), and infrastructure planning will be adjusted, as appropriate and if required.

20.2.5 Land Use

The Project is located within the boundaries of the CIS LRMP, which was completed in October 2000 and encompasses approximately 52,000 km^2 of northwestern British Columbia. LRMPs are sub-regional, integrated resources plans that establish the framework for land use as well as resource management objectives and strategies. The CIS LRMP acknowledges the mineral and energy resource potential within the plan area. Under the plan, exploration and development of mineral deposits, as well as construction of access roads, are allowable activities outside of protected areas, providing they occur in concordance with all relevant legislation.

The plan defines specific land and resource management objectives and includes three management categories: General Management Direction, Area-specific Management, and Protected Area Management. Objectives and strategies of the General Management Direction apply throughout the plan area and include the following key components: biodiversity; wildlife; aquatic ecosystems and riparian habitat; hunting, trapping, guide-outfitting, and fishing; recreation/tourism; visual quality; and timber.

The LRMP created 14 protected areas for which resource conservation is emphasized. Two provincial parks are adjacent to the Project area: Mount Edziza Provincial Park located to the east of the Project and Iskut River Hot Springs Provincial Park located close to the access road corridor. The eastern extent of the Mount Edziza Provincial Park, along Mess Creek, was officially incorporated into the park in 2003 under the Stikine Country Protected Areas Management Plan.

Primary road access in the region is by the main Highway systems: 37/37A, 16, and 113. There are also a number of unpaved routes connecting communities along these main routes. Backcountry areas are typically accessed by networks of industry-created trails in the region. These include forestry access roads and mine service roads that have been opened for public use. Mineral exploration and potential development projects have created private access routes in remote areas, and additional routes may be created by these activities in the region. Current major service roads include the Galore Creek and Eskay Creek project access roads and roads east of Dease Lake: the Turnagain River and Klappan access roads.

Other land user activities include private and public fishing, excursions, commercial and recreational tourism, and subsistence and trapline use. These users typically gain access to activity areas by utilizing existing roads and trails, waterways, or helicopters. Air access in the region is available by helicopter and airplane and is served by a number of airport and airstrips. Terrace and Smithers regional airports are the primary hubs in the region and provide service to a number of carriers in the province. Airstrips at Dease Lake, BQL, and Burrage Creek, as well as serviced helipads in Stewart and Bell II, provide additional access to remote areas and serve as staging areas for exploration activities in the region.

A number of land users are found in the regional area representing corporate, community, and personal interests. Land tenures include primary industry categories such as mineral exploration and forestry, as well as hydroelectric and water resource licences. A number of small, privately owned land tenures are also found in the area, including trapline, guide outfitting, angling and fishing operations, grazing, recreation and tourism, subsistence harvesting, and commercial recreation activities. Resident hunters also utilize the area.

Four guide outfitting operations operate in the study area, including Northwest Ranching and Outfitting, Misty Mountain Outfitters, Golden Bear Outfitting, and Kinaskan Lake Outfitters. Areas utilized by these operations include those along the north side of the Stikine River, sections of Mount Edziza and Kinaskan Lake Provincial Parks, Mess Creek, and sections of the Coast and Spectrum mountain ranges.

Eight registered traplines are located within the Project area, including areas of the Project footprint. Areas focused by trapping activities include Mess Lake and the Klappan area. Commercial recreation enterprises in the area include Bear Enterprises Mountaineering whose activities include skiing, mountaineering and climbing, and rafting excursions. Areas utilized by these commercial recreation ventures include areas adjacent to Mount Edziza Provincial Park and between the Iskut and Stikine Rivers, as well as the Unuk and Spatsizi Rivers. Angling operations are not required to hold permits in the province; however, Bell II Lodge is reported to run angling tours along the Unuk River and Iskut River systems.

In addition to commercial operations, recreational tourism is common in the region but is primarily confined to areas accessible from Highway 37. A number of land use activities associated with tourism in the area include hiking, wildlife viewing, photography, horseback riding, canoeing/kayaking, snowmobiling, and all-terrain vehicle use. Mount Edziza Provincial Park also offers a variety of trails connecting Highway 37 to the north regions of the park. Support infrastructure along Highway 37 is available for tourism but is separated by long distances and includes Bell II Lodge and Tatogga Lake Lodge. Additional services are available to travelers in Dease Lake.

20.3 Human Health Setting

20.3.1 Drinking Water

Due to its location in remote wilderness, there are currently little-to-no industrial developments or human activities within the Project area. The nearest light industrial activities (e.g., highway maintenance) are at BQL on Highway 37 North, roughly 60 km southeast of the Project. The community of Telegraph Creek (population of approximately 300) is located roughly 60 km north of the Project. Because of the distance between the Project and BQL and Telegraph Creek, human consumption of local surface waters for drinking is minimal.

There are no permanent residents in the Project area and no water licences exist in the immediate area. There are several in the regional vicinity based on the baseline land use study area. These include two commercial water licences east of the Project; one is issued to Ministry of Transportation and Infrastructure for enterprise-related water withdrawal from BQL, and another is held by Coast Mountain Hydro (owned by AltaGas Ltd.) for run-of-river power generation. However, neither is related to domestic (i.e., drinking water) use. North of the Project area, a number of water licences for irrigation and domestic use are located on tributaries of the Stikine River. All are situated at least 1 km upstream of the confluence of Mess Creek and the Stikine River; therefore, any potential Project activities would not affect these drinking water sources. This includes the community of Telegraph Creek.

The Project area covers a number of overlapping land use activities and tenures. These include Tahltan land use, trapping, guide outfitting, and commercial recreation. Information from land use interviews conducted for the Project indicates that no specific surface water body is used as a drinking water source in the Project area. However, various land users, including guide outfitters, hunters, and recreational users typically consume untreated surface waters. Therefore, an assessment of drinking water quality is warranted.

20.3.2 Country Foods

The country foods assessment indicates that baseline concentrations of most metals present no health risk to consumers. Current consumption patterns of all foods can be maintained based on background concentrations. There is a very low potential health risk for zinc intake from eating moose muscle. Also, on rare occasion, very low potential health risk is associated with copper intake from consumption of moose kidney and grouse, and with mercury intake from consumption of rainbow trout. However, conservatism employed in the risk assessment and in developing tolerable daily intake rates means that these identified potential risks are highly unlikely for any individuals. There is considerable uncertainty associated with the assessment since background levels of metals from all other pathways have not been resolved, but these are likely very low. Therefore, there is no evidence of health risks from consumption of country foods under current conditions.

20.3.3 Noise

Background noise levels are low in the Project area because of its location in a remote wilderness area with negligible anthropogenic activity. Telegraph Creek, the community closest to the Project site, is located approximately 60 km north. Highway 37 is approximately 50 km east of the Project site.

The average equivalent noise level in the Project area is 45 dBA, with equivalent noise levels ranging from 40 dBA to 48 dBA. These baseline equivalent continuous sound level (Leq) values are at or slightly above the permissible sound level defined in the Noise Directive (Alberta Energy Regulator, 2007), which is 40 dBA Leq during night time (and 50 dBA during day time) within 0.5 km of a new facility. Although this limit is a regulation for the energy industry in Alberta but not British Columbia, it provides an understanding of reasonable sound level conditions in the environment and puts the monitoring results into context. Baseline data therefore indicate that background noise in the Project area is higher than typical work sites. However, noise levels are within the range of baseline noise levels (39 dBA to 51 dBA) reported for the Galore Creek Project in 2005 (Rescan, 2006). For the Project area, the noise maxima ranged from 62 dBA to 90 dBA and the noise minima ranged from 27 dBA to 35 dBA.

Wind effects such as whistling through and rustling of trees or other vegetation are the cause of the majority of the noise in the Project area. Wind speeds recorded at the Saddle meteorology station generally increased during the night throughout the baseline study period. This is consistent with the trend of increasing night-time noise.

20.4 Environmental Management and Monitoring

20.4.1 Environmental Management Plans

Environmental management plans for the Project would need to be developed to support EA and permitting. Environmental management plans are plans developed to be site-specific and to ensure that necessary measures are identified and implemented in order to protect the environment and comply with environmental legislation. They include legislative requirements, policies, BMP, committed mitigation measures, and monitoring and reporting commitments. Environmental management plans may include, but are not limited to the following:

- Surface Water Management and Monitoring Plan
- Groundwater Management and Monitoring Plan
- RSF Management and Monitoring Plan
- Waste Management Plan
- Dangerous Goods and Hazardous Materials Management Plan
- ML and ARD Management Plan
- Fish and Aquatic Habitat Management Plan
- Noise Management Plan

- Traffic and Access Management Plan
- Sediment and Erosion Control Plan
- Air Quality Management and Monitoring Plan
- Emergency and Spill Response Plan
- Wildlife Management and Monitoring Plan
- Vegetation and Wetland Management Plan
- Hazardous Materials Management Plan
- Archaeological and Cultural Resources Management Plan
- Transportation Management Plan
- Contingency Plans

20.4.2 Water Management

Construction water management at the TSF should commence approximately two years prior to mill start-up and coincide with initial construction of the facility. This phase is characterized by extensive clearing, grubbing, and stripping; the development of access roads and haul roads; and establishing water management and sediment control systems.

Non-contact water will be diverted around the TSF during operations to the extent practical to minimize the volume of the supernatant pond and reduce surplus water that would need to be discharged. Un-diverted runoff will be managed within the TSF. Process water will be discharged into the TSF with the tailings slurry and supernatant water reclaimed back to the mill for use in Mineral Resource processing.

Seepage from the TSF will be largely controlled by the tailings beach, zoned embankments, and the embankment drains that will discharge into the water management ponds downgradient of the embankments. Surface water runoff from the embankment face, seepage through the embankment fill material, or other runoff and seepage from impacted areas in the vicinity of the TSF will be collected and directed to the water management ponds located at topographic low points along the downstream toe of the embankments.

Surplus water will be removed from the TSF throughout operations to limit the accumulation of water within the facility and to maintain adequate beach length. Surplus water will be removed via a floating pump barge and discharged into the East Diversion Channel, which will then flow out into Schaft Creek. TSF discharge quality and quantity criteria will be developed in consultation with regulators.

Water collected in the downstream water management ponds will be continuously monitoring and pumped back into the TSF depending on water quality and permit limits (which have yet to be defined).

20.4.3 Waste Management

20.4.3.1 Waste Management (Tailings, Waste Rock, and Overburden)

Tailings Management

Tailings will be impounded in a TSF in the Start Creek and Skeeter Creek Valleys, located to the northeast of the open pit, and due north of the plant site.

The TSF has been designed to comply with relevant standards and guidelines including the following:

- CDA Dam Safety Guidelines (2013; 2014)
- Health Safety and Reclamation Code for Mines in British Columbia (2021) and accompanying Guidance Document (2016)
- Mining Association of Canada's (MAC) Guide to the Management of Tailings Facilities (2019)

The TSF concept is presented in Section 18.8 of this report.

Waste Rock Management

Mine waste rock will be securely and permanently stored in RSFs and within the TSF embankments. The waste rock storage concept includes the following considerations:

- Protection of the regional groundwater and surface waters, both during operations and after closure
- Maintain geotechnical stability
- Achieve effective reclamation at mine closure

Overburden Management

Topsoil and overburden materials removed from the foundations of various site infrastructure (TSF, Open Pit, Plant Site, etc.) will be stored in stockpiles for use in ongoing and final reclamation.

Soil stockpiles will be constructed with minimum 3H:1V side slopes and will be removed at closure as material is used for constructing the closure covers for the TSF and RSFs, and for reclaiming the access roads and other site infrastructure.

20.4.3.2 Hazardous Waste Management

A variety of supplies and materials classified as potentially hazardous will be required for general operations at the mine and mill. Hazardous materials generated on site will be backhauled from the site by a licenced contractor on an ongoing basis. Materials to be removed from the site include waste batteries, oil and solvents and empty petroleum and reagent drums, carboys, and pails.

Hazardous materials will be segregated and stored using accepted management practices. The storage area will be a safe, non-smoking work area designed to prevent contamination of the environment and contain spills. The storage area will include spill kits, protective equipment, fire prevention systems, spill cleaning, and prevention equipment.

Waste minimization procedures can be implemented before start-up and as an ongoing program during operation. These procedures may include the use of non-hazardous materials in lieu of hazardous materials, developing alternative methods or processes to reduce the generation of high volume wastes, segregating and handling waste streams to minimize cross contamination of wastes, and making waste minimization procedures a part of employee training programs.

20.4.3.3 Non-hazardous Waste Management

Non-hazardous waste is defined as kitchen, biological, and general camp waste; industrial waste includes inert bulk wastes other than mining wastes.

Inert solid wastes will be deposited into a landfill in a small, drainage-controlled area. Inert waste will be progressively buried by overburden soils, encapsulating it.

Non-food waste products that are not incinerated or landfilled immediately will be collected, sorted, and placed in designated areas. Waste will be placed in sealed, wildlife-resistant containers and stored for backhaul to off-site disposal or recycle facilities or for transport to the incinerator or landfill.

An incinerator will be provided for the incineration of combustible waste and will also burn waste oil. Incinerator ash will be collected in sealed, wildlife-resistant containers, and transported to the landfill.

20.5 Closure and Reclamation

20.5.1 General

The British Columbia *Mines Act* (1996) and *Health, Safety and Reclamation Code for Mines in British Columbia* (2021) require mining operations to carry out a program of environmental protection and reclamation to ensure that, upon termination of mining, land, watercourses and cultural heritage resources will be returned to a safe and environmentally sound state and to an acceptable end land use. Closure objectives will be developed in consultation with Project stakeholders.

Closure and reclamation planning for the Project will contribute to the success of closure and reclamation during mining and at the end of mine life, which will reduce the need to restructure mine components, limit the amount of material re-handling, and reduce the environmental effects of the Project. Mine development and operation will incorporate techniques to minimize surficial disturbance and, where possible, progressively reclaim areas affected during construction and operation. Stabilizing and rehabilitating surfaces reduces the potential for degradation of resources due to extended exposure to climatic factors and reduce closure-related capital costs at the cessation of mining activities.

The following subsections provide an overview of the reclamation and closure plan for the Project. This plan will be revised as necessary to reflect updates to the mine plan in the development and advancement of the Project.

20.5.2 Reclamation Objectives

Under the British Columbia *Mines Act* and *Health, Safety and Reclamation Code for Mines in British Columbia*, the primary objective of the closure and reclamation plan will be to return areas disturbed by mining operations to acceptable land use and capability.

The following goals are implicit in achieving this primary objective:

- Long-term preservation of water quality downstream of decommissioned operations at the compliance point
- Long-term stability of engineered structures, including the RSF, open pit, and TSF
- Decommissioning, removal, and proper disposal of access roads, structures, and equipment that will not be required after the end of the mine life
- Long-term stabilization of exposed erodible materials
- Natural integration of disturbed areas into the surrounding landscape, and restoration of a natural appearance to the disturbed areas after mining ceases
- Establishing a self-sustaining cover of vegetation that is consistent with existing forestry and wildlife needs

20.5.3 Reclamation Prescriptions for Site Components

Long-term physical stability of mine features is considered during the design of permanent structures, such as the open pit, TSF, and RSFs. The following sections describe the reclamation prescriptions and approaches for key site components following operations.

20.5.3.1 Tailings Storage Facility

The TSF will be developed as described in Section 18.0. During the final years of mill operations, whole tailings would be selectively discharged around the TSF to establish a final tailings beach to facilitate surface water management and reclamation.

The surface of the TSF would be capped with a layer of rockfill from either the open pit or the RSFs (depending whether the mine is still in operation when closure is initiated), and a layer of soil from one of the stockpiles. The cover soils may include a combination of inorganic soil such as till and organic topsoil recovered during initial construction. Swales would be excavated in the tailings beaches to make the grade less uniform and promote drainage towards the spillway. The reclaimed surface would then be hydroseeded and planted with appropriate vegetation. Downstream embankment slopes would be capped with a layer of soil and hydroseeded.

Surface water diversion channels and access roads not required for long term monitoring would be removed. A permanent outlet channel and spillway at the north embankment would be constructed to enable discharge of surface water from the TSF pond to Skeeter Creek.

A seasonal pond may be present in the facility, close to the closure spillway. The TSF surface pond would be designed to attenuate storm inflows to minimize the magnitude of spillway discharge flows.

Tailings distribution and reclaim water infrastructure (including the surplus water system, reclaim barges, sand cyclone plant, etc.) will be removed at the end of operations and either salvaged or disposed of off-site.

Water management ponds and collection systems would be removed at such time that suitable water quality for direct release is achieved.

20.5.3.2 Open Pit

At the end of mine life, the pit would be allowed to fill naturally with groundwater and precipitation (i.e., no pumping required) following closure and discharge to Schaft Creek.

20.5.3.3 Rock Storage Facilities

Concurrent reclamation of the RSFs would occur throughout the life of the mine. The remaining unreclaimed surfaces of the RSFs would be covered with a layer of topsoil, hydroseeded, and planted with appropriate vegetation at the end of the mine life. Removal of the water management ponds and collection systems would occur at such time that suitable water quality for direct release is achieved.

20.5.3.4 Mine Site Infrastructure

Associated mine facilities include all of the buildings and structures developed to support the mine operation including the process plant, camp, administration and maintenance shop, fuel storage, and explosives storage. All of these features would be demolished at closure and reclaimed in a consistent manner.

Salvageable items within the buildings would be removed and sold. Most of the non-hazardous, inert building materials would be disposed in an acceptable disposal facility. Concrete footings installed to support the associated facilities would be broken up and buried onsite. The landfill would be closed using best practice methods and provincial guidelines for landfill closure.

Hazardous wastes identified during closure would be remediated through best practices and removed from the site and disposed of in an approved facility. Any metal-contaminated soils would be removed and disposed of in the tailings pond.

Following removal of the facilities and any associated contamination, all disturbed areas would be recontoured to blend into the surrounding topography. Salvaged reclamation material placed in stockpiles and windrowed during construction would be replaced over the disturbed areas. These areas would be fertilized and seeded with ground cover crop and revegetated with planting of native shrub and tree seedlings either from cuttings or nursery grown or seeded with erosion control mix depending on the moisture conditions.

20.5.4 Post-Closure Monitoring

The level of post-closure monitoring will be a function of the environmental performance of the mine site. Monitoring requirements are expected to decrease over time as the potential impacts to the receiving environment decrease.

Post-closure monitoring will likely consist of the following:

- Upkeep of water management ponds and recycle pumps used to collect seepage and embankment runoff, which will be retained until monitoring results indicate that runoff and seepage from the TSF and RSFs are of suitable quality for discharge.
- Groundwater monitoring wells and geotechnical instrumentation retained for long-term monitoring and assessed on a schedule defined in the detailed closure plan developed prior to closure.
- Annual inspection of the TSF and ongoing evaluation of surface water quality, flow rates, and instrumentation records.
- Ongoing Dam Safety Reviews conducted at an appropriate interval as per relevant provincial guidelines (EMLI, 2016; 2021).
- Monitoring of metal uptake in revegetation and effectiveness of revegetation of disturbed areas.
- Maintenance of site roads and electrical infrastructure that are necessary beyond closure to support ongoing monitoring and inspection requirements.

20.5.5 Closure and Reclamation Cost Estimates

20.5.5.1 Financial Assurance (Bonds)

The *Mines Act* stipulates that the Chief Inspector of Mines may require that the mine owner provide monetary security. Security is required for all, or part of, outstanding costs associated with mine reclamation and the protection of land, watercourses, and cultural resources, including post-closure commitments. Security held under the *Mines Act* can also be used to cover regulatory requirements of legislation, permits, and approvals of other provincial agencies.

British Columbia's reclamation security objective is to provide reasonable assurance that the provincial government will not have to contribute to the costs of reclamation and environmental protection if a mining company defaults on its obligations. In the case of a company default, security should allow government to successfully manage environmental issues at the mine site, complete any outstanding reclamation requirements, and continue to monitor and maintain the site for as long as is required. In general, the Ministry of Energy, Mines, and Low Carbon Innovation reviews reclamation security at a mine site every five years, or whenever significant changes occur at the mine. Security can increase or decrease depending upon assessed liability at the time.

Security is held by the Minister of Finance and remains in effect until the Chief Inspector of Mines is satisfied that all reclamation requirements for the operation have been fulfilled. Reduction in reclamation security is realized when a liability has been removed or the reclaimed land meets the approved end land use and permit conditions. Exploration activities and work programs intermittently from 2013 to 2018 have been completed under the Notice of Work and Reclamation Permit MX-1-647. A deposited security of $695,000 for Schaft Creek is held under Permit MX-1-647 with the Minister of Finance.

20.5.5.2 Reclamation and Closure Costs

A cost estimate for the conceptual closure and reclamation plan has been prepared for the Project. The cost estimate is based on the following objectives of the plan:

- Minimize or eliminate residual environmental effects following closures.
- Establish conditions that allow the natural environment to recover from mining activities.
- Establish long-term physical, chemical, and ecological stability in the disturbed area.

The estimate was developed by identifying the tasks required to achieve the closure and reclamation objectives. The estimate is based on neat-line quantity take-offs with allowances for construction variances. Where sufficient detail does not exist to develop quantities for a line item, a lump sum or provisional sum allowance was based on similar projects and estimates. The total estimated costs were also adjusted based on industry benchmarks.

Unit rates were developed using production rates, material costs, and contractor equipment rental rates from the following sources:

- Caterpillar Performance Handbook
- British Columbia Road Builders and Heavy Construction Association Equipment Rental Rate Guide (British Columbia Blue Book)
- RS Means Heavy Construction Cost Data

A summary of the closure and reclamation costs for each area are summarized in Table 20-3 below.

Table 20-3: Closure and Reclamation Cost Estimate

Area	Estimated Costs
TSF Reclamation	$56.8 million
Site Access and Development Reclamation	$4.2 million
Waste Rock and Stockpile Reclamation	$20.6 million
Open Pit Reclamation	$0.3 million
Decommissioning of Mine Site Infrastructure	$6.0 million
Miscellaneous and Indirects	$33.2 million
Contingency	$32.9 million
TOTAL	**$154.0 million**

20.6 Environmental Assessment and Permitting

20.6.1 Provincial Process

In British Columbia, EAs are managed by the EAO, a neutral regulatory agency within the provincial government that works with and seeks input from scientific professionals, Indigenous groups, proponents, the public, local governments, and federal and provincial agencies to ensure that major projects meet the goals of environmental, economic, and social sustainability. The EAO follows a clearly defined process in the *Environmental Assessment Act* (SBC, 2018) to conduct the assessment of a major project.

The British Columbia Reviewable Projects Regulation (British Columbia Reg. 67/2020) provides criteria for determining which projects should be required to undergo an EA, by defining prescribed project categories and providing thresholds for each category that seek to indicate the potential for adverse effects for their specific project type. Projects that fall into a prescribed category and meet the thresholds specific to its category require an assessment under the Reviewable Projects Regulation. Most major projects are reviewable; for mineral mines, a new mineral mine facility that, during operations, will have a production capacity of greater than 75,000 t/a of mineralized materials must obtain an EA certificate. Based on the mine plan production estimates provided in Section 16.0, the Project would see Mineral Resource mined from an open pit up to a rate of up to 133,000 t/d, which exceeds this threshold.

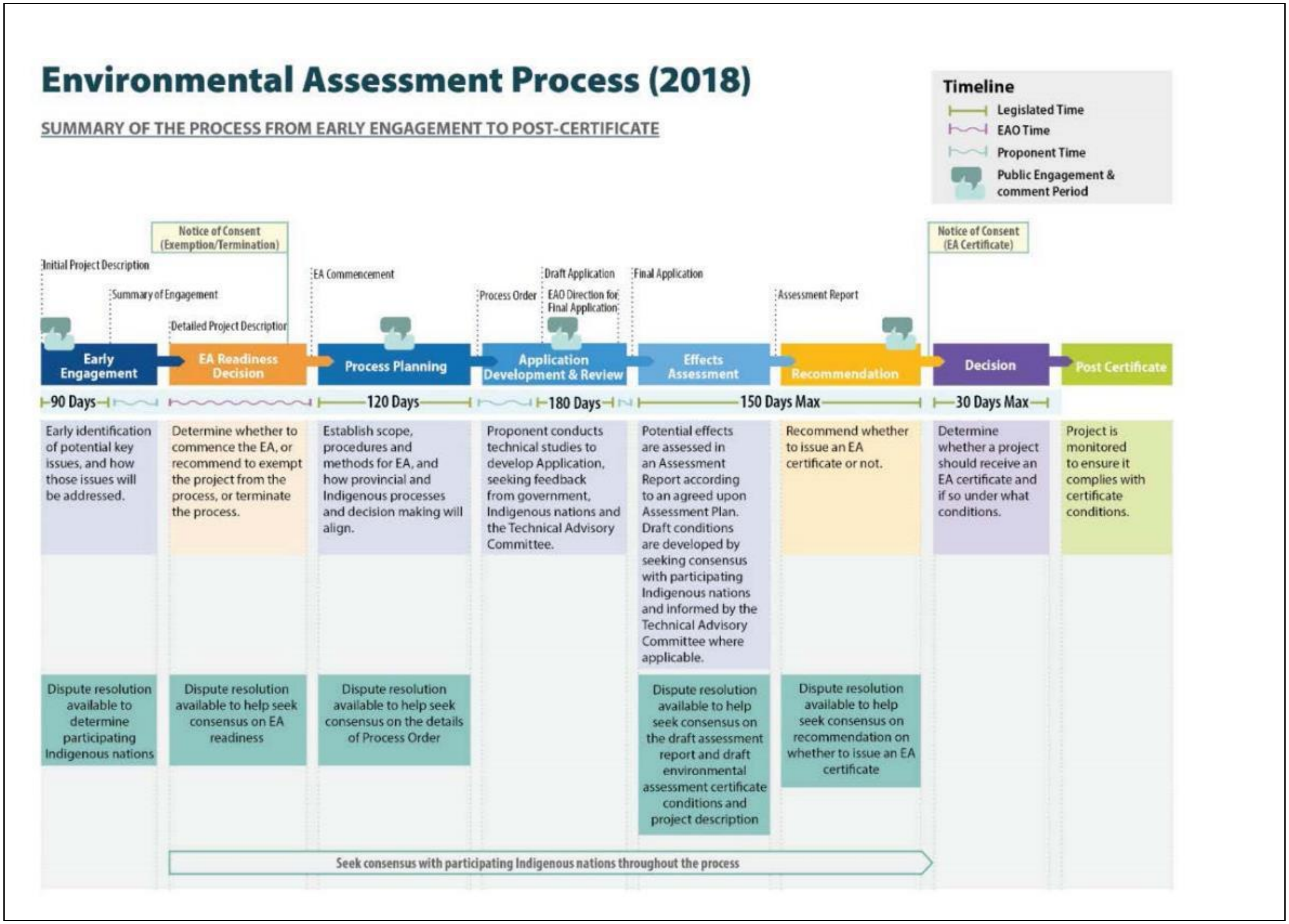
The BCEAA and accompanying regulations and guidance documents establish the overarching regulatory framework for undertaking EA in British Columbia. Within this framework, each project must assess its potential environmental, economic, social, heritage, and health effects that may occur during the life cycle of the project using project-specific scope, procedures, and methods with each assessment tailored specifically to the circumstances of the proposed project. The assessment process also ensures that the issues and concerns of the public, Indigenous groups, communities, and government agencies are considered. This approach allows for the assessment to focus on key issues relevant to the Project when determining whether or not the Project should proceed.

The new *Environmental Assessment Act* (2018) is in force and regulations, policies, and guides are currently being developed to support the new Act and EA Revitalization. The EA Revitalization process resulted in changes to EA legislation, regulation, policies, and practices that focus on three objectives:

- Enhancing public confidence and meaningful engagement
- Advancing reconciliation by implementing the standards set out in the United Nations Declaration on the Rights of Indigenous Peoples, the Truth and Reconciliation Commission Call to Actions, and the Supreme Court of Canada *Tsilhqot'in* decision in the context of EA
- Protecting the environment while offering clear pathways to sustainable project approvals

The following outlines the phases of the EA process as outlined in *EAO User Guide Introduction to Environmental Assessment Under the Provincial Environmental Assessment Act (2018)* (EAO 2020 [March 30]; Figure 20-1).

Figure 20-1: Overview of Environmental Assessment Process Under the Environmental Assessment Act (2018)



1. **Early Engagement:** This phase is the start of the regulatory process with the EAO and provides an opportunity for all participants to better understand the project and establish a foundation for the rest of the EA. Early Engagement begins when an Initial Project Description and Engagement Plan are accepted and ends when the Summary of Engagement is published, and the list of participating Indigenous nations is confirmed. Based on the initial project description, participating Indigenous nations and provincial governments, as well as local communities and the public have the opportunity to identify key questions and issues early. Information gathered during this phase may inform the development of the Detailed Project Description. This phase includes a minimum 30-day public engagement and comment period on the Initial Project Description and EAO has 90 days from when the Initial Project Description is accepted to produce a Summary of Engagement.

   After EAO has published the Summary of Engagement, the proponent has up to one year to submit a Detailed Project Description.

2. **EA Readiness:** This phase begins when the Detailed Project Description is submitted and ends when a decision on whether a project should proceed to an EA. During this phase, the EAO will seek consensus with participating Indigenous nations on the decision or referral. There are several decision options including requiring a revised Detailed Project Description, proceeding to a typical EA, proceeding to an EA via assessment body, exempting the project from the EA requirement, and terminating the project from the process. The timeline for a decision will vary and there is no legislated timeline for this phase.

3. **Process Planning:** This phase starts with the notice of decision to proceed to an EA and ends when the Process Order is issued, a designated timeline of 120 days. During this phase, a project-specific Process Order is developed in a consensus seeking process with participating Indigenous nations. Each Process Order is customized based on a standard order to reflect the scope and circumstances of the project and will include an Assessment Plan, Permitting Plan, and Application Information Requirements. The EAO conducts a public engagement and comment period of at least 30 days on the draft Process Order. A Technical Advisory Committee (TAC) is established for each EA and is expected to function as a single reviewing body and act as the forum for the detailed, independent technical review of all the proponent's documents and technical studies.

4. **Application Development and Review:** Begins when the Process Order is issued and ends when the Application is accepted through a consensus seeking process with participating Indigenous nations that issues have been adequately addressed by the proponent. A proponent has 3 years from the date of the issued Process Order to develop an Application. During Application Development, a proponent conducts technical studies and engagement established in the Process Order to develop an Application for an EA Certificate. Application Review starts when the proponent submits an Application. During the Application Review portion of this phase, the EAO, participating Indigenous nations, and TAC review and comment on the Application. Technical issues would need to be addressed and incorporated into the proponent's revised Application. The Application Review timeline is 180 days and includes a minimum 30-day Public Engagement and Comment Period. The revised Application must be submitted within one year of the EAO directing the proponent to prepare the final version.

5. **Effects Assessment and Recommendation:** Begins with the acceptance of the revised Application and ends when the referral package for provincial decision makers is finalized. Through a consensus-seeking process with participating Indigenous nations and with input from the TAC, EAO conducts the effects assessments and develops a referral package that includes

the Assessment Report, EA Certificate with conditions and certified project description, and recommendations to Ministers on whether the project is consistent with the promotion of sustainability by protecting the environment and fostering a sound economy and well-being of British Columbians and their communities. Timeline for this phase is 150 days.

6. **Decision:** Begins when the referral package is submitted and ends when Minister of Environment and Climate Change Strategy and the Responsible Ministers decide to issue or refuse an EA Certificate and publish reasons for the decision. Timeline for this decision is 30 days.
7. **Compliance and Enforcement:** The Compliance and Enforcement branch of the EAO conducts compliance inspections of regulated parties and projects, and where required, uses enforcement to ensure that projects are designed, built, operated, and decommissioned or reclaimed in compliance with the legally binding requirements of the Act, its regulations, and any EA Certificates or Exemption Orders. Compliance and enforcement continue throughout the life of the project.
8. **Post-Certificate:** If an EA Certificate is issued, post-certificate activities include mitigation effectiveness reporting and may include audits, certificate amendments, extensions, and transfers. Post-certificate activities may occur throughout the life of a project.

The decision to issue or not issue an EA certificate for a mining project is made by the Minister of Environment and Climate Change Strategy and the Minister of Energy, Mines and Low Carbon Innovation (the responsible Ministers). In making their decision, the Ministers must consider the Assessment Report, the Chief Executive Assessment Officer's recommendations, the sustainability and reconciliation purposes of the EAO, and any other matters they consider relevant to the public interest.

Upon completion of an EA, reviewable projects may be granted an EA certificate under Section 17(3) of the *Environmental Assessment Act*. Reviewable projects must obtain an EA certificate (or exemption order) before undertaking any activity to construct, operate, modify, dismantle, or abandon all or part of the facilities associated with the project. A certificate is legally binding and contains conditions that must be followed for the life of the project to mitigate potential adverse effects. An EA certificate is a pre-requisite for authorizations or approvals under other provincial or federal statutes. An EA certificate will describe the permissible physical works and activities of the Project, describe the conditions of how the project will be implemented, and specifies the deadline for the holder of the certificate to substantially start the project not more than 10 years after the issue date. The holder of a certificate may apply for a one-time 5-year extension upon request and at the Chief Executive Assessment Officer's discretion. Once the Project has substantially started, the certificate remains in effect for the life of the Project, subject to suspension or cancellation. Proponents may apply to amend their certificate as project circumstances change.

The EA process was initiated for the Project by Copper Fox, entering into the pre-application phase in 2006, and subsequently withdrawn upon Schaft Creek JV's request in March 2016.

The Schaft Creek AIR/EIS Guidelines was issued by the EAO and the CEA Agency on February 7, 2011. Development of new application information requirements will be required for the Project for the EA process under the new *Environmental Assessment Act* (2018). However, much of the information gathered during the development of the Schaft Creek 2011 AIR/EIS guidelines can be used toward the development of the required documents of the EA process. For example, Copper Fox consulted with local, provincial and federal government representatives, Tahltan Nation, and the public regarding issues and concerns related to the Project. Early in the pre-application stage, Copper Fox held open houses in Telegraph Creek, Dease Lake, Iskut, Terrace, and Stewart, and with the EAO's Schaft Creek advisory working group to identify issues and concerns to be identified and addressed in the EA process documents and Application. The EAO's advisory Schaft Creek Working Group formed in 2011 was comprised of representatives from federal, provincial, local governments, Tahltan Nation, the State of Alaska, and the US federal government.

20.6.2 Federal Process

The Project is also subject to federal EA requirements pursuant to the *Impact Assessment Act* (2019) and its regulations, which replaced the CEAA (2012) on August 28, 2019. The Impact Assessment Agency of Canada (formerly the Canadian Environmental Assessment Agency) released new procedures, policy, and guidance documents to reflect these new legislated changes. Importantly, under the new *Impact Assessment Act*, the Government of Canada is committed to meeting the objective of "one project, one assessment" in its review of projects as per the established and updated Canada-British Columbia Impact Assessment Cooperation Agreement between Canada and British Columbia that sets out ways the jurisdictions will work together on impact assessment that require both levels of government.

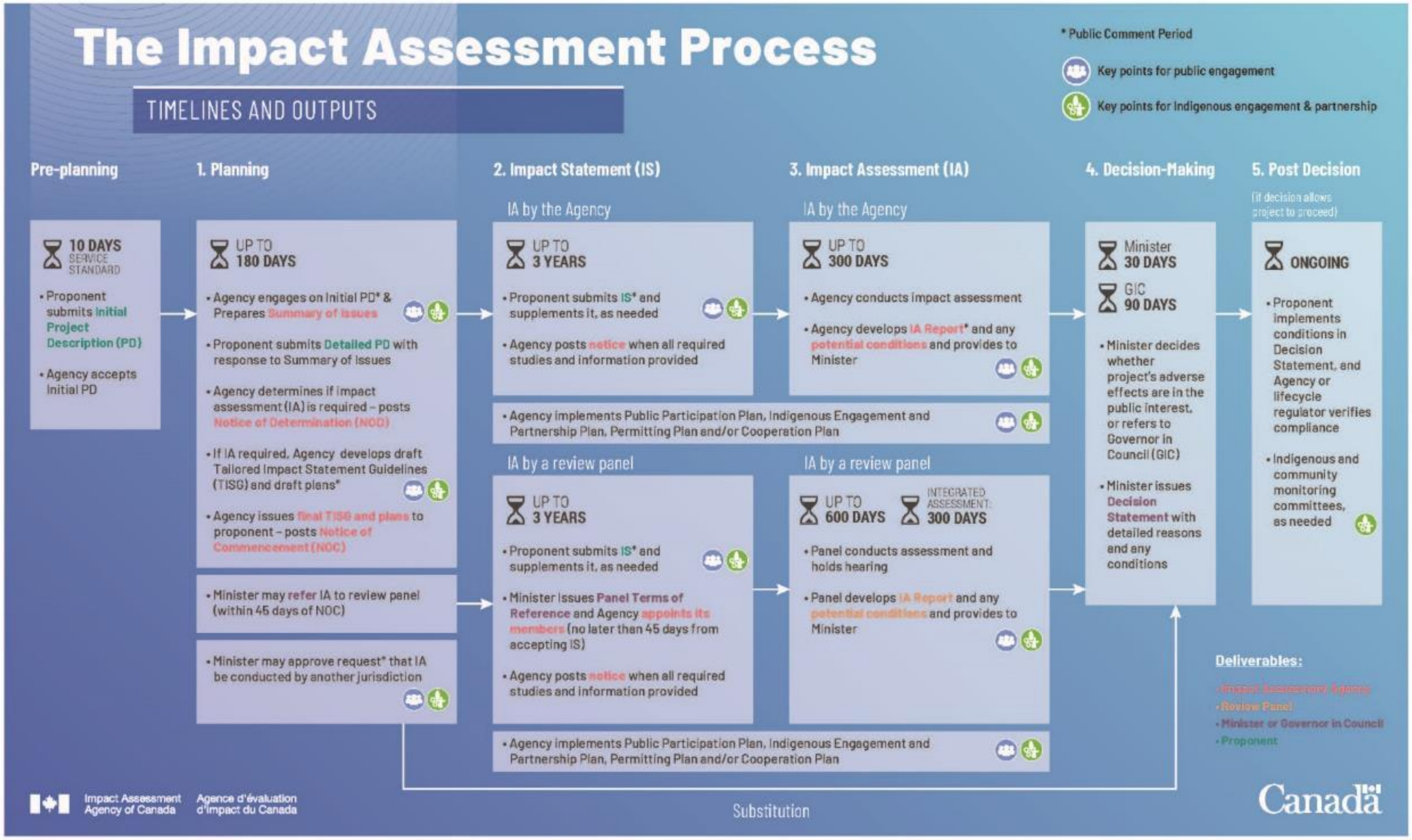
This agreement allows provincial and federal agencies to work together by either (1) entering into a substitution agreement to allow the provincial process to be substituted for the federal process, or (2) working together in a coordinated manner to complete the review of a project.

Under the new *Impact Assessment Act*, regulations released called the Physical Activities Regulation (SOR/2019-285) identify thresholds for mineral mine projects that may be subject to a federal assessment process, as follows:

> Section 18: The construction, operation, decommissioning, and abandonment of one of the following:
>
> (a) a new coal mine with a coal production capacity of 5 000 t/day or more;
>
> (b) a new diamond mine with an ore production capacity of 5 000 t/day or more;
>
> (c) a new metal mine, other than a rare earth element mine, placer mine or uranium mine, with an ore production capacity of 5 000 t/day or more;
>
> (d) a new metal mill, other than a uranium mill, with an ore input capacity of 5 000 t/day or more;
>
> (e) a new rare earth element mine with an ore production capacity of 2 500 t/day or more;
>
> (f) a new stone quarry or sand or gravel pit with a production capacity of 3 500 000 t/year or more.

Since the Project is a new metal mine, with a Mineral Resource production capacity of 133,000 t/d, the Project is subject to the federal impact assessment process (as summarized below and in Figure 20-2).

Figure 20-2: Overview of Impact Assessment Process Under the Impact Assessment Act (2019)



9. **Phase 1 Planning:** Planning phase includes preparing and submitting an Initial Project Description, followed by a Detailed Project Description as per the Information and Management of Time Limits Regulation (SOR/2019-283). Following Agency review of both documents, a Summary of Issues is provided for proponent response. The Agency then determines if an impact assessment is required and posts the decision and reasons for the decision on the Registry. If an impact assessment is required, the Agency continues engagement with Indigenous groups, the public, other jurisdictions, and federal expert developments in order to develop the Public Participation Plan, Indigenous Engagement and Partnership Plan, Impact Assessment Cooperation Plan, Permitting Plan, and the Tailored Impact Statement Guideline, which includes the scope of the factors that are considered as part of the assessment. Once finalized, the Agency posts the documents with a Notice of Commencement. A 180-day time limit for the planning phase starts when the Initial Project Description is accepted and posted by the Agency; however, the Agency can extend the planning phase by up to 90 days to enable cooperation with other jurisdictions.

10. **Phase 2 Impact Statement:** During this phase, the proponent collects information and conducts studies as required in the Tailored Impact Statement Guidelines and undertakes analysis of the potential impacts of the designated project. Following consultation with Indigenous nations and the public, the proponent prepares the Impact Statement containing the information and studies outlined in the Tailored Impact Statement Guidelines and submits it to the Agency. There is a three-year timeline on the proponent for submitting the required information in the Tailored Impact Statement Guidelines.

11. **Phase 3 Impact Assessment:** Once the Agency posts the Notice of Determination that it is satisfied that the Impact Statement contains all of the required information and studies outlined in the Tailored Impact Statement Guidelines, the time limit of up to 300 days begins to review the Impact Statement. The Agency develops and seeks review on the Impact Assessment Report, potential conditions, and Consultation Report during this phase.

12. **Phase 4 Decision-Making:** Upon completion of Impact Assessment, the Agency provides the Minister with the Impact Assessment Report, Consultation Report, and potential conditions. The Minister then must determine if the adverse effects within federal jurisdiction and the adverse direct or incidental effects are in the public interest, or refer the determination to the Governor in Council. Once the determination is made, the Minister issues a Decision Statement, which must be issued within 30 days (or 90 days for Governor in Council) after the Impact Assessment Report is posted.

13. **Phase 5 Post Decision:** Implementation and compliance phase of the process whereby the proponent implements mitigation measures and follow-up programs and the Agency verifies and posts information relating to compliance and enforcement.

20.6.3 Provincial Permits

While the BCEAA prohibits issuance of provincial permits before an EA certificate is issued, the Concurrent Approval Regulation (British Columbia Reg. 371/2002) allows for parallel review of related provincial permit applications. This regulation applies to provincial permits, authorizations, and approvals necessary to undertake works that are within the scope of the assessment under the Act. Statutory permit approval processes are normally more specific than those required for the EA level of review, for certain permits require detailed and possibly final engineering design information. To be eligible for concurrent review, the approval must be required to construct, operate, modify, abandon, or otherwise undertake part of the 'reviewable project' that is the subject of the EA.

The British Columbia Major Mine Permitting Office typically leads the permitting process for major mines across British Columbia. This office works with proponents, Indigenous nations and government technical advisors to coordinate multi-agency regulatory permits and implements the efficient and timely review of applications for new major mines and major expansion projects.

Table 20-4 presents the list of provincial authorizations, licences, and permits that are anticipated to be required for the construction and / or operation of the Project. The list is not intended to be exhaustive due to the complexity of government regulatory processes and the large number of minor permits, licences, approvals, consents and authorizations, and potential amendments that would be required throughout the life of the mine.

Table 20-4: Anticipated Provincial Authorizations, Licences, and Permits Required for the Project

Authorization	Statute	Agency
Environmental Assessment Certificate	*Environmental Assessment Act* (2002)	British Columbia Environmental Assessment Office
Permit Approving the Work System and Reclamation Program	*Mines Act* (1996)	British Columbia Ministry of Energy, Mines and Low Carbon Innovation
Explosives Storage and Use Permit	*Mines Act* (1996)	British Columbia Ministry of Energy, Mines and Low Carbon Innovation
Mining Lease	*Mineral Tenure Act* (1996)	British Columbia Ministry of Energy, Mines and Low Carbon Innovation
Air Emissions Discharge Permit	*Environmental Management Act* (2003)	British Columbia Ministry of Environment and Climate Change Strategy
Effluent Discharge Permit	*Environmental Management Act* (2003)	British Columbia Ministry of Environment and Climate Change Strategy
Fuel Storage Permit	*Environmental Management Act* (2003)	British Columbia Ministry of Environment and Climate Change Strategy
Refuse Discharge Permit	*Environmental Management Act* (2003)	British Columbia Ministry of Environment and Climate Change Strategy
Hazardous Waste Registration	*Environmental Management Act* (2003) Hazardous Waste Regulation (1988)	British Columbia Ministry of Environment and Climate Change Strategy
Sewage Permit; Municipal Effluent Discharge Registration	*Environmental Management Act* (2003) Municipal Sewage Regulation (1999) Municipal Wastewater Regulation (2012)	British Columbia Ministry of Environment and Climate Change Strategy
Special Waste Generator Identification	*Environmental Management Act* (2003) Hazardous Waste Regulation (1988)	British Columbia Ministry of Environment and Climate Change Strategy
Potable Water System Construction Permit	*Drinking Water Protection Act* (2001) Drinking Water Protection Regulation (2003)	British Columbia Ministry of Health Northern Health Authority
Potable Water System Operation Permit	*Drinking Water Protection Act* (2001) Drinking Water Protection Regulation (2003)	British Columbia Ministry of Health Northern Health Authority

table continues…

Authorization	Statute	Agency
Food Premises Permit	*Public Health Act* (2008) Food Premises Regulation (1999)	British Columbia Ministry of Health Northern Health Authority
Water Licence / Authorization related to surface and or groundwater supply, withdrawal, point of diversion	*Water Sustainability Act* (2014) Water Sustainability Regulation (2016) *Water Protection Act* (1996)	British Columbia Ministry of Forests, Lands and Natural Resource Operations and Rural Development
Approval or Notification of "Changes in and about a Stream"	*Water Sustainability Act* (2016)	British Columbia Ministry of Forests, Lands and Natural Resource Operations and Rural Development
Groundwater Well Registration	*Water Sustainability Act* (2014) Groundwater Protection Regulation (2016)	British Columbia Ministry of Forests, Lands and Natural Resource Operations and Rural Development
Occupant Licence to Cut	*Forest Act* (1996)	British Columbia Ministry of Forests, Lands and Natural Resource Operations and Rural Development
Licence of Occupation	*Land Act* (1996)	British Columbia Ministry of Forests, Lands and Natural Resource Operations and Rural Development
Road Use Permit	*Forest and Range Practices Act* (2002) Forest Use Regulations (2009)	British Columbia Ministry of Forests, Lands and Natural Resource Operations and Rural Development
Special Use Permit	*Forest and Range Practices Act* (2002) Provincial Forest Use Regulation (1995)	British Columbia Ministry of Forests, Lands and Natural Resource Operations and Rural Development
Approval for Oversized Loads or Bulk Haul	*Motor Vehicles Act* (1996)	British Columbia Ministry of Transportation and Infrastructure
Controlled Access Permit	*Transportation Act* (2004)	British Columbia Ministry of Transportation and Infrastructure
Heritage Inspection Permit	*Heritage Conservation Act* (1996)	British Columbia Ministry of Forests, Lands and Natural Resource Operations and Rural Development
Burn Registration	*Wildfire Act* (2004) / Wildfire Regulation (2005)	British Columbia Ministry of Forests, Lands and Natural Resource Operations and Rural Development
Wildlife Salvage and Removal – General Permit	*Wildlife Act* (1996) / Permit Regulation (2000)	British Columbia Ministry of Forests, Lands and Natural Resource Operations and Rural Development

20.6.4 Federal Permits

In addition to the Decision Statement by the Minister of the Environment and Climate Change, the Project also requires other federal permit authorizations described below and listed in Table 20-5.

Table 20-5: Anticipated Federal Authorizations, Licences, and Permits Required for the Project

Authorization	Statute	Agency / Authorized By
Environmental Assessment Decision Statement	*Impact Assessment Act* (2019)	Minister of the Environment
Fisheries Authorization	*Fisheries Act* (1985)	Fisheries and Oceans Canada
Schedule 2 Amendment	Metal and Diamond Mining Effluent Regulation (SOR/2002-222)	Environment Canada
Explosives Licence	*Explosives Act* (1985) Explosives Regulation, 2013	Natural Resources Canada
Radio Licence	*Radio Communications Act* (1985)	Innovation, Science and Economic Development Canada

20.6.4.1 Fisheries Authorization

Fish and fish habitat are protected under the *Fisheries Act* (1985), as well as other federal acts, regulations, and principles. Recently, the *Fisheries Act* was modernized (on June 21, 2019), with new provisions and stronger protections for all fish and fish habitat and strengthened Indigenous role in project reviews. Any project or activity that causes the death of fish, other than by fishing, and the harmful alteration, disruption, or destruction of fish habitat requires an authorization under section 35(2) of the *Fisheries Act*. The Act now includes the ability to make regulations that clearly define the projects, or parts of projects, that would always require a ministerial permit. Once these regulations designating projects are developed under the new Act, proponents will know which projects always require a permit and have greater certainty around process and associated timelines for authorization.

A Fish Habitat Compensation Plan must be submitted to Fisheries and Oceans Canada for an authorization under Section 35(2) of the *Fisheries Act* for the Project. The primary objective of the plan will be to offset unavoidable impacts to fish habitat through the creation or new habitat or improvement of existing habitat.

20.6.4.2 Metal and Diamond Mining Effluent Regulation (SOR/2002-222)

The Metal and Diamond Mining Effluent Regulation (SOR/2002-22) is enacted under the *Fisheries Act* (1985) and applies to metal and diamond mines in Canada that exceed an effluent flow rate of 50 m^3 per day and deposit a deleterious substance in any water or place referred to in subsection 36(3) of the *Fisheries Act*. These regulations impose effluent discharge limits for cyanide, arsenic, copper, lead, zinc, nickel, radium-226, and suspended soils, as well as maximum and minimum pH levels. These regulations also prohibit the discharge of effluent that is acutely lethal to fish (rainbow trout).

Under the regulations, proponents must conduct environmental effects monitoring programs to monitor and report on mine effluent quality, flows, and the results of periodic effluent scans to identify adverse effects of mine effluent (if any) on fish, fish habitat, and on the use of fisheries resources. Environmental effects monitoring studies include effluent characterization, receiving water quality monitoring, sub-lethal effluent toxicity tests, site characterization, fish population surveys, fish tissue analysis, and benthic invertebrate community surveys.

20.6.4.3 Schedule 2 Amendment of the Metal and Diamond Mining Effluent Regulation

Section 5(1)(a) of the Metal and Diamond Mining Effluent Regulation authorizes a proponent to deposit waste rock or effluent that contains any concentration of a deleterious substance into a tailing impoundment area that is listed as a waterbody set out in Schedule 2 – Tailings Impoundment Areas. An amendment to Schedule 2 of the regulation is required if a project intends to construct the tailing management facility and dispose (and storage) of tailings in habitat or natural waterbody frequented by fish. As per Section 18.0, the TSF footprint encroaches on Start Lake, a fish bearing waterbody; thus a Schedule 2 Amendment is likely required for the Project. If the Schedule 2 amendment is required, it could impact the project schedule duration. Additional data collection effort is expected to be required.

Environment Canada administers the Schedule 2 amendment process. Under Section 27.1 of the Metal and Diamond Mining Effluent Regulation, a Fish Habitat Compensation Plan is required to offset losses of fish habitat.

20.7 Environmental and Socio-cultural Considerations

20.7.1 Environmental and Socio-Cultural Factors and Risks

There are no significant environmental risks to prevent the Project from advancing to the next logical phase of study along its development toward becoming an operating mine. That being said, projections of environmental and community matters, and associated costs and permitting schedule are subject to a number of known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here. This includes projections as to permitting timelines, timing and conditions of permits required to initiate mine construction, and potential delays in the issuance of permits; as well as changes to government regulation of mining operations, environmental issues, permitting requirements, and social risks; or unrecognized environmental, permitting and social risks, and title disputes or claims.

Moving forward, planning of the development and associated studies should consider the following project-specific environmental and social factors and risks.

20.7.1.1 Water Quality

The water management plan assumes that mine contact water can be released to the receiving environment without major mitigation in the form of water treatment for residuals from explosives, acidic conditions, metals, or other parameters of potential concern. The geochemical characterization program indicates that only a small portion of the mine waste generated during operations is net acid generating, and as such, widespread ARD is not expected during operations or during post-closure. However, neutral leaching of metals from mine waste could, even at low levels, require treatment prior to release to the receiving environment. Further studies, moving forward, should include an assessment of water quality predications, associated aquatic risk assessment, and consideration of residual nitrates from explosive use to characterize the potential for water quality concerns.

Contact water from the mine rock stored within the Schaft Creek floodplain presents a management risk if runoff quality is not suitable for direct release to the environment as challenges may occur in the capture of water. In addition to further assessment of water quality (as noted above), consideration should include evaluation of management opportunities and mitigation strategies to minimize this risk.

20.7.1.2 Fisheries Authorization and Schedule 2 Amendment

As stated in Section 20.6.4, a fisheries authorization and Schedule 2 Amendment under the Metal and Diamond Mining Effluent Regulation are expected to be required for project development. Further baseline studies to support project feasibility and EA would also include additional assessments of fish habitat to inform the Fish Habitat Compensation Plan required for these authorizations. Potential risk to the project cost and timeline can be mitigated by inclusion and consideration of the development and implementation of this Fish Habitat Compensation Plan and Schedule 2 amendment permitting process duration in the overall project schedule and planning. Construction and mine development can proceed before Fisheries Authorization or Scheduled 2 Amendment is issued if development does not impact habitat under review.

20.7.1.3 Listed Wetlands and Wetland Compensation Plan

As noted in Section 20.1.11, six listed ecosystems of conservation concern were identified during baseline studies in the Project area. The Federal Policy on Wetland Conservation (1991) applies to federal departments and agencies when addressing the loss of wetlands and their functions from a project, if that project requires a federal permit, licence, and or authorization, which through the issuance of, would result in affecting wetland(s) designated as ecologically or socio-economically important to a region. Since the Project requires federal permits, further baseline studies moving forward should include an assessment of wetlands impacted by project infrastructure to inform whether a Wetland Compensation Plan is required.

20.7.1.4 Wildlife

Wildlife has been identified as an important environmental component for the Project area and region, including the presence of large ungulates, grizzly bear, and furbearers. Previous wildlife baseline studies in the Project area, as detailed in Section 20.1.12, has included mountain ungulate summer and winter surveys as well as moose winter surveys. The study found a substantial population of mountain goat, with presence of Stone's sheep distributed throughout the Project area, as well as suitable occupied wintering habitat for moose. Further studies and data collection of wildlife presence, distribution, habitat, and sensitive areas should form an important component of the baseline program moving forward in preparation for project permitting, EA, and development of wildlife mitigation and management plans to support project development, including road access.

20.7.1.5 Indigenous Peoples Land Claims, Rights, and Title

The Project area lies within the Tahltan territory. The Tahltan collectively hold rights to hunt, fish, trap, and harvest berries and other food and medicinal plants throughout their territory. As far as is known by Copper Fox, the Tahltan Nation is the primary claimant of Indigenous rights and title in the Project area.

British Columbia's *Declaration on the Rights of Indigenous Peoples Act* (2019, Bill 41 - First Reading [Declaration]) passed in November 2019 affirms the application of the United Nations Declaration on the Rights of Indigenous Peoples (UNDRIP) and commits to harmonizing existing provincial laws with the individual and collective Indigenous rights proclaimed by UNDRIP. Canada has also committed to legislating UNDRIP's implementation. Free, prior, and informed consent is an integral part of UNDRIP. The provincial and federal governments recognize that meaningful engagement with Indigenous peoples aims to secure their free, prior, and informed consent when Canada and British Columbia proposes to take actions which impact them and their rights, including their lands, territories and resources.

20.7.1.6 Archaeological and Heritage

The Project area is within a traditionally highly used area close to a significant source of obsidian potential for Tahltan use and archaeological value. As per Section 20.2.4, overall and to date, 51 archaeological sites and 43 historic sites have been identified throughout the Project area. All of the sites are obsidian lithic scatters, ranging from single artifact finds to larger sites with numerous artifacts and debitage (waste chips created when making stone tools). The *Heritage Conservation Act* protects all archaeological sites, whether on provincial or private land, that predate AD 1846 ("pre-contact"), while burial sites and rock art sites are protected regardless of age. This includes as-yet unrecorded sites and archaeological materials from disturbed contexts.

Due to the high archaeological potential of this area and pursuant to the *Heritage Conservation Act*, AIA must continue as part of baseline programs and throughout project development, with a focus on areas identified for surface development. Should an archaeological site be found within the infrastructure area during project development, an archaeologist would be required to assess the site and recommend viable options for managing adverse impacts to the site. A Schaft Creek archeological chance find procedure is implemented as part of project related activities.

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimate

Tetra Tech compiled a Capex for this PEA scoping study. The total estimated pre-production capital cost for the design, construction, installation, and commissioning for all facilities and equipment is USD$2,653.2 million. The LOM sustaining capital costs is estimated at USD$848.7 million, inclusive of USD$154 million in Closure Costs. A summary of the Capex is presented in Table 21-1.

This estimate has been prepared in accordance with the Class 5 Cost Estimate standards of the AACE. The accuracy of the estimate is ±30% (see NR dated September 20, 2021). A detailed cost breakdown is presented in Table 21-1.

Table 21-1: Pre-production Capital Cost Summary

Area Code	Description	Initial Capex (USD$ million)
Direct Costs		
10	Overall Site	137.3
20	Mining	188.8
30	Primary Crushing	50.3
35	Stockpile and Reclaim	41.8
50	Grinding, Flotation and Regrind	500.0
60	TSF Tailings Storage Facility (TSF)	137.2
70	Environmental (included in sustaining capital)	-
80	Site Services and Site Utilities	29.5
85	Ancillary Buildings	117.8
86	Plant Mobile Fleet	6.8
87	Temporary Services	4.3
88	Off-site Infrastructure and Facilities	82.2
	Subtotal – Direct Costs	**1,296.1**
Indirect Costs		
90	Project Indirect Costs (including owner's costs)	771.0
	Subtotal – Direct + Indirect Costs	**2,067.1**
99	Contingencies (~25%) + Provisions	586.1
	Total	**2,653.2**

Note: Figures are rounded to hundred thousands. Sums may not add up due to rounding.

This estimate includes direct field costs required to execute the Project, plus indirect costs associated with design, construction, and commissioning. This estimate is based on pricing as of Q4 2020, with no allowances for inflation or future escalation. All currency in this Capex is expressed in United States dollars, unless otherwise noted.

21.1.1 Class, Base Date, and Validity

Estimate Classification and Accuracy

This estimate has been prepared in accordance with the Class 5 Cost Estimate standards of the AACE. The overall accuracy range of the estimate is within ±30%.

Estimate Base Date

This PEA estimate is prepared with a base date of Q4 2020 and does not include any escalation beyond this date.

Validity

This estimate is based on pricing as of Q4 2020, with no allowances for inflation or future escalation.

21.1.2 Project Currency, Foreign Exchange Rates, and Measurement System

Currency and Foreign Exchange Rates

This estimate uses United States dollars (USD) as the base currency. All dollar figures are presented in USD unless otherwise noted.

Foreign exchange rates, noted in Table 21-2, were applied as required.

Table 21-2: Foreign Exchange Rates

Base Currency	Currency
CAD 1.00	USD 0.77
CAD 1.00	EUR 0.70

Measurement System

The Metric System of measurements is used in this estimate.

21.1.3 Scope of the Estimate

The scope of the estimate includes the following:

Overall Site

Overall site includes site preparation, access road, site security, site lighting, site fire alarm system, and site-wide communication system.

Open-Pit Mining

Open-pit mining includes the cost of pre-production development, major and ancillary mining equipment fleet, and infrastructure development.

Process

Process includes crushing, conveying, grinding, classification, regrind, flotation, concentrate dewatering, reagent preparation/storage, and concentrate handling. Mill building, pebble building, concentrate handling, and tailings pumping and sand cycloning are included in this area. The cost estimates also include overall process plant control systems.

Tailings Storage Facility

Tailing storage facility area includes site preparation, disposal and reclaim, tailings embankment, seepage collection, sediment control, TSF electrical distribution, and geotechnical instrumentation.

Site Services and Site Utilities

Site services and utilities include water supply, plant and instrument air, waste management and fuel and propane storage, and electrical distribution.

On-Site Infrastructures

Site infrastructures and ancillary buildings include administration building, laboratory, truck shop and warehouse, gate houses, cold storage, truck wash, lube storage, and camp.

Off-Site Infrastructures

Off-Site Infrastructures include transmission lines and storage facilities at Stewart.

Owner's Costs

Owner's costs include project insurance, head office costs, project-related costs, pre-start up costs, permitting and environmental costs, and owner's engineering team.

Indirects

Indirect costs include Engineering, Procurement, and Construction Management (EPCM), initial fills, capital and operating spares, commissioning, vendor assistance during construction, freight and construction indirects such as temporary facilities, utilities and services, 100-t and above crane rental and services, and housekeeping and catering at the camp.

21.1.4 Responsibility

Tetra Tech is responsible for the development of the overall Capex with inputs from the following consultants and client.

- Tetra Tech – Open-pit mining equipment and pit dewatering, overall site, access roads, site services and utilities, process plant, and all associated onsite and offsite infrastructure
- Knight Piésold – TSF and associated infrastructure, overall site water management, and power transmission line
- Copper Fox – Owner's costs

21.1.5 Estimate Structure

The estimate has been assembled according to the following hierarchical structure:

- Level 1 – Major Area
- Level 2 – Area
- Level 3 – Sub-Area

Table 21-3 outlines the major areas of the Work Breakdown Structure (WBS). The Major Area format is used to report the initial Capex used in Table 21-1.

Table 21-3: Major Areas (Level 1)

Major Area (Level 1)	Description
10	Overall Site
20	Mining
30	Primary Crusher
35	Stockpile and Reclaim
50	Grinding, Flotation, and Regrind
60	Tailings Storage Facilities
70	Environment
80	Site Services and Site Utilities
85	Ancillary Buildings
86	Plant Mobile Fleet
87	Temporary Services
88	Off-Site Infrastructure and Facilities
90	Project Indirects
98	Owner’s Costs
99	Contingency and Provisions

Major Area coding is based on the area numbering system and is intended to be location-specific (the “where” component of the estimate).

21.1.6 Methodology

This estimate has been developed based largely on first principles. The work to complete the estimate is broken down into four categories:

- Estimate supporting documents
- Cost basis – direct costs
- Cost basis – direct field costs
- Cost basis – indirect costs

21.1.7 Estimate Supporting Documents

The Capex was based on the following documents:

- Project scope of facilities
- Design criteria

- Process flow diagrams
- Process equipment list
- General arrangement drawings
- Area layout drawings
- Discipline material take-offs
- Major equipment quotations updated from vendors
- Project WBS
- Supplemented sketches where required
- Tetra Tech in-house database
- Inputs from Knight Piésold
- Inputs from Copper Fox

21.1.8 Cost Basis – Direct Costs

Engineering material take-offs were based on neat line quantities, unless specified otherwise, derived from the project drawings and sketches. Allowances were included for each discipline as appropriate. Conceptual quantities were prepared where drawing information was not available.

Mining

The following items were used to complete the mining portion of the estimate:

- Pre-production development
- Open-pit mining infrastructure costs were based on historic data
- Land freight, assembly, and erection activities were based on in-house information.

Civil (Bulk Earthwork / Site Preparation)

Bulk earthwork and site preparation quantities were based on updated layout drawings.

Concrete and Structural Steel

Concrete and structural steel quantities were based on the updated area layout and general arrangement drawings. These quantities were rolled up in each WBS sub-area to the standard concrete type and steel type as per PEA standard.

Concrete and rebar supply costs were based on information from recent projects. The cost of concrete includes the supply of concrete, forms, supply of cement, aggregates, and mixer trucks.

Structural steel unit rate used in the estimate included install and supply and was based on information from recent projects.

Architectural

Architectural quantities were rolled up into a single line item per WBS sub-area as allowance based on the updated area layout and general arrangement drawings where available.

Mechanical

The mechanical equipment quantities, sizing, and power consumption were derived from the process flow sheets and equipment list. Electric or hydraulic motors were itemized and priced with the equipment.

Major equipment is based on updated quotations from vendors.

Allowances were included for all mechanical bulks such as platework, tanks, vessels, pump boxes, and chute work. Cost of tanks and vessels is calculated based on the estimated equipment size.

The standard installation man-hour database was used to calculate the installation hours for mechanical equipment.

Piping, Electrical and Instrumentation

Piping, electrical, and instrumentation costs were rolled up for each WBS sub-area and presented as allowances. The piping estimate includes pipes, valves, fittings, hangers/supports, testing, and installation labour. The electrical and instrumentation cost estimate includes electrical equipment, field instruments, cables, control wires, bus work, hangers/supports, termination, testing, and installation labour cost.

21.1.9 Cost Basis – Direct Field Costs

Labour Rates

The construction schedule (and hence labour cost) has been based on shifts of 10 hours/day for 7 days/week. The work rotation has been assumed to be 3-weeks-on/1-week-off.

The labour rate used for the estimate is based on blended labour rate, including:

- Base rate
- Mobilization and demobilization
- Vacation and statutory holiday pay
- Fringe benefits and payroll burdens
- Overtime and shift premiums
- Small tools

- Consumables
- Personal protective equipment
- Non-productive time (such as tool box briefing, breaks, etc.)
- Site supervision
- Contractor's general administration costs
- Contractor's overhead and profit

Productivity Factor

A productivity factor of 1.2 is used for this study.

21.1.10 Cost Basis – Indirect Costs

Construction Indirect

This area includes cost for contractor management and general support, and all temporary buildings, utilities and services required during construction and commissioning. The administration and warehouse buildings are assumed to be erected at the start of construction and will be used as offices and temporary storage during construction.

Construction indirect was estimated to be 20% of total direct cost.

Spares

Allowances have been made for spares based on the value of the original equipment. The following allowances have been included in the estimate:

- 3% of equipment cost for capital spares
- 2% of equipment cost for commissioning spares.

Initial Fills

Initial fills were estimated to be $15.0 million which includes initial fills of process consumables, such as reagents, liners, and grinding media for the first three months supply.

Freight and Logistics

Freight and logistics were estimated based on 8% of equipment and bulk material costs. Generally, it includes the following.

- Land and ocean transportation
- Loading and offloading, including craneage
- Customs duties and brokerage
- Bonds and insurance

Commissioning and Start-Up

This estimate includes an allowance for commissioning and start-up based on 2% of total mechanical supply cost.

Engineering, Procurement, and Construction Management

EPCM cost of $225.2 million was a factor of total direct cost, according to Cost Estimator's experience, and with inputs from consultants and client.

Vendor Assistance During Construction

The estimate included an allowance for vendor assistance based on 2% of total mechanical supply cost.

Owner's Costs

Owner's costs of $129.6 million were based on inputs from Owner. This cost was also benchmarked against other similar size projects.

21.1.11 Contingency and Provisions

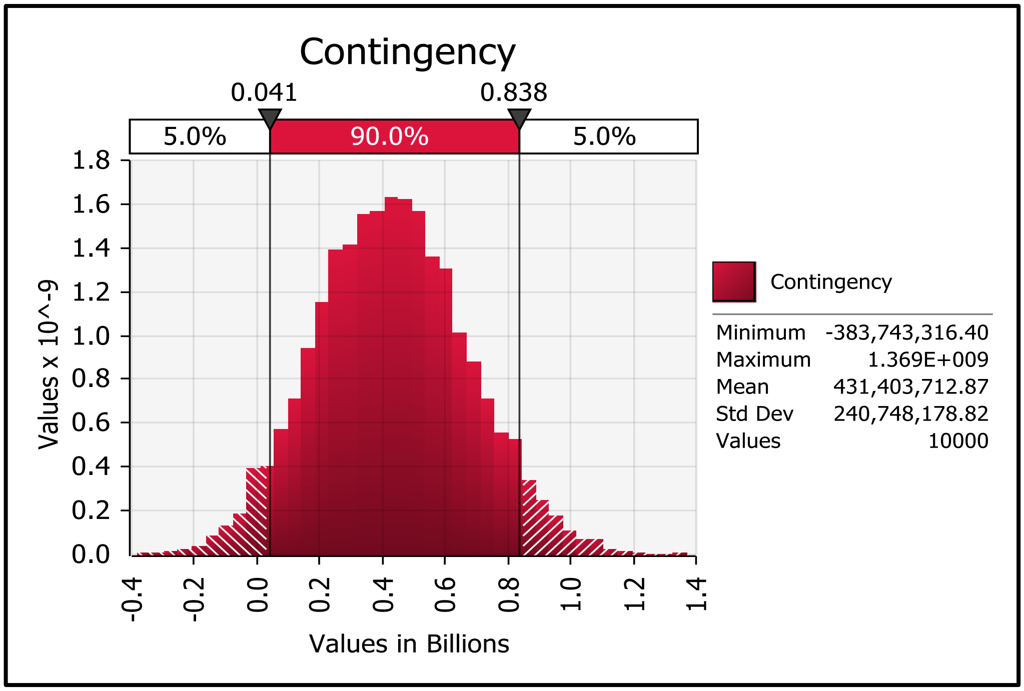
A Monte Carlo risk analysis was developed using range-estimating methods to develop contingency percentages. The cost estimate used the contingency at P85 amount.

Contingency excludes:

- Major scope changes, such as changes in end product specification, capacities, building sizes, or location of the asset or project
- Extraordinary events, such as major strikes and natural disasters
- Management reserves

The overall contingency produced by the Monte Carlo simulation at P85 and provisions is $586.1 million, which is 28.4% of the total direct and indirect costs, including 25.4% for contingency which is presented in Figure 21-1.

Figure 21-1: Summary Statistics for Contingency



21.1.11.1 Taxes and Duties

Taxes and duties are not included in the estimate.

21.1.11.2 Escalation

No allowance for escalation beyond fourth quarter of 2020.

21.1.12 Assumptions and Exclusions

Assumptions

- All equipment and materials will be purchased new
- Build-up of craft labour benefits and burdens is based on local Union Collective Agreements at the time of estimate
- Contracts awarded for construction, equipment, and engineering will be based on competitive bid

Exclusion

The following items have been excluded from this Capex:

- Working or deferred capital
- Financing costs

- Refundable taxes and duties
- Land acquisition
- Currency fluctuations
- Lost time due to severe weather conditions
- Lost time due to force majeure
- Lost time due to pandemic
- Additional costs for accelerated or decelerated deliveries of equipment, materials, or services resultant from a change in project schedule
- Warehouse inventories, other than those identified and supplied in initial fills
- Any project sunk costs (studies, exploration programs, etc.)
- Mine reclamation costs (included in financial model)
- Mine closure costs (included in financial model)
- Allowance for work space restriction
- Decommissioning and demolition of project permanent facilities at the end of mine life
- Escalation costs
- Community relations

21.2 Operating Cost Estimate

21.2.1 Summary

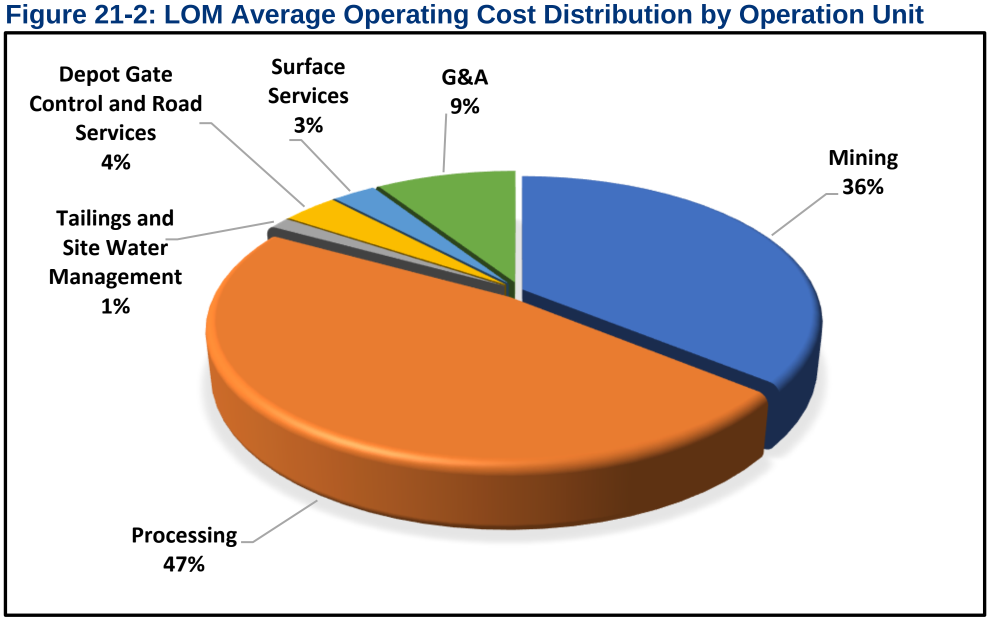
The operating cost estimate for the Project consists of mining, processing, G&A, surface services, gate services (gate control, internal road maintenance and concentrate hauling), and tailings and site water management costs, which are summarized in Table 21-4 and Figure 21-2. The total LOM average operating cost is estimated to be USD$8.66/t mill feed processed. The operating cost estimate is expressed in US dollars, unless specified.

Table 21-4: LOM Average Operating Cost Summary* (per tonne processed)

Function	Operating Cost (USD$/t processed)
Mining	3.11
Processing	4.08
Tailings and Site Water Management	0.11
Gate Control and Road Services**	0.32
Surface Services	0.25
G&A	0.79
Total Cost	**8.66**

Notes: *LOM average operating, which is slightly different from the unit cost at the nominal process rate of 133,000 t/d.
**Gate control and road services include concentrate hauling from the site to the gate.

Figure 21-2: LOM Average Operating Cost Distribution by Operation Unit



21.2.2 Mining Operating Cost

The estimated mining haulage hours were the basis of the operating cost estimate. Fleet size, personnel requirements, and operating hours were derived from the haulage hours and the mine production schedule. Table 21-5 outlines the LOM average cost per tonne mined by mining activity. The LOM average mining operating cost was estimated to be $1.56/mined or $3.11/t milled.

Table 21-5: Life of Mine Average Mining Operating Cost

Area	Unit Cost, USD$/t Mined
Drill and Blast	0.25
Excavation/Loading	0.16
Hauling	0.94
Support Equipment	0.15
Supervision	0.05
Dewatering	0.01
Total	**1.56**

21.2.3 Mill Operating Cost

The average annual operating cost for the nominal process rate of 133,000 t/d is estimated to be $198.4 million per year, or $4.08/t of mill feed. This estimate includes:

- Staffing and salary/wage level estimates, based on the 2020 Q4 mining industry labour rates in BC.
- Power consumption estimates, based on equipment load list power draw estimates, including power requirements for ball and regrind mills estimated by the Bond work index equation and power requirements for SAG mills estimated from the JK SimMet SAG mill grinding circuit simulations.
- Unit power cost estimate of $0.039/kWh based on forecasted BC Hydro power unit. The cost excludes the power demand charge, which is included separately in G&A cost estimates.
- Major grinding steel consumables estimates, based on the abrasion index for steel consumption, the estimated prices for mill liners and the unit prices from quotations for grinding steel balls in Q3/Q4 2020.
- Reagent consumable estimates, based on laboratory optimum dosages determined by locked cycle tests from the test programs, and the quoted prices for reagents in Q3/Q4 2020.
- Operating and maintenance supply cost estimates, based on approximately 5% to 7% of major equipment capital costs, benchmarked against comparable operations in BC and worldwide.

No taxes have been accounted in the estimates, unless specified.

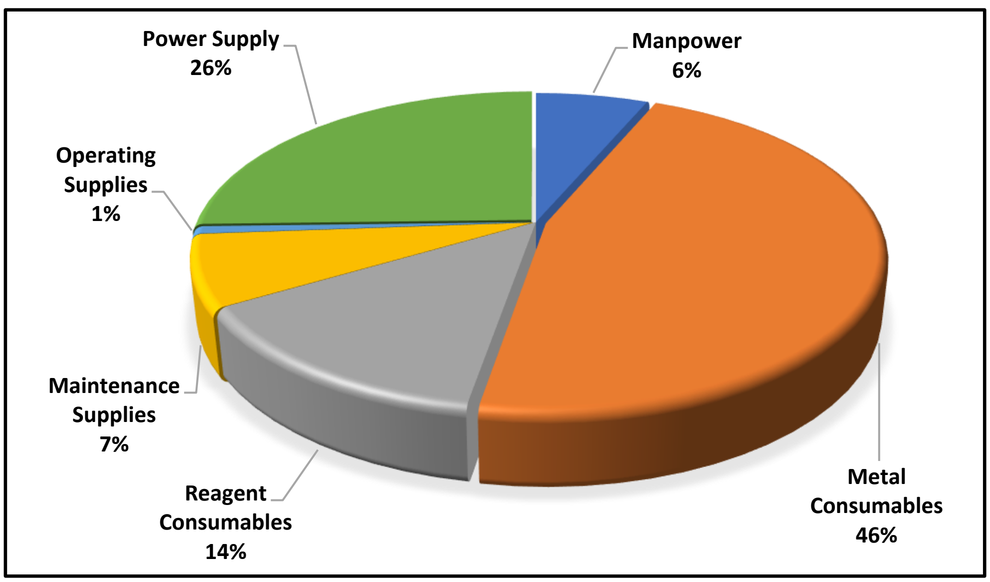
Table 21-6 Table 21-6: summarizes the estimated processing costs which include molybdenum concentrate production and assay costs for mining and geological samples. Figure 21-3 shows the average operating cost distribution by area.

Table 21-6: Summary of Processing Costs (including Mo Recovery Circuit)

Description	Manpower	Annual Cost ($)	Unit Cost ($/t milled)
Manpower			
Subtotal Manpower	**155**	**12,816,000**	**0.26**
Major Consumables			
Metal Consumables	-	91,750,000	1.89
Reagent Consumables	-	27,387,000	0.56
Subtotal Major Consumables	**-**	**119,137,000**	**2.45**
Supplies			
Maintenance Supplies	-	14,541,000	0.30
Operating Supplies	-	1,537,000	0.03
Power Supply	-	50,351,000	1.04
Subtotal Supplies	**-**	**66,429,000**	**1.37**
Total	**155**	**198,382,000**	**4.08***

Note: Rounded to the nearest cent.

Figure 21-3: Average Operating Cost Distribution by Area



Power and steel consumable costs constitute approximately 72% of total processing costs. The cost estimates are detailed in the following section.

Process Operating Cost – Bulk Copper-Molybdenum Flotation

Labour Cost

Total process labour costs at the nominal process rate of 133,000 t/d are estimated to be $11.9 million per year, or $0.25/t of mill feed. Salary estimates are based on 2020 labour rates for the 2020 mining industry labour rates in BC, Canada. The processing plant will be staffed with 143 personnel including 12 in general supervisory and technical services, 58 in operational roles, 57 in maintenance roles, and 16 for the metallurgical lab and assay lab.

Loaded salary includes base salary and burden, which is estimated to be 40% of the base payment. The burden includes holiday and vacation payments, overtime payments, pension plan, Medical Services Plan, Canada Pension Plan, Employment Insurance, Workers' Compensation Board insurance and tool allowance costs.

Power Cost

The average annual power consumption, based on the nominal processing rate of 133,000 t/d of mill feed, is estimated to be 1,276,000 MWh/a. At an average power unit cost of $0.039/kWh, the annual power cost is estimated to be $50.1 million, or $1.03/t of mill feed.

Maintenance and Operating Supplies Cost

Maintenance and operating supplies, excluding molybdenum circuit operating costs, are estimated to cost $15.8 million per year, or $0.33/t of mill feed. Maintenance supplies are estimated to cost $14.3 million per year or $0.30/t of mill feed. Operating supplies are estimated to be $1.5 million per year, or $0.03/t of mill feed.

Major Consumable Costs

Major consumable costs excluding molybdenum circuit operating costs, are estimated to be $114.7 million per year at the nominal processing rate of 133,000 t/d, or $2.36/t of mill feed.

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

- Consumption rates for primary crusher and pebble crusher liners are based on the data provided by the potential crusher suppliers, or in-house data.
- Consumption rates for primary grinding mill liners is based on the Bond abrasion index equation, using the deposit's average abrasion index, and the estimated equipment power consumption. An alloy quality factor has been applied to reflect improved liner quality and comparable operations.
- Regrinding mills, i.e., tower mills, are lined with magnetic liners, thus the tower mill shell liner consumption is not reflected in the estimate. Instead, the tower mill screw liner consumption is shown in the estimate.
- Consumption rates for grinding media are based on the Bond abrasion index equation, using the average abrasion index of the deposit, and the estimated equipment power consumption. Considering the experience of similar operations and taking into account improved steel quality, grinding media quality factors have been applied to the estimate. Primary grinding and regrinding steel ball prices are based on the quotations from potential suppliers. The estimated unit operating cost for the steel ball consumptions is $1.48/t milled.

- Consumption rates for flotation reagents are based on the optimum dosages as determined by locked cycle tests from testing programs. The prices for the reagents are based on a similar operating data or the quotations from potential suppliers. The estimated unit operating cost for the reagent consumptions is $0.48/t milled.
- No taxes have been accounted in the estimates, unless specified.

21.2.3.1 Process Operating Cost – Molybdenum Flotation and Leaching

Table 21-7 provides a summary of estimated processing costs for molybdenum separation flotation and molybdenum concentrate leaching circuits. At the nominal process rate of 133,000 t/d, the operating cost is estimated to be $0.12/t of mill feed.

Table 21-7: Operating Costs – Molybdenum Flotation and Leaching

Description	Manpower	Annual Cost ($)	Unit Cost ($/t milled)
Manpower			
Subtotal Manpower	**12**	**880,000**	**0.018**
Major Consumables			
Metal Consumables	-	479,000	0.010
Reagent Consumables	-	3,917,000	0.081
Subtotal Consumables	**-**	**4,396,000**	**0.091**
Supplies			
Maintenance Supplies	-	231,000	0.005
Operating Supplies	-	19,000	0.001
Power Supply	-	298,000	0.006
Subtotal Supplies	**-**	**548,000**	**0.012**
Total	**12**	**5,824,000**	**0.121**

Labour Cost

Process labour costs for molybdenum flotation and leaching circuits are estimated to be $0.88 million per year, or $0.018/t of mill feed at the nominal process rate of 133,000 t/d. Salary estimates are based on 2020 labour rates for the 2020 mining industry labour rates in BC, Canada. The payment has included base salary and burden, which is estimated to be 40% of the base payment. The molybdenum flotation and molybdenum concentrate leaching circuits will be staffed with 12 personnel.

Power Cost

The average annual power consumption, based on the nominal processing rate of 133,000 t/d of mill feed, is estimated to be 7,592 MWh/a for the molybdenum flotation and leaching circuits. At an average power unit cost of $0.039/kWh, the annual power cost is estimated at $298,000, or the unit cost is $0.006/t of mill feed.

Maintenance and Operating Supply Costs

Maintenance and operating supplies for the molybdenum flotation at the copper and molybdenum separation circuit and molybdenum concentrate leaching circuits are estimated to cost $250,000/a, or $0.005/t of mill feed.

Major Consumable Costs

Major consumable costs for the molybdenum flotation and leaching circuits are estimated to be $4.4 million per year, or $0.091/t of mill feed. The metal consumables and reagent consumables operating costs are based on quotations from potential suppliers, or similar operating data.

21.2.4 General and Administrative

For the LOM average, G&A expenses are estimated at approximately $38.7 million per year, or $0.79/t milled. The costs include:

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

Manpower cost and general expenses are estimated to be approximately $0.12/t milled and $0.67/t milled respectively. The major costs are accommodation and crew air and bus transportation, estimated at approximately $10.0 million per year, and electricity power demand charge, estimated to be approximately $14.8 million per year.

21.2.5 Surface Services

The LOM average site service cost is estimated at $0.25/t milled or approximately $12.1 million per year. The estimate includes:

- personnel – general surface services manpower
- surface mobile equipment and light vehicle operations
- potable water and waste management
- general maintenance including yards, roads, fences, and building maintenance
- off-site operation expense
- building heating

- avalanche control

21.2.5.1 Gate Control and Road Maintenance and Concentrate Hauling Between Site and Access Gate

The LOM average service cost for gate control at the Highway 37 access gate and road maintenance and concentrate hauling between the site and the gate is estimated to be $0.32/t milled or approximately $15.5 million per year. The major item is the trucking cost for transporting the concentrates from the mine site to the gate.

21.2.6 Tailings and Site Water Management

The LOM average cost for tailings and site water management is estimated at $0.11/t mill feed, including $0.04/t milled for cyclone sand preparation. As estimated by Knight Piésold, the total cost for the life of the mine is approximately $40.3 million.

22.0 ECONOMIC ANALYSIS

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

22.1 Introduction

The economic analysis of Schaft Creek has been derived from the inputs described in previous chapters. The economic analysis has been performed on a 100% basis using Q4 2020 US dollars, and its unlevered post-tax FCF has been discounted using mid year discounting at a rate of 8% per annum. NSR, capital, operating, sustaining and closure costs, NPI payments, BC Mineral tax, and Federal and Provincial income taxes are included in the financial analysis.

For the 21-year mine life and 1.03 Bt mill feed tonnage and the metal prices and foreign exchange rate shown in Table 22-1 (base case), the following investment returns and select financial metrics were calculated:

- Pre-tax IRR of 15.2%
- Post-tax IRR of 12.9%
- Pre-tax NPV of USD$1,383 million at an 8% discount rate
- Post-tax NPV of USD$842 million at an 8% discount rate
- 4.4-year payback period for pre-tax estimate and 4.8-year payback period for post-tax estimate

22.2 Inputs/Assumptions to the Preliminary Financial Analysis

The inputs to the preliminary economical assessment include:

- mine production schedule, including dilution and waste rock handling
- metal recovery rates to develop the annual recovered metals based on metallurgical test work results
- initial capital expenditures, including required construction and development before commercial production, including mining development and overall project construction expenditures as specified in Section 21.0
- Opex, including the costs for mining, processing, tailings management, overall site services, and general administration
- off-site expenditures, such as applicable treatment, refining, transportation, and insurance costs (the offsite charges are detailed in Section 22.3)

- sustaining capital costs on a year-by-year basis over the LOM, including the USD$154 million mine closure and reclamation costs distributed over the LOM
- metal prices and foreign exchange rates as shown in Table 22-1
- net proceeds interests & royalties detailed in Section 22.4

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

Metal price and foreign exchange rate inputs to the financial model are shown below in Table 22-1. Prices and exchange rate shown are flat across all years.

Table 22-1: Metal Pricing and USD/CAD Exchange Rate Inputs

Copper	USD$/lb	3.25
Gold	USD$/oz	1,500
Molybdenum	USD$/lb	10.00
Silver	USD$/oz	20.00
CAD/USD Exchange	-	1.30

The financial analysis does not include sunk costs and any costs associated with financing. In addition, all the cash flows in the financial model were based on Q4 2020 US dollars without any adjustment for inflation.

The years used for the preliminary financial analysis are for illustrative purposes only and do not necessarily represent a commitment to the start dates or actual production.

22.3 Assumptions on Treatment Charges/Refining Charges and Concentrate Transport Costs

There is no study conducted on smelting terms. The generic smelter terms outlined in Section 19 are used for the analysis based on the in-house data as below.

22.3.1 Metal Payable and Smelting Terms

22.3.1.1 Copper Concentrate

Copper payables:

- At concentrate grades less than 28.0% / dmt, the net payable is the grade less an absolute deduction of 1.1%.
- Between 28.0% and 32.0%, the net payable is the grade less an absolute deduction of 1.0%.
- Between 32.0% and 38.0%, the net payable is 96.65% of the grade.

Gold payables:

- The payment for the gold in the concentrate is based on the payment schedule shown in Table 22-2.

Table 22-2: Gold Payables

Ceiling (g/dmt)	1	3	5	8	10	15	50	>50
Payable as % of grade:	0.00%	90.00%	92.00%	95.00%	96.00%	97.00%	97.50%	98.00%

Silver payables:

Silver payables are as follows:

- At concentrate grades less than 30 grams / dmt, the payable is 0%.
- At concentrate grades equal to or above 30 grams / dmt, the payable is 90%.

Copper Concentrate Treatment Charges

- For the copper concentrate, treatment costs are assumed to be USD$90/dmt concentrate. Refining costs net of payables are USD$0.09/lb payable copper, USD$5.00/oz payable gold, and USD$0.50/oz payable silver.

22.3.1.2 Molybdenum Concentrate

Molybdenum payables are 99.0% of the molybdenum concentrate grade (per dmt). For molybdenum concentrate, treatment and refining costs net of payable are assumed to be USD$2.50/dmt concentrate and USD$1.52/lb molybdenum, respectively.

22.3.2 Transportation Cost and Other Cost Estimates

The estimated transportation costs for the copper and molybdenum concentrates and the costs related to marketing and insurance are as follows in Table 22-3.

Table 22-3: Transportation Assumptions

		Copper Concentrate	Molybdenum Concentrate
Concentrate Moisture Content	%	9.00%	5.00%
Concentrate Freight	USD$/wet t	76.67	154.05
Marketing & Others	USD$/wet t	0.50	0.50
Insurance	% of Net Value	0.15%	0.15%
Concentrate Loss	% of Net Value	0.08	0.08

22.4 Royalty

The Project is subject to two separate NPI payments. Royal Gold holds a 3.5% NPI on certain mineral claims within the resource area. Based on the PEA, the LOM NPI payments to Royal Gold are estimated to be approximately USD$258.5 million.

The Schaft Creek JV has an obligation to Liard in the form of a 30% NPI in certain mineral claims within the Project (the "Indirect Interest"). Liard is owned 85.5% by the Schaft Creek JV, 1.55% by Copper Fox, with the remaining 12.95% held by third parties. Based on the PEA, the LOM NPI payments to Copper Fox and Liard minority interests were estimated to be approximately $334.7 million.

22.5 Assumptions on Taxes

The following general tax regime was recognized as applicable at the time of report writing:

22.5.1 Canadian Federal and BC Provincial Income Tax Regime

Federal and BC provincial income taxes are calculated using the currently enacted corporate rates of 15% for federal and 12% for BC .

For both federal and provincial income tax purposes, capital expenditures are accumulated in pools that can be deducted against mine income at different rates, depending on the type of capital expenditure.

Mineral exploration costs are accumulated in the Canadian Exploration Expense (CEE) pool. The CEE pool is generally amortized at 100%, to the extent of taxable income from the mine.

Resource property acquisition costs and most pre-production mine development costs are accumulated in the Canadian Development Expense (CDE) pool. The CDE pool is generally amortized against income at 30% on a declining balance basis.

Fixed assets used in mining operations are accumulated in an Undepreciated Capital Cost pool, Class 41. The Class 41 pool is amortized at 25% on a declining balance basis.

Over the LOM, the estimated corporate federal and provincial income taxes will be approximately USD$1,101 million and USD$881 million respectively.

22.5.2 BC Mineral Tax Regime

The BC Mineral Tax regime is a two-tier tax regime, with a 2% tax and a 13% tax.

The 2% tax is assessed on "net current proceeds", which is defined as gross revenue from the mine less mine Opex. Hedging income and losses, royalties and financing costs are excluded from Opex. The 2% tax is accumulated in a Cumulative Tax Credit Account (CTCA) and is fully creditable against the 13% tax.

All capital expenditures, both mine development costs and fixed asset purchases, are accumulated in the Cumulative Expenditures Account, which is amortized at 100% against the 13% tax.

The 13% tax is assessed on "net revenue", which is defined as gross revenue from the mine, less mine Opex, less any accumulated cumulative expenditures account balance. As such, the 13% tax is not assessed until all pre-production capital expenditures have been amortized.

A "new mine allowance" is provided for capital costs of new mines to encourage mine investment in BC. This allowance provides that 133% of capital expenditures incurred prior to commencement of production are included in the cumulative expenditures account. Under current legislation, the provision for the new mine allowance applies to mines that begin producing minerals before December 31, 2025. The post-tax model is calculated on the assumption that the mine allowance will apply.

Notional interest of 125% of the prevailing federal bank rate is calculated annually on any unused cumulative expenditures account and CTCA balances and is added to these pools.

BC Mineral Tax is deductible for federal and provincial income tax purposes.

The estimated BC mineral tax will be approximately USD$922 million during the 21-year mine life.

22.6 Financial Model Summary

22.6.1 Financial Evaluations: NPV and IRR

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

Metal revenues are derived from copper, gold, molybdenum, and silver sales. Operating cost for mining, processing, site services, G&A, tailing storage and handling, as well as off-site charges (smelting, refining, transportation) were deducted from the revenues to derive annual operating cash flow.

Initial and sustaining capital costs as well as closure and reclamation costs have been incorporated on an annual basis over the mine life and deducted from the operating cash flow to determine the net cash flow before taxes. Initial capital expenditures include costs accumulated prior to first production of concentrate, including all pre-production mining costs. Sustaining capital includes expenditures for mining and processing additions, replacement of equipment, and TSF expansions.

Initial and sustaining capital costs applied in the economic analysis are USD$2.653 billion and USD$0.848 billion, respectively. Pre-production construction period is estimated to be five years. NPV and IRR are estimated at the start of this five-year period.

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

The post-tax NPV at an 8% discount rate was estimated to be USD$842 million after royalties. The post-tax IRR was estimated to be 12.9%. The initial capital investment post-tax payback year is expected to 4.8 years after the start of production. Table 22-4 shows the cash flow and the key economic parameters.

Table 22-4: Financial Model Summary

Description	Value	Units
Metal Price		
Copper	$3.25	USD$/lb
Gold	$1,500	USD$/oz
Silver	$20.00	USD$/oz
Molybdenum	$10.00	USD$/lb
Currency Exchange Rate		
USD$:CAD$	1.00:1.30	
Financial Analysis Summary		
LOM	21	years
Tonnes Mined Including Waste Rock	2,073,623	kt
Tonnes Processed	1,030,207	kt
Annual Tonnage Processed	49,057	kt
Tonnes Concentrate Produced (Dry Mass) Copper	8,091	kt
Copper Recovered to Concentrate	4,994,616	klb
Gold Recovered to Concentrate	3,695	koz
Silver Recovered to Concentrate	16,412	koz
Tonnes Concentrate Produced (Dry Mass) Molybdenum	205	kt
Molybdenum Recovered To Concentrate	226,457	klb
Offsite Smelting, Transportation and Other Costs	$2,296.27	USD$ million
NPI	$593.1	USD$ million
Net Revenue from Sales	$21,250	USD$ million
Average Net Smelter Return per Tonne	$20.63	USD$/t
Operating Costs per Tonne Processed (On-Site)		
Mining	$1.55	USD$/t mined
Mining	$3.11	USD$/t processed
Processing	$4.08	USD$/t processed
General and Administrative	$0.79	USD$/t processed
Surface Service	$0.25	USD$/t processed
Tailings Management	$0.11	USD$/t processed

table continues…

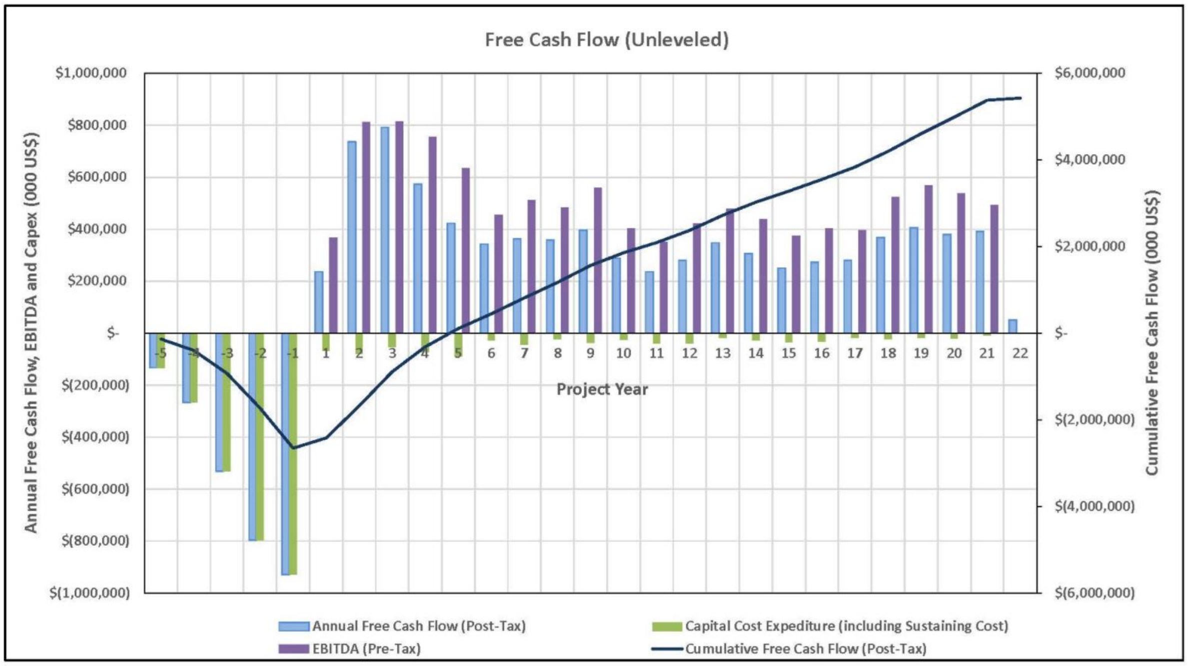
TETRA TECH

Description	Value	Units
Copper and Moly Conc Transport Cost from Site to Depot	$0.32	USD$/t processed
Total LOM operating costs	$8.66	USD$/t processed
Capital Costs		
Pre-Production Capital Costs	$2,653	USD$ million
LOM Sustaining Costs, Including	$849	USD$ million
Environmental Management and Closure Cost	*$194*	*USD$ million*
Total Life of Capital Expenditures	$3,502	USD$ million
Unit Cash Costs		
Before By-Product Credits	2.56	USD$/lb Copper
After By-Product Credits	1.00	USD$/lb Copper
All-in Sustaining Costs	1.18	USD$/lb Copper
Cash Flow		
Pre-Tax Operating Cash Flow	$7,376	USD$ millions
NPV at a Discount Rate of 8%	**$1,383**	**USD$ millions**
Pre-Tax Internal Rate of Return (IRR)	**15.2%**	
Payback Years	**4.4**	**years**
Post-Tax Operating Cash Flow	$5,395	USD$ millions
Post- NPV at a Discount Rate of 8%	**$842**	**USD$ millions**
Post-Tax Internal Rate of Return (IRR)	**12.9%**	
Payback Years	**4.8**	**years**

The Schaft Creek deposit contains low, but significant, gold, molybdenum, and silver concentrations that account for approximately 33% of the CuEq production. In the first five years at full production, the average recoverable CuEq production is estimated to be approximately 398.1 million lbs. (180.6 Kt). LOM average revenue split by commodity is approximately copper (66.6%), gold (22.7%), molybdenum (9.3%), and silver (1.3%).

The project's post-tax FCF summary is shown in Figure 22-1. FCF is well distributed over the LOM.

Figure 22-1: Post-Tax Annual and Cumulative FCFs, EBITDA, and Capex



Federal, Privincial, and BC Mineral Tax payables based on the PEA financial model are calculated and shown below in Table 22-5. The BC Mineral Tax (Provincial Resource Tax) is deductable from Federal and Provincial taxes payable.

Table 22-5: Project Cash Flow Summary

Copper Price (USD/lb)	2.75	3.00	3.25	3.50	3.75
EBITDA (USD billions)	8.88	9.85	10.81	11.78	12.75
Net Cash Flow (pre-tax USD billions)	5.45	6.41	7.38	8.34	9.31
Net Cash Flow (post-tax USD billions)	3.99	4.69	5.39	6.10	6.81
NPV @8% (pre-tax USD billions)	0.73	1.06	1.38	1.71	2.03
NPV @8% (post-tax USD billions)	0.36	0.60	0.84	1.08	1.32

22.7 Sensitivity Analysis

Sensitivity analysis for post tax financial parameters for both NPV at a discount rate of 8% and IRR were conducted and are presented in Figure 22-2 to Figure 22-4. The financial parameters include:

- copper price
- gold price
- exchange rate

- development capital expenditure
- operating costs

Figure 22-2: Post-tax NPV Sensitivity

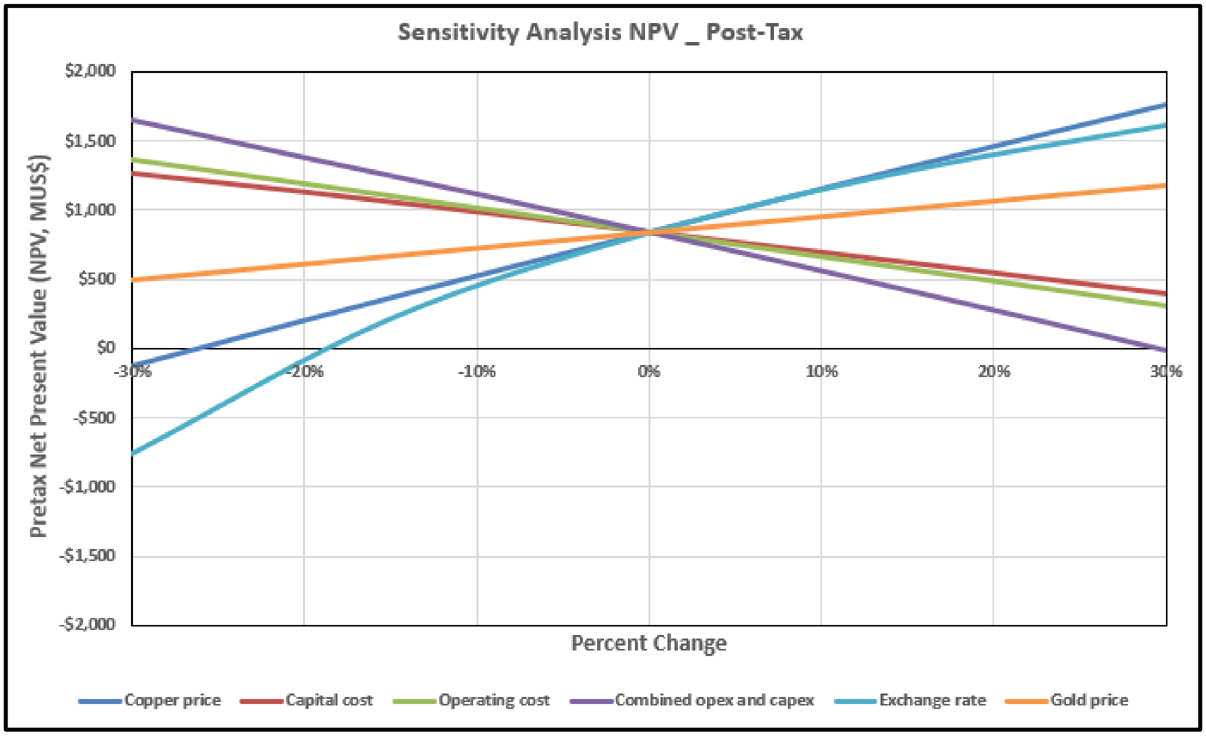


Figure 22-3: Post-tax IRR Sensitivity

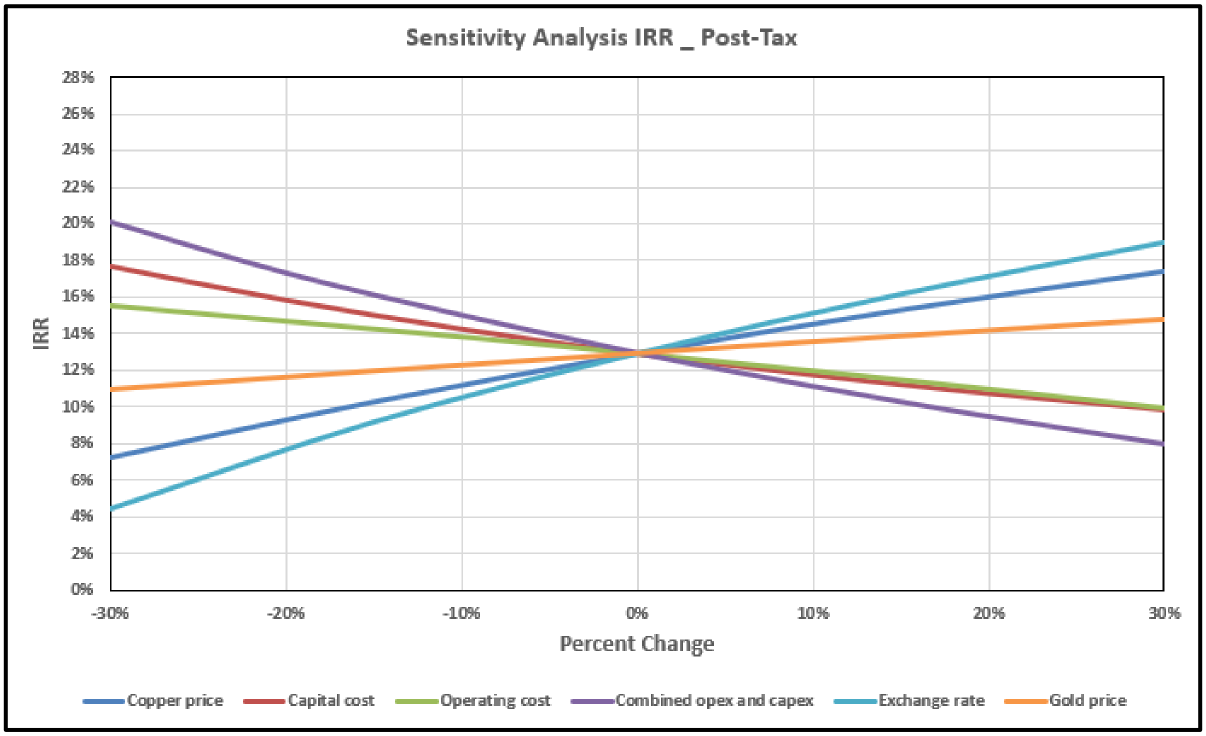


Figure 22-4: The Post-Tax Sensitivity of the EBITDA, FCF, and NPV Based on Incremental Changes in Metal Prices, Exchange Rate (CAD/USD), Opex, and Initial Capex (Based on 8% Discount Rate)

	1st 5 years EBITDA Base Case US$695M	1st 5 years After-Tax FCF Base Case US$574M	After-Tax NPV Base Case US$842M
CADUSD *(+/- 10 points)*	662; (5%) / 724; 4%	558; (3%) / 587; 2%	566; (34%) / 1,098; 29%
Copper Price *(+/- $0.25/lb)*	636; (9%) / 755; 9%	532; (7%) / 615; 7%	613; (28%) / 1,094; 28%
Opex *(+/- 5%)*	675; (3%) / 715; 3%	560; (2%) / 587; 2%	766; (10%) / 943; 10%
Gold *(+/- $100/oz)*	674; (3%) / 717; 3%	558; (3%) / 589; 3%	778; (9%) / 931; 9%
Initial Capex *(+/- 5%)*	695; 0% / 695; 0%	567; (1%) / 580; 1%	781; (9%) / 928; 9%
Molybdenum *(+/- $1.00/lb)*	687; (1%) / 704; 1%	568; (1%) / 580; 1%	814; (5%) / 895; 5%
Silver *(+/- $1/oz)*	695; (0%) / 696; 0%	573; (0%) / 574; 0%	852; (0%) / 857; 0%

Relative to these variables, the project post-tax NPV at a discount rate of 8% is more sensitive to copper pricing and USD/CAD exchange rate, followed by changes in gold pricing and operating and capital costs.

The project post-tax IRR shows a similar sensitivity pattern to these variables, except that it is most sensitive to USD/CAD exchange rate, followed by copper pricing.

Federal, Privincial, and BC Mineral Tax payables based on the PEA financial model are calculated and shown below in Table 22-5. The BC Mineral Tax (Provincial Resource Tax) is deductable from Federal and Provincial taxes payable.

Table 22-5 shows the incremental effect on financial parameters based on an incremental USD$0.25/lb increase in copper price.

23.0 ADJACENT PROPERTIES

There are no material adjacent properties.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Introduction

The Project Execution Strategy (PES) describes the strategy for completing all required engineering, procurement, construction, and commissioning, including mine, tailings, and environmental activities. The PES also ensures that the core elements of Schaft Creek JV's Sustainable Development Model, regarding the inter-relationship between the project stakeholders, community, and Indigenous groups, are maintained from the mineral exploration stage through construction and operations stages.

24.2 Health, Safety, Environmental, and Security

Health, Safety, and Environmental (HSE) programs and initiatives are essential to project success. These programs will be in accordance with conditions of the provincial and federal EA certificates, the provincial Mines Act Permit, and other regulatory approvals as they are obtained. A fully-integrated program will be implemented to achieve a "zero-harm" goal. To achieve this, key project stakeholders will be asked to share in the responsibility by providing the leadership and commitment to attain the highest HSE standards and values. A high level of communication, motivation, and involvement will be required in the development of HSE practices, including alignment with site contractors on topics such as safety training, occupational health and hygiene, hazard and risk awareness, safe systems of work, and job safety analysis. Tools will be implemented for performance tracking and accountability, including procedures for incident management.

All contractors will be required to pay particular attention to construction safety, and to provide individual safety programs and safety plans to the meet or exceed Schaft Creek JV's HSE Management System requirements.

All design and engineering stages incorporate criteria for responsible management of process flows, effluent and waste products to meet established capture and containment guidelines. The design also incorporates basic clean plant design standards, including operational safety and maintenance access requirements. A Hazard and Operability Analysis (HAZOP) will be conducted by the project design team during the detailed design stage for each area of the plant. This systematic team approach will identify hazards associated with operability that require attention in order to eliminate undesirable consequences. Environmental protection will be incorporated in the design of the main processes of the plant, as well as in the transportation, storage, and disposal of materials within and outside the boundaries of the plant.

Safety tagging and lockout procedures will be established in order to identify the status of equipment and systems, and to identify equipment "ownership" as the commissioning process moves from completion of construction to mechanical completion.

Schaft Creek JV will provide a well-equipped first-aid facility, ambulance, and fire engine for project-wide use. The first aid facility will be staffed with medical practitioners and nurses who will be available 24 hours per day to ensure continuous coverage. The first-aid staff will live at the camp. Contractors will be expected to provide basic first aid-stations for their workers at the site. The main first aid station will be located by the truck shop and three satellite first aid stations will be located by the primary crusher, the process plant, and the camp.

Schaft Creek JV will supply a 24-hour staffed site security program during initial field mobilization. Access to the site will be controlled at the principal road entrance at Highway 37, where the gatehouse will be constructed. All personnel required to be at site must complete safety induction training; no personnel will be allowed at site without this training.

24.2.1 Execution Strategy

The execution strategy for the successful monitoring and control of the Project will be based on traditional methods used for industrial development, combined with Schaft Creek JV's sustainable development model. The following establishes the framework for the execution of the Project during the engineering, procurement, construction and commissioning/start-up phases, and then the operations phases.

24.2.1.1 Management Procedures

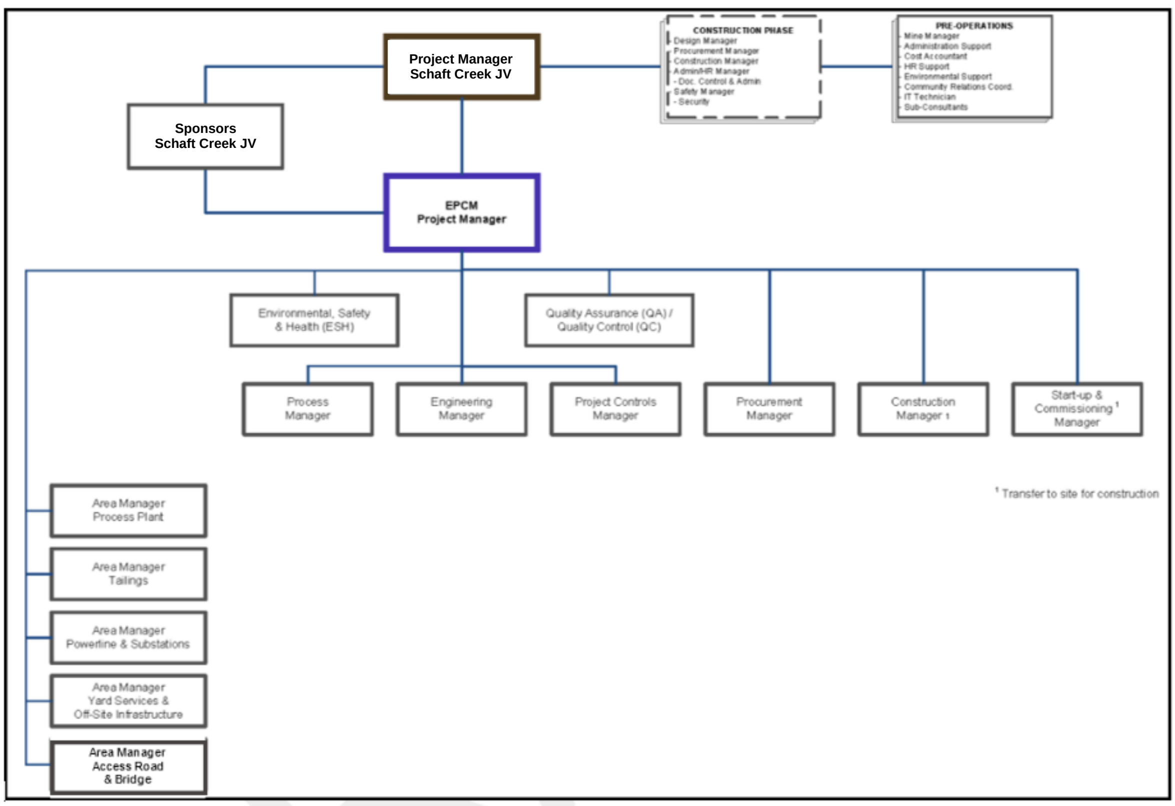
Although this section refers to the EPCM consultant as a single entity, Schaft Creek JV may issue separate contracts for engineering and procurement services, and construction management services.

The team will combine the experience of Schaft Creek JV personnel, along with engineering and construction managers who will be responsible for using the PES to complete the Project on time and within budget. Figure 24-1 shows the EPCM team organization plan. The EPCM group, in conjunction with Schaft Creek JV, will develop comprehensive EPCM/EPC procedures (the Procedures) that will establish the structures, procedures, and requirements for the execution and administration of the Project. The Procedures will be described in a Project Procedures Manual and will outline:

- project organization
- communication matrix
- responsibility matrix
- reporting requirements
- data management
- document control
- drawing and specification preparation including numbering protocols, levels of issue, and transmittal procedures
- equipment and materials procurement procedures
- project scheduling requirements, tools, formats, and issue times
- project accounting methods including the cost reporting and forecasting systems
- construction contract procedures including bidding and awarding the work
- site administration procedures including camp administration rules

- site security
- field engineering
- safety procedures
- quality assurance expectations
- site and office personnel rules and regulations
- emergency site procedures and contact information
- construction temporary facilities (power, water, offices, and camp)
- site housekeeping and hazardous waste management
- mechanical completion expectations including lock-out procedures
- commissioning procedures outline
- project close-out and hand-over procedures
- media relations
- other administrative matters and issues specific to the Project

Figure 24-1: Project Management Organization Chart



24.2.1.2 Risk Management

A Risk Register will be created during the engineering phase and will be further reviewed and updated throughout the procurement, construction, and commissioning stages. These reviews will identify the relevant risks and/or opportunities associated with the Project, assess them against outcome objectives, and then determine the best way to eliminate/control the risks or take advantage of opportunities.

24.2.1.3 Management of Engineering Deliverables

The various engineering groups will be responsible for identifying and scheduling their deliverables in accordance with the overall Project Schedule (the Schedule). Effective document management practices will be crucial to the Project's successful implementation. A collaborative document control system will be implemented that provides status and version control for all issued documents. The system will be capable of publishing documents, text, drawings, photographs, or 3D models to the internet. These documents, particularly vendor drawings and manuals, will be linked to the equipment database in order to have an organized, accessible control system that can be turned over to the operations group at the completion of the design and construction phases.

24.2.1.4 Project Scheduling and Progress Reporting

The overall Schedule identifies the preferred critical sequences and target milestone dates that must be managed to successfully execute the Project. While executive-level reports will provide an overview of the Project's status and forecasts, the detailed schedules will track the planned and actual progress throughout the duration of the Project, using information provided by the engineering groups, contractors, vendors, the field management staff and Schaft Creek JV.

Activities related to engineering deliverables are based on information derived from the engineers' internal work plans. Information pertaining to only "Issued for Tender" and "Issued for Construction" drawings will be transferred to the Schedule. These elements form the basic requirements for the development of the various contract scopes and form of tenders.

Activities related to procurement of capital equipment are derived from the engineer's equipment list (i.e., all items that have an equipment number), as well as bulk materials—such as piping, pipe racks, electrical cable, cable tray, high-bay lighting, instrumentation, structural steel—that may be purchased by Schaft Creek JV. Expediting activities related to equipment and bulk materials will be undertaken by the construction management personnel, while the engineers will expedite the capital equipment shop drawings required to complete the design work.

For activities related to construction, the contracting plan (including the assignment of work scopes to contract packages) will be based on the timing of the engineering deliverables and the stated deliveries of the capital equipment and bulk materials. Ongoing project scheduling will involve input from all project groups, including Schaft Creek JV's operating personnel. Individual contractor schedules will be based on the overall schedule; each contractor will be expected to provide a two- or three-week look-ahead schedule that will establish smaller windows of activities. Critical sequences in particular will be micro-scheduled as needed.

Project engineering progress reporting will be based on budget versus actual manhours used. Construction progress reporting will be based on earned versus budget manhours where available, or, alternatively, weighted value progress measurement in-place work.

Following Schaft Creek JV's review and securing financing period, the Schedule will be implemented. Detailed engineering activities for the plant and infrastructure will begin immediately with main emphasis on the design of long-lead capital equipment items such as shovels, trucks, primary crushers, SAG mills, ball mills, vertimills, pebble crushers, concentrate filter, airport navigation system, power line, major transformers, main substation equipment, and other long delivery electrical equipment.

The following basic project tasks will require attention as early as possible after the Project has been approved, in order to ensure planning certainty and to maintain a proper monitoring program for all long-lead items and engineering deliverables:

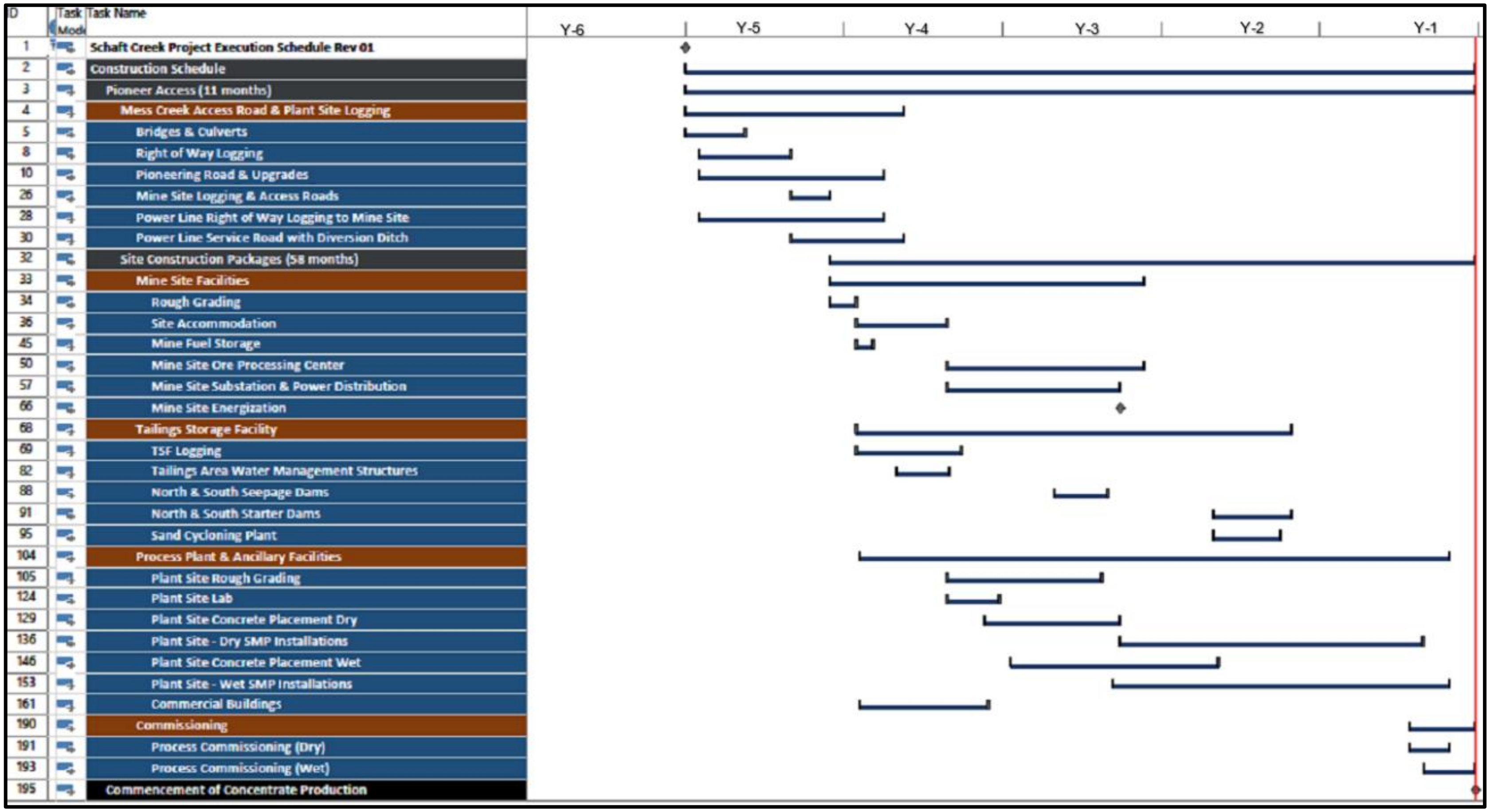
- confirmation of the project delivery method and selection of EPCM contractor(s)
- long-delivery capital equipment ordering
- establishment of the cost reporting system based on the approved Capex
- verify the Schedule
- establishment of the Procedures
- establish the logistics procedures consolidation points, rail head, trucking, and receiving port location
- establishment of standardized purchasing and contract forms

The overall project execution period is expected to be 60 months after completion of the Galore Creek and Mess Creek roads. The period between field construction and mechanical completion of Phase 1 is expected to span 54 months, and Phase 2 is expected to be completed 6 months after Phase 1. The 60-month project execution duration assumes that financing and all permits will be in place to allow all phases of the Project to proceed at the projected start times.

A summarized Schedule is provided in Figure 24-2.

Figure 24-2: Summarized Conceptual Construction Schedule (after Completion of Galore Creek Road and More Canyon Bridge)

(Insert Hi-Res PDF version in final PEA report in PDF)



24.2.1.5 Cost Reporting and Forecasting

A project WBS will be developed to define the cost elements of the project scope that will be monitored and controlled on an ongoing basis. This structure will be used for the assignment of cost codes to invoices for the Project. Once established, the cost report will become the governing cost reporting document for the Project and the Capex becomes a reference document.

A definitive cost estimate will be prepared when the engineering work is approximately 30% complete. This revised estimate will become the ongoing cost control budget, and the cost report will be modified accordingly. The definitive estimate is intended to provide a capital control budget for the Project to within ±10% of anticipated final costs.

Cost forecasting identifies deviations from the capital control budget as they occur, allowing the team to quickly evaluate and minimize cost overruns. A project cost management system will provide comprehensive cost reporting, forecasting, and trending and requires the integration of trend forecasting personnel within the engineering and construction teams. The team will implement a coordinated program to review the project cost report at regular intervals.

A cash flow has been established for the Project and will be modified as the Project progresses and scheduling and other conditions change. Short-term cash requirements will be forecast as accurately as possible when required to release money into the Project for payment of project associated invoices.

24.3 Engineering

24.3.1 Engineering Strategy

Primary engineering work will include the following categories:

- water, waste, and tailings design
- open pit mine design
- process and ancillary facilities
- site development and infrastructure
- main access road
- More Canyon Bridge
- transmission line
- diesel fuel storage

24.3.1.1 Basic Engineering

Basic engineering tasks will include the following:

- Review and update the current open pit mine design and finalize blasting design.
- Process and Utility Flow Diagrams (PFDs) will be updated and finalized to include major equipment vendor information. PFDs will also be rebalanced for all existing and new process equipment.
- The equipment list will be updated to include specific information from the firm price bids for each selected vendor.
- Piping and Instrumentation Diagrams (P&IDs) will be updated to include major equipment vendor information and preliminary control logic will be incorporated.
- The development of a plant control philosophy and methodology (full control or semi-automated control).
- A central control room will be located in the process plant.
- The preliminary instrument list will be updated based on the P&IDs.
- The general arrangement drawings for new facilities will be advanced to incorporate certified vendor information as received. Secondary equipment, including main pipe headers, ducting, air conditioning equipment, etc., will be incorporated into the layouts.
- Preliminary primary and secondary distribution electrical single line diagrams (SLDs) will be updated or prepared for the Project.
- Electrical load analysis and load flow reports will be completed.
- The engineering contractor will provide licensing and permitting assistance to the Owner as required.
- Specifications will be developed to obtain competitive and firm price bids for major equipment that requires a long lead for delivery, or which needs to be purchased to avoid high future escalation. All manufacturing hold points will be included where the engineer may decide that inspection is required. Bids will be technically and commercially evaluated by the EPCM or EP contractors and recommendations made to the Owner for purchasing.
- Purchase Orders (POs) for long delivery equipment will be issued by the EPCM contractor on behalf of the Owner. All long-lead delivery equipment items will include a clause for early delivery of certified vendor drawings so detail engineering will not be delayed while awaiting critical vendor information.
- Civil design for roads and rough grades for all areas will be completed.
- Once the Capex has been approved, it will be “converted” to a control budget and updated with recent cost information including the values of the released POs.
- The detailed project schedule will be developed.
- Purchasing procedures and standard documentation will be customized and implemented.
- A construction contract boiler plate will be developed for the Owner’s approval.

24.3.1.2 Detailed Engineering

Calculations, specifications, drawings, material requisitions, and other items related to detailed engineering will be completed in accordance with the engineering procedures. In addition, the detail design engineering team will complete the following tasks:

- Complete the engineering calculations, detailed drawings, and specifications associated with the construction of new facilities.
- Produce design and construction drawings based on the results of site investigations, including surveys, inspections, and commensurate with the details set out in the technical reports to as great an extent as possible.
- Develop equipment specifications for the remainder of the non-long lead equipment together with the commercial and technical analysis and recommend for purchase to the Owner. All manufacturing hold points are to be included in the specifications.
- Provide material requisitions for the purchase of bulk materials based on material take-offs for items such as electrical cables, cable trays, hi-bay lighting, and piping.
- Provide the construction technical specifications to be included in the construction contract bid packages (such as civil, concrete, structural steel, pre-engineered buildings, general mechanical, piping, electrical, and instrumentation), as well as pertinent drawings, vendor information when available, and quantities for the Form of Tender.
- Provide all technical reports that will be referred to by the contractors when bidding the construction work for items such as the geotechnical analysis, studies, and special information from vendors for installation of equipment and special requirements for handling.
- Review all vendors certified drawings.

24.4 Procurement and Contracts

24.4.1 Procurement and Expediting

The Contractor's Procurement Group (the Procurement Group) will provide capital equipment procurement services, vendor drawing expediting services, and equipment inspection services (when required). The Procurement Group will package the technical and commercial documentation and manage the bidding cycle for equipment and materials. Standard procurement terms and conditions approved for the Project will be utilized for all equipment and material POs. Suppliers will be selected based on location, quality, price, delivery, and support service.

The Procurement Group will organize bulk materials purchases, assemble contract tendering documents, establish qualified bid lists, issue tenders, analyze and recommend suitably qualified contractors to Schaft Creek JV, and prepare executed contracts for issue.

A field procurement manager will support ongoing construction needs for miscellaneous materials and services provided by Schaft Creek JV, as well assist with expediting tasks. The field procurement manager will also be responsible for the receipt, storage, and disbursement of purchased materials and equipment at the job site.

A plan and schedule for expediting equipment POs based on the project schedule and equipment list will be prepared. Expediting will be coordinated by the construction group. Third-party resources may be used to inspect equipment in various parts of the world during manufacture. The extent of expediting of individual POs will be based on the order's complexity, manufacturing cycle time, and schedule criticality. For POs requiring the highest level of expediting, a resident expediter may be placed at the supplier's facility. Expediting reports will be entered into the material control reporting system time the Procurement Group makes contact with suppliers.

24.4.1.1 Logistics

A logistics study was conducted to ensure best possible routing was selected. The Contractor will direct logistics and freight for incoming equipment and materials, to be transported by ship to the port of Vancouver and/or Kitimat, by rail to Kitwanga and/or Smithers, then by truck to the construction laydown. From there the trained pilot guide will accompany truck drivers on the private road to site. A single-point freight forwarding company will coordinate with manufacturing facilities, establish shipping points and dates, forward the shipments to the most convenient ports, and complete trans-shipments to the project site.

An experienced freight forwarder with representation in BC will function under the direction of the Contractor, who will issue timely shipment and materials status reports.

24.5 Construction

24.5.1 Construction Management

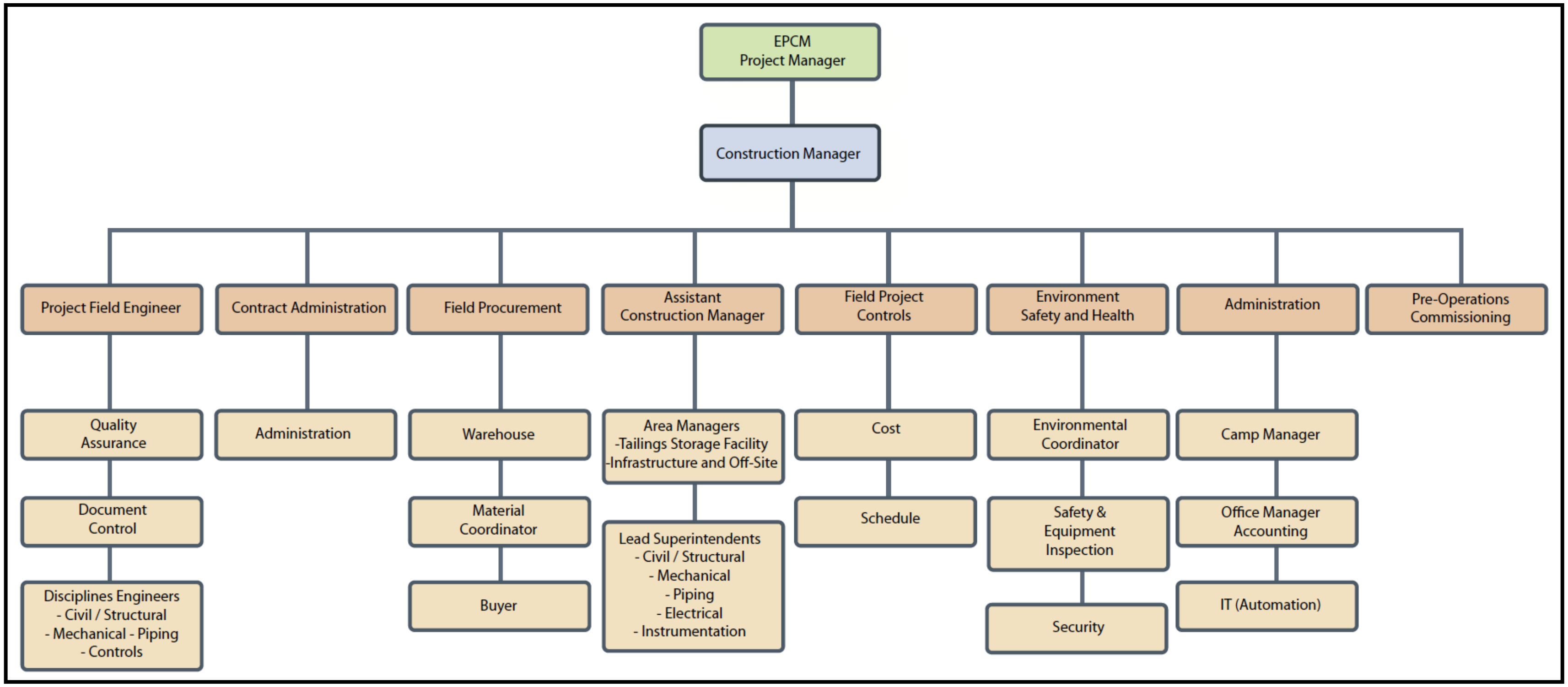
The Construction Management group will be responsible for the management of all field operations. Reporting to the Project Manager, the Construction Manager will plan, organize, and manage construction quality, safety, budget, and schedule objectives.

Construction of the Project will be performed by contractors under the direction of the Construction Management team, reporting to the Project Manager. The Construction Management key objectives are to:

- Conduct HSE policy training and enforcement for all site and contractor staff. Site hazard management tools and programs will be employed to achieve the no harm/zero accident objective.
- Apply contracting and construction infrastructure strategies to support the project execution requirements.
- Develop and implement a construction-sensitive and cost-effective master project schedule.
- Establish a project cost control system to ensure effective cost reporting, monitoring, and forecasting, as well as schedule reporting and control. A cost trending program will be instigated whereby the EPCM contractor will be responsible for evaluating costs on an ongoing basis for comparison to budget and forecasting for the cost report on monthly basis.
- Establish a field contract administration system to effectively manage, control, and coordinate the work performed by the contractors.
- Solicit tenders from the contractors and award the construction contracts to successful contractors.

- Apply an effective field constructability program, as a continuation of the constructability reviews performed in the design office.
- To develop a detailed field procurement of bulk materials, expediting, logistics, and material control plan to maintain the necessary flow and control of material and equipment to support construction operations.
- Meet the schedule for handover of the constructed plant to the Owner's team.
- The construction organization chart (Figure 24-3) shows the Construction Management team organization plan for the site.

Figure 24-3: Construction Management Organization Chart



24.5.1.1 Field Engineering

Surveying

The Construction Management survey crew will verify the accuracy of the existing control system before construction begins. Contractors will use only applicable control data for the Project. Additional monuments will be set as needed. The Construction Manager will verify surveys prior to construction. Contractors will supervise day-to-day field surveying, and the Construction Management team will provide spot checks.

Quality Control/Quality Assurance

Contractors will establish and observe their own quality control program in accordance with the construction technical specifications and the applicable codes and standards. The Construction Management Field Engineering Team will employ independent CSA-qualified quality assurance specialists to ensure quality control.

Document Control

All drawings, specifications, and other documents will be electronically transferred to the field office, logged in, updated in the master stick-file, and distributed to contractors and their supervisors.

Liaison with Detailed Engineering

The Construction Management team will be in communication with the detailed engineers on an ongoing basis to ensure that the flow of information is uninterrupted between the construction site and their office.

24.5.1.2 Commissioning and Start Up

Pre-operation Testing

Pre-operation testing is performed by a team of the personnel from engineering, construction, vendors, and the Owner's team. Comprehensive tests and inspections as defined by a quality assurance plan issued by engineers and vendors will be conducted to ensure the plant is constructed as meant to operate safely and produce the design capacity. Each system will be checked for compliance with the drawings, specifications, vendor data, lubrication, and other required tests.

Commissioning

The Owner's team will take over the custody of the plant and will start commissioning with the assistance of the engineer, the Construction Management team, and the contractors' labour to ensure that the plant is performing as was design. The Owner's team will be responsible for the training of the Owner's personnel.

Construction Strategy

Although much of the construction work may be awarded to a single general contractor or split among several contractors, the construction contracts for the TSF, access roads, More Canyon Bridge, fuel storage, substation, and high voltage power line are expected to be awarded to separate specialized contractors in Y-6. Emphasis will be on the Mess Creek and Galore Creek access roads, More Canyon

Bridge, which will provide access for all construction equipment and delivery of materials. The road is scheduled to be completed in Y-4. Early works will be developed in parallel to construction of the access road, using small equipment items that can be transported by helicopter. The final contract packaging strategy will be based on the level of available engineering details, cost advantages, scheduling issues, permits and approvals, cash flow, and availability of experienced contractors.

Table 24-1 indicates the currently-anticipated contract packaging strategy. There will be other contracts and options available during the execution phase.

Detailed engineering of the TSF, power line, main access road, and plant site bulk earthworks will be a priority. The level of design detail should be sufficient to tender and award construction contracts for these areas in Y-6. The existing exploration camp will be upgraded and used as an initial construction camp with the capacity of 100 people. The new camp facility will be installed before the start of the process facilities construction. This camp will be installed in several phases as required to accommodate the construction workforce. The construction camp will be converted into the permanent operation camp.

Table 24-1: Project Contract Packages

Contract Description	Type	Comments
Mess Creek Access Road	Unit Price	--
More Canyon Bridge	Fixed Price	Design/build
Early Works (plant site preparation, construction access to different areas, etc.)	Unit Price	Including upgrading exploration camp, clear cut for all areas, construction access roads, grub and top soil removal where required, blasting required for new airport, truck shop, and new camp area
Tailings Construction and Haul Roads	Unit Price	--
Plant Site Roads and Bridges	Unit Price	--
Bob Quinn Lake Airport Upgrade	Unit Price	Including runway extension, building terminal, instrument landing system
287 kV Power Line – Design/Build	Fixed Price	--
Diesel Fuel Storage	Fixed Price	--
Truck Fuel Station – Design/Build	Fixed Price	--
Truck Shop and Warehouse – Design/Build	Fixed Price	Pre-engineered building
Administration Complex – Design/Build	Fixed Price	Modular
Assay Lab	Fixed Price	Pre-engineered building
Cold Storage	Fixed Price	Sprung structure
Concentrate Load-out	Fixed Price	Pre-engineered building

Contract Description	Type	Comments
		table continues…
Concrete Supply	Unit Price	Site based batch plant
Concrete Installation	Unit Price	-
Structural Steel Installation	Unit Price	Could be fixed price depending on engineering detail
Cladding and Roofing	Unit Price	Could be fixed depending on engineering detail
Mechanical and Piping	Fixed Price	Will be mixed with piping unit prices
Insulation	Fixed Price	May be unit price depending on level of engineering completeness
Flotation Cells – Design/Build	Fixed Price	Dependent on engineering details
Large field erected tanks	Fixed Price	Supply and erect
Electrical and Instrumentation	Unit Price	Some parts will be fixed price
Process Control Package	Unit Price	Some parts will be fixed price
Electrical Distribution System MCC	Unit Price	Some parts will be fixed price
Main Substation	Fix Price	-
Architectural Finishes	Unit Price	Subject to changes as the Project moves ahead

The scope of construction activities scheduled to commence in Y-5 is outlined in the following sections.

24.5.1.3 Main Access Road

The main access road is from provincial Highway 37 and is 105 km long. The road has three distinct sections:

- The first section is Galore Creek road (approximately 65 km long), which is 40 km partly constructed.
- Need to complete the Galore Creek access road and More Canyon Bridge.
- The second section is Mess Creek Road (approximately 40 km long), which will start construction parallel with Galore Creek road and is expected to be completed in Y-5.
- More Canyon Bridge, which is estimated to be completed by Y-5.

The exploration camp will be upgraded to provide sufficient accommodation for workers when Early Works and construction of the Mess Creek Road will begin in Y-5; construction of this road will be completed by Y-4.

The temporary camps for access road construction will be placed in a strategic location for use as winter refuge stations for truckers and maintenance workers.

24.5.1.4 287 kV Power Line

The 287 kV power line will be connected at the BC Hydro Bob Quinn substation, which is anticipated to be completed by Y-4. The power line runs parallel to the Galore Creek and Mess Creek Roads in most sections and is also scheduled to be completed in Y-4.

24.5.1.5 Construction Laydown

This facility will have a staging area near or adjacent to Highway 37 that will facilitate the transfer of all products delivered to and from the site. This facility will provide land space for contractor to establish temporary facilities such as mobile office trailers, small repair facilities, etc.

Facilities such as explosives handling, cement receiving and handling, diesel storage, lime storage and handling, and reagents management may be operated by owner or specialty contractor teams from the outset, and certainly during operation.

24.5.1.6 Tailing Storage Facility

Specific locations within the TSF will be constructed in Y-4, involving the following construction activities.

- TSF logging, cleaning and grubbing is scheduled as the first construction activities. Sediment BMP will be implemented.
- Sediment controls will be established within the TSF using diversions, BMP, ditches, water holding ponds, and pump return systems, as required.
- The north and south dam area water control structures will be constructed using a cofferdam and core trenches.
- The north and south dam embankment will be constructed following the dewatering and core trench excavation work noted above.
- On the west side of the TSF, the haul road will be established so that mining trucks will be able to haul waste material that will be used to build the North and South Dams.
- In addition to the haul road, there will be a standard service road, including diversion ditches, the tailings piping, and the process water return piping.

24.5.1.7 Plant Site

The following construction tasks will be carried out in and around the plant site in two phases. Phase 1 will be called the Early Works Phase, which will be conducted at the same time as the Mess Creek Access Road. Phase 2 will begin after the Mess Creek Access Road is completed, so major construction and process equipment can be delivered to the site.

Phase 1 of construction will comprise the following activities:

- The existing exploration camp will be updated to accommodate 100 personnel.
- The forested area will be logged, cleared, and grubbed for all areas, including site roads.

- Starting from the existing camp, the site roads will be developed to construction level to provide access to all major areas.
- The truck shop area, camp site, and laydown areas will be roughly excavated and graded, and the surplus cut material will be used as fill material where needed.
- The plant site will be progressively blasted to the plant site elevation and will advance through the grinding area toward the flotation area and the concentrate area.
- The primary crusher foundation area will be blasted/excavated.
- The auxiliary fuel tanks will be constructed near the primary crusher and truck shop. A fuel supplier will supply the fuel for construction activities.
- Phase 2 will begin after construction of the Mess Creek Access Road is completed, which will allow large construction equipment to be delivered to the site.

Phase 2 of construction will comprise the following activities:

- All bulk earthworks will be completed after the access road will be finished.
- Detailed excavation and concrete works will start in the grinding area and follow bulk excavation closely through the flotation area and, ultimately, the concentrate area. It is currently anticipated that concrete works within the grinding area will be completed in Y-2, with the exception of internal slab on grade work.
- The construction of the grinding and flotation areas will be completed in two phases. Phase 1 will be 65,000 t/d, including all necessary supporting equipment that is common for both streams and the molybdenum concentrate system. Phase 2 will start immediately after the similar task in Phase 1 is completed. This will allow for earlier mechanical completion and commissioning/operation of Phase 1 (65,000 t/d) and troubleshoot any operation issues before commissioning.

24.5.2 Construction Labour Requirements

The construction contracting strategy and the PEA Capex are based on an open shop construction approach. The Project is not a Canadian Labour Relations Association (CLRA) union-designated project, but will involve several unions as well as non-union organizations. This approach will allow the Project to utilize local unions, Tahltan Nation and local workforce, as well as contractors from anywhere in Canada.

The Schedule is based on a 70-hour work week. Crews will rotate on a three-weeks-on-site and one-week-off-site basis.

About 5,200,000 manhours of direct and indirect construction are expected to be required, excluding mine pre-development and engineering work. The number of on-site direct construction workers will peak at about 900.

24.5.3 Construction Camp

The Early Works Phase will utilize the existing exploration camp with a capacity of up to 100 personnel. Some facilities, such as the kitchen, must be updated to accommodate this number of workers. Development of a full-service modular, single-storey, propane-heated camp for construction contractors will begin at the end of Y-4. The camp will provide single occupancy accommodations, and will include recreational facilities and a commissary. Firearms and drugs/alcohol will not be permitted at the camp.

Schaft Creek JV's operating personnel (including mining pre-stripping operators) will also use the camp. Contractors with construction crews working on the high-voltage power line and main access road will be responsible for providing their own crew camps, which will be located close to their work area, or use the existing exploration camp. These workers will not require accommodations at the mine site construction camp.

The team will manage the camp and catering activates to ensure adequate hygiene, food storage and handling, menus, and meal nutritional value.

24.5.4 Bob Quin Lake Airport Upgrade

The existing Bob Quin Lake Airport will be upgraded with a new navigation/instrument landing system and terminal. The runway will be expanded to the required length suitable for the type of aircraft. A charter airline company will provide regularly scheduled flights to and from the airport. Scheduled bus services will be provided to facilitate personnel transportation between the Airport and Mine Site.

24.5.5 Housekeeping and Hazardous Waste Management

Specific procedures for waste management and spill response will be defined in the Procedures and observed during the construction period. The Procedures will address compliance, auditing and reporting requirements, as well as procedures for ongoing cleanup and rubbish removal, safe materials handling, and storage and disposal of batteries, fuels, oil, and hazardous materials. Waste will be recycled to the extent feasible.

Ongoing dust suppression and rain water management programs will also be established and observed for the duration of the construction phase. Specific storage areas will be designated for the demolition waste prior to recycling or removal from the plant. Non-recyclable waste will be disposed of in a landfill; biologically degradable wastes will be incinerated. Used truck oil will be collected and transported to the nearest collection/recycling facility available.

24.5.6 Sewage Treatment Plant

A modular sewage treatment plant will be supplied and installed in the early stages to accommodate the Construction phase and will be located near the Construction Camp.

Sewage from other areas such as the mill site will be collected in localized sewage lift stations and trucked to the sewage treatment plant periodically as required.

The proposed sewage treatment plant uses a membrane bioreactor system with a chemical phosphorus removal process and will be designed so that its function can be reduced for the operations phase as staff requirements are diminished.

24.5.7 Construction Equipment

Construction equipment will generally be the responsibility of individual contractors. Contractor equipment safety and operability must comply with the requirements of the Mine Safety Branch. The Owner's safety personnel will perform spot checks to ensure compliance. No mobile equipment will be permitted to operate on site unless it complies with the BC Mines Act regulations and no cranes will be permitted to operate unless they have undergone a recent inspection. Equipment modifications must be certified fit for operation, particularly with respect to welding.

To reduce costs and duplication of construction equipment, the Owner may supply the large construction equipment (e.g., cranes) to be managed by the construction management team.

24.5.8 Communication

Schaft Creek JV will determine the appropriate construction-phase and permanent telecommunications technologies for the Project, with input from the team where needed. Requirements will include voice and data link technologies to support growth in both construction and plant operation needs.

The communications framework for management offices will be installed early in the construction period. The system will be supplemented with the installation of telephones in common areas, and for individual room internet access.

24.5.9 Construction Power

Approximately 2 MW of construction power will be required, together with a lower-power unit for the temporary construction period. This power will be supplied by low noise, low emission generator sets. In Y-3, permanent power line will be completed and the power will be used for all mine equipment and for the construction phase. After construction, the generator sets will supply emergency power for the duration of the mine life.

24.5.10 Mechanical Completion

"Mechanical Completion" is a term used to indicate the point when a contractor's work progress allows the Owner to operate the facility in a safe manner. Mechanical Completion may refer to individual systems (such as a building or fresh water system), and indicates only that a system can be operated. At Mechanical Completion, the Contractor and Owner's team will develop a full "punch list" of outstanding deficiencies, which is used to measure the progress to Final Completion. Because the Mechanical Completion stage often introduces a number of safety risks, procedures will be established before Mechanical Completion is reached to address safety hazards, electrical lock out procedures, and communication protocols.

Each process system or ancillary facility will be checked for compliance against drawings and specifications, vendor data, and lubrication requirements. Mechanical and electrical capital equipment will be checked for proper installation, alignment, and rotation. Conveyors will be operated without load to verify belt alignment. Tanks and piping will be water/air tested. Electrical equipment and circuits will be checked for proper installation. Instrumentation circuits will be checked and instruments will be zero-calibrated.

When all installations have been verified, each system will be operated under no-load conditions. Permanent records will be maintained for each piece of equipment.

Mechanical Completion of systems and facilities will preface overall plant commissioning. Critical utility features will have been completed before Mechanical Completion, to ensure that water will be available for piping and tank hydro testing. The air will be available for a pneumatic test, and permanent power will be available to "bump" the motors.

24.5.11 Commissioning

The Contractor will be responsible for the installation of the facilities (except for mining facilities) until they reach Mechanical Completion.

As facilities near Mechanical Completion, Schaft Creek JV will develop the commissioning plan in conjunction with the team. The systems will be identified and scheduled for delivery by priority. Once the various systems are determined by the team to be free of deficiencies that would prevent safe operation, they will be transferred to Schaft Creek JV's operations team. The operations team will consist of plant operators and maintenance staff who conduct dry-run and wet-run tests, with the assistance of vendors, contractors, and construction management personnel. Once the systems are operational, the transfer of systems will be formally documented and Schaft Creek JV will be provided with all mechanical/electrical testing documents and vendor's information.

24.5.12 Construction Methods

The Schedule is presented in Figure 24-2. The critical path runs through the access road construction, the Early Works Phase, Process Plant Phase 1, SAG and ball mills and large motors installation.

A number of the most important elements of the construction phases of the Project were examined during the study. The following sections summarize the various recommendations.

24.5.12.1 Early Works

The Early Works Phase will include all work conducted during the construction of the Mess Creek Access Road, when the site can only be accessed by helicopter. At this time, the exploration camp will be updated to accommodate up to 100 workers for clear cutting, topsoil stripping, and blast area development. This area will later be developed for construction camp, truck shop, processing plant, substation and construction road at plant site. Work for this phase will be completed using small equipment that can be flown in by helicopter. This work will allow the expedited commencement of future work when the Mess Creek Access Road is completed and large equipment can be delivered to site by ground transportation.

24.5.12.2 Crushers Installation

Once the foundations for the primary crusher and the semi-mobile crusher are completed, the retaining wall will be constructed concurrently with the structural backfill and the mine truck ramp. The mine truck ramp will be constructed using mine overburden waste trucked to the crusher area by the mine trucks.

Once the ramp is in place, the superstructure for the overhead crane will be installed and the overhead crane mounted and energized. Once energized, the 110 t capacity overhead crane will be used to install the crusher shell sections, as well as the pre-assembled main shaft. The main shaft assembly is 102 t and is the heaviest section of the crusher; its preassembly will be performed directly under the overhead crane on a purpose-built platform or low-bed trailer, and then lowered down through the crusher pocket at the appropriate time.

24.5.12.3 SAG Mills and Motors Installation

The SAG mills will be outfitted with pinion drives. A rigging review indicated that using a mobile crane (or cranes) from either inside the grinding section of the building, or on the outside of the building, to lift the trunnion sections into position, is not practical or cost-efficient. A mobile crane inside the building would be “boom bound”; a mobile crane outside the building would require a crane of a capacity exceeding 600 t. This option was deemed cost-prohibitive.

The review ultimately recommended using a higher grinding bay and a higher-capacity overhead crane to accommodate the installation of the stator sections. The additional cost to raise the height of the building in the mill section and increase the capacity of the overhead are somewhat offset by the high cost of using mobile cranes and future maintenance issues that may arise during the operation of the mill.

24.5.12.4 Mill Installations Inclusive of Operating Floor Requirements

The ball mills and the SAG mills will be installed using the overhead cranes. The overhead crane capacities have been based on the heaviest section of the mills. There will be one overhead crane situated over the ball mills and SAG mills to aid concurrent assembly of the ball mills and the SAG mills. It has been recommended to install the complete operating floor before the mills are installed. This step will eliminate the need to construct temporary platforms around the mill foundations for the millwrights to perform surveying work and begin assembling the various shell and head sections.

The sections will be brought into the mill building through the oversized, overhead door. The sections will come to rest on the basement floor, and will then be cleaned and prepared for installation.

24.5.12.5 Flotation Tanks

The flotation tanks are too large to transport to the mine site in an assembled state. Therefore, the rolled plates will be field-assembled close to the concentrator building opening, located at the concentrate end of the building. Once assembled, the tanks will be lifted into the building with an overhead crane and installed onto the appropriate foundation inside the building.

The flotation system is also divided in to two phases. Phase No. 1 will be assembled first and ready for commissioning before Phase No. 2 will start, following the same procedure as Phase No. 2.

24.5.12.6 TSF Earthworks and Borrow Pits

The TSF will be constructed in staged lifts throughout the mine life. Construction of the TSF will commence four years prior to mill start-up to ensure sufficient water collection from the spring freshets for use in the mill process.

The engineer's quality specifications will govern the construction of the TSF. Several TSF components cannot be built during the winter and spring melt periods, including:

- foundation preparation activities such as topsoil stripping, embankment drain construction, cut-off trench excavation and backfill (frozen ground considerations)
- placement of the filter and transition zones within the embankments.

Initial construction of the TSF will begin with a starter embankment at the north end of the TSF by a contractor. The starter embankment will be constructed using materials from local borrow sources adjacent to the North Dam. Once the haul road from the open pit to the TSF is complete, embankment raises will be constructed using mine waste rock delivered by the mine fleet to the TSF embankments.

24.5.12.7 Permanent Power

Permanent 287 kV power will be available in Y-3. The power line from BC Hydro's Bob Quinn Substation is scheduled to be completed at the same time as the main substation at the plant site. The main substation will start after the Mess Creek Access Road is completed and the main components (transformers, switch gear, etc.) can be delivered by trucks to site. Temporary power will be required for the early construction period before the permanent power is energized.

24.5.12.8 Mine Pre-Production

The main transformer will become operational in Y-3. Mine pre-stripping will commence in Y-1, with dozer shaping of the pit access roads and mining of the conveyor cut and Phase North 1. Overburden and waste rock will be used to construct the service and mine roads and the crusher pad. Mineralized material will be stockpiled in a temporary site and the long-term stockpile for mill commissioning in Y-1. The initial TSF North Dam will be constructed with material from a borrow pit close to the North Dam. At the end of pre-production, mined overburden and waste rock will be used to construct the TSF North and South Dams.

24.5.13 Project Team Responsibilities

Understanding the relationship and responsibilities of the various groups that comprise the Project Team during the engineering and construction phase is fundamental to the success of the Project.

The organization charts in Figure 24-1 and Figure 24-3 illustrates how the Project will move into detailed engineering and construction. Schaft Creek JV has taken on the responsibility for project management and as such is responsible for directly managing the various groups that contribute to the Project execution. There will be robust links between the engineering and construction management groups who need to be in touch with each other during the execution stages.

25.0 INTERPRETATION AND CONCLUSIONS

25.1 Geology

The Schaft Creek deposit is an example of calc-alkaline porphyry-copper-molybdenum style mineralization. Significant but low concentrations of gold and silver occur with the copper mineralization.

The lithology, structural, and alteration controls on the mineralization in the Schaft Creek deposit is sufficiently understood to support estimation of Mineral Resources.

The Schaft Creek deposit consists of three contiguous zones of porphyry style mineralization: Liard (Main) Paramount, and West Breccia.

The exploration potential of the Schaft Creek project is described in detail in the Technical Report entitled “Mineral Resource Estimate Update for the Schaft Creek Property, British Columbia, Canada”, prepared by Tetra Tech Canada Inc. with an effective date of January 15, 2021. The mineralization in the Schaft Creek deposit is open in several directions and additional drilling is required to test the extension of the mineralization in these directions.

The Schaft Creek Project covers a 12 km long mineralized trend that hosts the Discovery and LaCasse zones located between 1.5 km to 3.0 km north of the Schaft Creek deposit. Limited diamond drilling intersected significant intervals of porphyry style Cu-Mo-Au-Ag mineralization that is open in several directions. Several copper showings have been found north of the Discovery/LaCasse area and south of the Schaft Creek deposit. The exploration potential of the 12 km long trend is considered significant with potential to host additional porphyry style copper mineralization additional exploration is warranted.

25.2 Exploration, Drilling, and Analytical Data Collection in Support of Mineral Resource Estimation

The exploration and drilling programs completed to date are appropriate for the style of the deposit; the sampling methods employed are acceptable for Mineral Resource estimation.

Sample preparation, analysis, and chain of custody procedures were consistent with industry standards at the time the sampling was conducted.

The quantity and quality of the lithological, geotechnical, collar, and down-hole survey data collected during the exploration and delineation drilling programs are sufficient to support Mineral Resource estimation.

The sample data adequately reflects deposit dimensions, styles of mineralization, and alteration, and is representative of the grade variations in the deposits and reflects areas of higher and lower grades.

Silver data from the Teck programs completed in the 1980s is not acceptable to support Mineral Resource estimation.

Many samples were not assayed for gold and silver; these missing values have been estimated using a regression curve from the copper assay data.

The QA/QC programs, when completed, were in accordance with industry-accepted standards and adequately address issues of precision, accuracy, and contamination.

The QA/QC programs did not detect any material sample biases.

The data verification programs support the geological interpretations and constitute a database of sufficient quality to support the use of the data in Mineral Resource estimation.

25.3 Mineral Tenure, Surface Rights, Water Rights, Royalties, and Agreements

Information provided by Copper Fox supports Schaft Creek JV's title to the mineral tenures comprising the Project and is sufficient to support declaration of Mineral Resources.

Surface rights, water rights, royalties, environmental permitting and liabilities, and social responsibility are discussed in detail in the applicable sections of this Report.

The Schaft Creek Project is subject to two separate NPI payments. Royal Gold holds a 3.5% NPI on certain mineral claims within the resource area. Based on the PEA, the NPI payments to Royal Gold are estimated to be USD$258.5 million LOM.

The Schaft Creek JV has an obligation to Liard in the form of a 30% NPI in certain mineral claims within the Schaft Creek Project (the "Indirect Interest"). Liard is owned 85.5% by the Schaft Creek JV, 1.55% by Copper Fox, with the remaining 12.95% held by third parties. Based on the PEA, the LOM NPI payments to Copper Fox stemming from this Indirect Interest is USD$35.8 million and USD$298.9 million to the other Liard minority interests.

Certain mineral claims located outside the mineral resource area of the Schaft Creek Project are subject to a 2% Net Smelter Return Royalty, one-half of which can be purchased for between CAD$1.0 to CAD$1.5 million.

There are no other significant factors and risks known to Copper Fox or the QPs that may affect access, title, or the right or ability to perform work on the Project that are not discussed in this Report.

25.4 Mining

Mining of the Schaft Creek deposit is planned as a conventional truck-shovel open-pit mining operation (drilling, blasting, loading, hauling) to extract the mineralization and waste. Total mine production is estimated to be 1.03 Bt of mill feed grading 0.26% Cu, 0.16 g/t Au, 0.017% Mo, and 1.23 g/t Ag and 1.03 Bt of waste rock resulting in a LOM 1:1 strip ratio, which is low when compared to other mines of similar scale for that proposed for the Schaft Creek Project.

Annual mined tonnage range from 46.9 to 165.0 Mt, averaging 98.7 Mt LOM. The 21-year mine plan utilizes approximately 60% of the mineral resource base and provides options to extend mine life and/or increases in throughput.

Mining operations will commence one year in advance of processing operations in an area of higher-grade mineralization within the Liard Zone. The higher-grade mineralization would be processed in years 1 to 5 of milling operations before transitioning to the south end of the Liard zone. The push back to the north results in increased annual tonnage of waste mined to provide access to mineralization mined in the next phase of the mine plan. The final phase extends to the ultimate pit bottom which based on the resource block model would end in mineralization. The mine plan includes a stockpiling strategy to ensure optimal LOM mill feed grade. ROM mill feed would be delivered to a gyratory crusher at the edge of the pit and transported via conveyor to a coarse mill feed stockpile near the mill site. Mine life is 21 years, and consists of six phases.

Waste material will be used for road, TSF and infrastructure construction with the balance stored in two separate areas located at varying distances from the proposed open pit.

Topographical relief, climate, haul distances, and geographic location present no substantive issues to the Project.

Factors that could impact production are dewatering the pit and slope stability.

25.5 Metallurgical Test Work

Metallurgical test work were appropriate to the mineralization type and to inform a conceptual processing flowsheet.

Samples of the various styles of mineralization collected for testing purposes were selected from a range of depths within the deposit and of sufficient sample mass to perform the testing.

Comminution characteristics indicate the mineralization can be classified as hard with respect to SAG mill and ball mill grinding, but varies distinctly with lithology. The average grindability parameter of A x b values to SAG mill grinding were determined to be 34 for the breccia, intrusive and porphyry mineralization while the volcanic lithology represents a distinctly harder mineralization type with an A x b value of 31. The Bond BWi ranges from 16.6 kWh/t to 22.4 kWh/t.

The current comminution facility was designed to process the mill feed at an average processing rate of 133,000 t/d at 150 µm grind size (at 92% availability) according to competency and hardness values for the four lithologic domains identified within the Schaft Creek deposit indicates.

The copper and molybdenum flotation test results show that the mineral samples tested responded well to the simple and conventional process. Projected metal recoveries and concentrate grades are based on metallurgical test work and are appropriate to the mineralization types and the selected process route.

The mineralization is amenable to production of a 28% copper concentrate with significant gold and silver by-product credits and a 50% molybdenum concentrate.

Multi-element assays on the bulk concentrates generated from the locked cycle tests showed that on average, the impurities in the copper and molybdenum concentrates should not attract smelting penalties as set out by most smelters.

Historical test work indicates the presence of significant concentrations of rhenium in the molybdenum concentrate. Further test work to confirm rhenium concentrations in the molybdenum concentrate is warranted.

Additional metallurgical test work is recommended for a better understanding of the metallurgical performances of various lithological domain mineralization and improving LOM average metallurgical performances.

25.6 Process Plant

The processing plant is designed with a planned nominal throughput of 133,000 tpd (with an availability of 92%). The annual throughput varies from 48.5 Mt to 51.5 Mt per year averaging 49.1 Mt per year, primarily due to the comminution characteristics of the mineralization.

The milling process is a conventional grinding and flotation circuit, consisting of two process trains to produce a high-quality copper concentrate with significant gold and silver by-product credits and a separate molybdenum concentrate. Each of the process trains consists of SABC primary grinding, bulk rougher/scavenger flotation, bulk concentrate regrinding and cleaner flotation circuits. The bulk concentrate produced will be separated to produce market grade copper-gold-silver concentrate and molybdenum concentrate. LOM metal recoveries to copper concentrate containing 28% copper are expected to be 83.1% for copper, 71.0% for gold and 40.3% for silver. The molybdenum recovery to the molybdenum concentrate (containing >50% Molybdenum) is estimated to be 60.1%. Tailings would be transported to the TSF through pipelines.

LOM, the copper concentrate is expected to contain on average 28% copper, 14.1 g/t Au and 63.0 g/t Ag with a moisture content of 9% and the Mo concentrate is expected to average 50% Mo with a moisture content of 5%. The Schaft Creek copper concentrate as modeled is considered a clean copper concentrate, with gold and silver by-products and low deleterious element content. The estimated LOM metal and "dry" Cu and Mo concentrate production is summarized below.

Table 25-1: Concentrate and Metal Production

Description	Unit	Years 2-6[1] Annual Average	Year 1-10 Annual Average	LOM Annual Average	LOM Total
Copper Concentrate	t (000s)	418	393	385	8,091
Copper in Concentrate	Mlbs	258	243	238	4,995
Copper in Concentrate	t (000s)	117	110	108	2,266
Gold in Concentrate	oz. (000s)	233	205	176	3,695
Silver in Concentrate	oz. (000s)	770	721	782	16,413
Molybdenum Concentrate	t	8,248	8,150	9,783	205,439
Molybdenum in Concentrate	lbs (000s)	9,092	8,984	10,784	226,457

Note: (1) Based on first five years of full production. Year 1 is partial year of production and not included. Numbers are rounded.

25.7 Infrastructure

The closest provincial road to the Schaft Creek Project is Highway 37. Power would be provided from the Northwest Transmission Line, located on Highway 37, owned by BC Hydro, the provincial electrical authority. The locations of project facilities and other infrastructure items were selected to take advantage of local topography, accommodate environmental considerations, and for efficient, safe and convenient operation of the mine.

The updated mine plan resulted in a more compact and capital efficient project configuration that reduces haul distances and eliminates one TSF embankment; one waste rock storage facility and the pipeline to deliver fuel to site set out in the 2013 FS.

The required Project Infrastructure includes:

a. Construction of the More Creek Canyon bridge.

b. An access and road use agreement for use of an existing road alignment that goes 65.2 km from Highway 37 to the Mess Creek Valley. This agreement would have to be acceptable to existing road users, the BC Government and the Tahltan for use of a secure, single-lane access road with select double-lane sections.

c. Construction of new road for approximately 40 km up the Mess Creek valley to the project site.

d. An 81 km long, 287 kV power line from BC Hydro's Bob Quinn station and a site power distribution network.

e. TSF with diversion channels including a reclaim water system and process and ancillary facilities.

f. Site haul roads.

g. Water supply and distribution system, sewage disposal plant, communications infrastructure.

h. Upgrade to the Bob Quinn Airport to receive aircraft with a capacity of up to 78 passengers.

The TSF has been designed in accordance with current regulations with a 1.0 Bt capacity, which can be expanded to approximately 2.0 Bt with little modification to the TSF footprint.

The Project is located in a low seismicity area.

Offsite infrastructures includes construction of a concentrate storage facility at Port of Stewart, British Columbia; this is the closest and most efficient route to deliver concentrates to overseas smelters.

25.8 Environmental, Permitting, and Social Considerations

The Schaft Creek project is located within the traditional territory of the Tahltan Nation in northwest BC. Copper Fox is committed to working with its JV partner to develop and operate the Schaft Creek Project in a safe, ethically and socially responsible manner while maximizing benefits and economic opportunities for local Indigenous groups and other communities, including employment, training, and using local service providers.

A review of the current permitting requirements and legislation indicates the Project would be subject to provincial and federal review processes.

Historical project-specific baseline studies completed between 2006 and 2012 included dust, noise, meteorological, groundwater, and surface water monitoring studies; aquatic and fisheries studies; collection of physical, chemical, and biological marine data; sediment quality, wetland, flora, and fauna surveys; species at risk surveys; site metal analysis; archaeological assessments; land use reviews; and cultural and socio-economic studies.

Collection of environmental data has continued as part of a long-term data collection effort for hydrology, climate, and hydrogeology intermittently since 2013.

The review of the current legislation related to the environmental permitting process indicates no major issues and that the Project would be permittable under the current provincial and federal legislation.

In British Columbia, the EAO, manages a clearly defined process set out in the *Environmental Assessment Act* (SBC, 2018) to conduct the assessment of a major project.

The Concurrent Approval Regulation (British Columbia Reg. 371/2002) allows for parallel review of related provincial permit applications.

The Project is subject to federal EA requirements pursuant to the *Impact Assessment Act* (2019). The Physical Activities Regulation (SOR/2019-285) is used to identify thresholds for mineral mine projects that may be subject to a federal assessment process.

Reclamation plans for the TSF, open pit and waste rock are set out in the PEA. These plans will be considered and updated throughout design, construction, and operation of the Schaft Creek Project to help ensure that these objectives can be achieved.

25.8.1 Closure and Reclamation Cost Estimates

Section 10 of the *Mines Act* stipulates that the Chief Inspector of Mines may require projects to provide monetary security with mine reclamation and post-closure commitments.

The reclamation and closure costs for the Project as contemplated in this study are estimated to be USD$154.0 million for reclamation of the TSF, open pit, project infrastructure, access road, and ongoing monitoring.

The Project currently conducts operations pursuant to Notice of Work and Reclamation Permit MX-1-647. A deposited security of $695,000 for Schaft Creek is held under Permit MX-1-647 with the Minister of Finance.

The key risks associated with release of water from the TSF into the environment have been considered, with a design incorporating surface water management control dams and inclusion in the TSF design of a low-permeability basin, cut-off drains, and monitoring.

25.9 Capital and Operating Costs

Compared to the 2013 FS study, initial capital and sustaining cost have been reduced primarily due to changes in mine plan and project infrastructure layout.

The Capex has been prepared in accordance with the Class 5 Cost Estimate standards of the AACE. The overall accuracy range of the estimate is within ±30%.This PEA estimate is prepared with a base date of Q4 2020 and does not include any escalation beyond this date.

Initial capital costs (direct costs, indirect costs, and contingency) are estimated to be USD$2.65 billion. The LOM operating cost is estimated to be USD$8.66/t processed.

Sustaining capital required over the 21-year mine life is estimated to be USD$849 million, including USD$154 million of closure cost.

Key changes in Capital and Operating costs in the 2021 PEA compared to the 2013 FS are:

- LOM average operating cost per tonne processed reduced from USD$13.25/t to USD$8.66/t.
- Initial capital costs reduced from USD$3.26 billion to USD$2.65 billion.
- Sustaining Capital Costs reduced from USD$1.20 billion to USD$848.7 million.

25.10 Economics

The PEA, Pre-Tax and Post-Tax project economic analysis of the Schaft Creek Project is based on payable metal and was prepared on a 100% basis using revenues and costs projected into the future on an annual basis and then discounted using mid year discounting at a rate of 8% per annum to yield the NPV and IRR.

In Summary:

- Pre-Tax NPV_8 of USD$1.383 billion and IRR of 15.2%
- Post-Tax NPV_8 of USD$842.1 million and IRR of 12.9%, with a 4.8 year payback period
- Average annual EBITDA(6) of USD$695.4 million based on first 5 years(1) (Years 2-6) at full production, and USD$10.8 billion LOM
- Average annual FCF before recovery of capital costs of USD$633.4 million based on first 5 years(1) (Years 2-6) at full production and USD$9.96 billion LOM
- NSR of USD$20.63 per t
- 21-year LOM producing approximately 5.0 billion lbs or 2.3 million M copper, 3.7 million oz gold, 226.0 million lbs molybdenum, and 16.4 million oz silver in concentrate
- 133,000 tpd LOM nominal milling rate with an availability of 92% processing 1.030 Bt of mill feed LOM, representing approximately 60% of identified mineral resources
- Estimated Initial Capital Costs of USD$2.653 billion, not including Sustaining Capital Costs of USD$848.7 million which is inclusive of USD$154.0 million Closure Costs. Operating Costs are estimated to be USD$8.66/t processed
- C1 Cost(7) (net of by-product credits); for first 5 years(1) (Years 2-6) at full production of USD$0.46 per lb of payable copper and USD$1.00 per lb payable copper LOM
- All in Sustaining Costs(7) for first 5 years(1) (Years 2-6) at full production of USD$0.72 per lb payable copper and USD$1.18 per lb payable copper LOM

Notes:

1. Years 2-6 are first five years of full production, excluding the first partial year of operations. The first 10 years includes the first partial year of operations
2. Annual Average for years 2-6 excludes the first partial year of operations. The first 10 years includes the first partial year of operations
3. Copper equivalent numbers are calculated by converting gold, molybdenum and silver production into copper equivalent lbs. using base case metal prices
4. Cash Costs before by-product credits allocate all costs, except for specific gold and silver refining costs and molybdenum concentrate freight costs and roasting charges to the payable copper produced; Cash Costs after by-product credits deduct the revenue received from gold and silver in copper concentrate and molybdenum concentrate sales net of specific gold and silver refining charges and molybdenum concentrate freight; Cash Costs are inclusive of all costs during operations
5. Payback is the number of years from first production that Initial Capital payback is achieved
6. EBITDA is a financial term showing earnings before deduction of interest, taxes, depreciation, and amortization Project EBITDA is net of NPI payments and BC Mineral Tax
7. C1 Cost and All in Sustaining Costs, ("AISC") are non-GAAP financial measures which does not have a standardized meaning prescribed by International Financial Reporting Standards (IFRS). These measures are meant to provide further

information to investors and should not be considered in isolation or used as a substitute for other measures of performance prepared in accordance with IFRS

This Project post-tax NPV is most sensitive to the copper price and USD/CAD exchange rate. The gold price, operating costs, and capital costs may also impact the Project's economics to a lesser degree.

The estimated Federal, Provincial, and BC Mineral Tax payables based on the PEA financial model are calculated and shown in Table 25-2.

Table 25-2: Estimated Taxes Payable

Tax Component	LOM Amount (CAD$M)
Corporate Tax (Federal)	1,432
Corporate Tax (Provincial)	1,145
BC Mineral Tax	1,198
Total Taxes	3,775

25.11 Mineral Resource Estimates

The classification of Measured, Indicated, and Inferred Resources conforms to Canadian Institute of Mining, Metallurgy, and Petroleum Definition Standards for Mineral Resources and Mineral Reserves dated January 15, 2021 (CIM Definition Standards).

Eighty percent (80%) of the resource estimate for the Schaft Creek deposit is in the Measured and Indicated resource categories, and 20% of the resource is classified in the Inferred category.

The current Mineral Resource estimate represents the minimum resource within the Schaft Creek deposit. There is potential to increase the resource base by additional drilling to expand the limits of the mineralization and upgrade Inferred resources to higher-confidence Mineral Resource categories.

Under the assumptions presented in this Report, and based on the available data, Mineral Resources show reasonable prospects of eventual economic extraction.

Michael O'Brien, P.Geo, believe the Mineral Resources were estimated in an appropriate manner using current mining software and procedures consistent with reasonable practices.

Mr. O'Brien is not aware of any mining, metallurgical, infrastructure, permitting, or other relevant factors that would materially affect the Mineral Resource estimates.

Factors that could affect the estimates include changes to long-term metal price assumptions; changes in local interpretations of mineralization geometry, fault geometry, and continuity of mineralized zones; changes to the NSR used to constrain the estimates; changes to the regression equation used to fill in missing gold and silver values; changes to metallurgical recovery assumptions; changes to the input assumptions used to derive the conceptual open pit outlines used to constrain the estimate; variations in geotechnical, hydrogeological, and mining assumptions; and changes to environmental, permitting, and social responsibility assumptions.

26.0 RECOMMENDATIONS

In this section, selected risks, opportunities, and recommendations of the Schaft Creek project are presented. Overall, it is recommended that the Project to be advanced to the pre-feasibility stage.

The recommended program is designed to allow the project to advance to the pre-feasibility stage. The location of the proposed drill holes and specific parameters for each program would be established by the Schaft Creek JV.

26.1 Geology

The recommended geological drilling is estimated to be 7,300 m. With a unit cost of CAD$620 per m all-included, the total cost is estimated at CAD $4.5 million. These proposed drill holes would increase the quality of the geological model and resource confidence, investigate potential extension to the mineralization, and provide additional information on the mineralogy and metal grades in these areas of the Schaft Creek deposit. As well these drill holes would be used to collect additional information to investigate opportunities to increase pit slope angles with the objective of reducing the LOM strip ratio.

The geotechnical drilling is designed to investigate the bedrock conditions in the proposed locations of the embankments for the TSF in the Skeeter Lake and Start Lake areas, the mill site and other infrastructure components of the Project. An estimated 8,000 m of drilling would be required to investigate these components of the Project. With a unit cost of CAD $650 per m all included, the total cost is estimated at CAD $5.2 million.

26.2 Mining

Table 26-1: Mining Risks, Opportunities, and Recommendations

Opportunities	Recommendations
Eliminate the in-pit crusher and reduce the stationary crusher size	▪ Blast modeling to investigate the particle size reduction trade-off
Eliminate West Dump by relocating the waste storage to Start Lake and East Dump	▪ Waste storage re-design
Autonomous haul trucks	▪ Conduct a haulage traffic study to confirm viability
To further optimize the current mine plan	▪ Subject matter expert review of the mine plan ▪ Optimize the head grade profile on the LOM mine production schedule, which shows low grades during pit phase in year 11 than other years in LOM. ▪ Possibility of drilling to locate higher grade material
RSF footprint has been reduced in 2021 PEA compared to 2013 FS with the elimination of south RSF. The south RSF still represents a suitable location for any future expansion beyond the current 21 years LOM	

26.3 Metallurgy and Process

Tetra Tech recommends further metallurgical test work focusing on the four geometallurgical domains within the Schaft Creek deposit to further investigate the geometallurgical variability and throughput assumptions for each geometallurgical domain. The parameters of the testwork should be designed to optimize process conditions and update metallurgical performances. The testwork should include the following:

- The sample selection for the proposed testwork should include consideration for expected average LOM head grades as well as expected average head grades for the first five-year and ten-year mine plans. The testwork should include a range of expected head grades within each geometallurgical domain according to the mine plan.
- Additional metallurgical response evaluations should be conducted to further investigate mineral liberation, copper mineral species and metallurgical response for each geometallurgical domain to update the process flowsheet.
- Bench scale test work to further investigate the metallurgical performance of samples representing geometallurgical domains and expected mill feeds from each domain in the initial five- and 10-year mill feeds, based on the updated mine plan.
- The testwork should focus on obtaining additional information on comminution parameters for each geometallurgical domain to determine if a more energy efficient, higher throughput circuit design is possible.
- Mineralogical investigation from previous testwork indicates that rhenium occurs as discrete particles associated with molybdenite. The testwork should include further copper-molybdenum separation tests to assess the potential additional value of rhenium in the molybdenum concentrate.
- A pilot plant campaign is recommended to verify the bench test results and generate samples for molybdenum separation from copper-molybdenum bulk concentrate and for smelting performance assessments. In addition, this testing will generate samples for concentrate dewatering tests and slurry property determination tests.

The cost of the test work is estimated to be approximately CAD$2,700,000. The costs associated with the drill core sample generation, including drilling and sample shipping, are excluded in the cost estimate, but included in Table 26-8.

Further mill optimization should be conducted during future project assessments. The engineering costs associated with the mill optimization and plant site geotechnical condition assessments are part of the future studies and shown in Table 26-8.

Table 26-4 summarizes potential risk, opportunities and recommendations for the project metallurgy and process.

Table 26-2: Metallurgy and Process Risks, Opportunities, and Recommendations

Risks	Mitigation Measures
Primary grinding circuit may not be able to achieve the designed throughput	▪ Further grindability test work to confirm hardness and work indexes ▪ Further simulation works to evaluate hardness and work indexes ▪ Modify design criteria and equipment selection
Variation in metallurgical performances	▪ Further test work is suggested for better understanding the metallurgical performances, particularly, the Paramount mineralization and the West Breccia mineralization
Opportunities	**Recommendations**
Overall metal recoveries – a coarser primary grind size can reduce Opex	▪ Further test work to verify the metal recovery vs. primary grind size
Eliminate the in-pit crusher and reduce the stationary crusher size	▪ Metallurgical test work and mineralogical evaluations confirm if ROM particle size can be reduced to target size without primary crushing
A higher Cu concentrate grade	▪ Further test program to optimize reagent regime and regrind size
Cleaning of Cu/Mo concentrate to improve Mo recovery	
Potential additional value of rhenium in the molybdenum concentrate	▪ Resource evaluation, metallurgical testing, and marketing study
Quad drive for SAG mills	▪ Investigate this option and conduct a trade-off study comparing quad drive and wrap around drive
Fluidized-bed (hydro-float) flotation	▪ A new test work program to determine viability
HPGR grinding	
Mill feed sorting	

26.4 Tailings, Water Management, and Environmental

Potential risks associated with tailings and water management for the Project are summarized in Table 26-3 below. Where relevant, potential mitigation measures have been identified for each associated risk factor, and an overall risk rating assigned.

Table 26-3: Tailings, Water Management and Environmental Risks and Mitigation Measures

Risks	Mitigation Measures	Overall Risk
Risk of avalanche or other geo-hazards to the TSF	▪ Extreme Dam Classification ▪ Sufficient freeboard to prevent wave run-up from inundation ▪ Sloped cover at closure to deflect or dissipate energy from avalanche/landslide	Low
Risk of failure in foundation (poor embankment foundation conditions)	▪ Extreme Dam Classification ▪ Large buttresses on embankments to create flatter embankment slopes ▪ Use of waste rock for added stability ▪ Larger starter embankments may be required (flatter slopes)	Low
Risk that change in tailings process impacts ability to use cyclone sand as construction material	▪ Use of waste rock for embankment construction	Low
Tailings geochemistry characterization incorrect (i.e., tailings are potential acid generating [PAG]). Cyclone sand no longer an option. Seepage water quality a concern.	▪ Use of waste rock for embankment construction ▪ Embankment filters designed for seepage management ▪ Water treatment implemented to treat seepage ▪ Tailings maintained saturated ▪ Tailings closure cover designed to maintain saturation and to limit oxidation of tailings	Low
Waste rock geochemical characterization incorrect (more PAG WR than anticipated)	▪ Prioritize NPAG material for construction ▪ Water management plan adjusted to collect runoff from PAG WR ▪ Potential to co-mingle PAG WR with tailings	Low
Climate/water balance assumptions not conservative enough (more annual surplus than estimated)	▪ Surplus water management system designed with appropriate contingency ▪ Prioritize discharge in high-flow conditions ▪ Allow for contingency storage in TSF for seasonal fluctuations in allowable discharge and inflows ▪ Current design allows for storage of 2 x PMF event above maximum operating pond volume	Low
TSF Supernatant water quality prediction not conservative enough	▪ Water treatment system implemented to treat prior to discharge to meet water quality guidelines	Low

table continues…

Risks	Mitigation Measures	Overall Risk
Tailings dam breach at South TSF embankment may inundate Plant Site and other Site Infrastructure	▪ Extreme dam classification and designed appropriately	Low
Climate Impacts to Schedule – Inclement weather affects construction schedule	▪ Prioritize construction using materials less affected by adverse weather conditions	Low
South WSF located in Schaft Creek floodplain. Challenging to capture runoff or collect intercepting water in its location; thus needs to be clean rock.	▪ Optimize layout to create more of a buffer between Schaft Creek and toe of WSF ▪ Potential to combine this WSF with South TSF Embankment (improves stability of TSF embankment while avoiding impacts to Schaft Creek)	Low
Schedule 2 amendment under MDMER required for impacts to Start Creek watershed and Start Lake	▪ Allow for Schedule 2 amendment application process to be built into project schedule and permitting timeline	Low
High value wildlife habitat (i.e., salt licks, moose habitat) in proximity to infrastructure and increased access to site	▪ Further baseline assessments and implementation of wildlife mitigation measures	Low
Project is within a traditionally highly used area close to a significant source of obsidian potential for Traditional Use / Archaeological concerns	▪ Continue engagement and dialog with Tahltan Nation ▪ Implement archaeological program	Low
Potential impact on water quality due to sediment runoffs from construction activities	▪ Establish buffer zone between project infrastructure and creek to allow for water management and to ensure downstream receiving environment is not impacted	Low
Flows in Schaft Creek are driven by glacial runoff and melt. Glacial retreat could see reduced flows.	▪ Opportunity for melting glacier to allow for controlled water flow ▪ Prioritize water discharge to Schaft Creek to maintain existing flow regime	Low

The PEA Study has highlighted several opportunities to increase Project economics, as well as to reduce the previously identified risks. The opportunities for tailings and water management, and environmental studies, are summarized in Table 26-4. The opportunity to construct the TSF embankments with waste rock is discussed below.

Table 26-4: Tailings, Water Management, and Environmental Opportunities and Recommendations

Opportunities	Recommendations
Optimize use of mine waste materials (combining waste rock and tailings) for construction materials.	▪ Complete drilling and site investigation programs to characterize foundation conditions for the TSF and other site infrastructure. ▪ Complete additional test work on construction materials and tailings to characterize geotechnical and geochemical properties, and confirm assumptions for cyclone sand. ▪ Advance designs to FS level to incorporate results of test work and site investigations.
Optimize construction sequencing to reduce initial capital costs and handling of materials.	
Foundation conditions (particularly TSF south embankment) may not be as adverse as anticipated.	
Limit impacts to fish habitat and/or avoid Sch.2 amendment through TSF optimization.	▪ Continue baseline environmental data collection programs at the critical times of the year (summer and winter low flows, freshet, and fall rain events). ▪ Complete predictive models to determine if water treatment is required for the Project and/or to complete designs of water treatment systems. ▪ Explore opportunities to combine/integrate with nearby projects. ▪ Continue to monitor evolving mining, mine waste disposal, and water treatment technologies for opportunities to optimize the Project.
Potential that water quality predictions may be better than anticipated (no treatment required prior to discharge).	
Potential that water balance / climate predictions are conservative resulting in smaller annual surplus and reduced project discharge.	
Opportunity to integrate Project with nearby projects (i.e., Galore Creek).	
Potential for advancements in tailings technologies, water treatment technologies, waste management, mineralized materials processing technologies, etc., which may result in reduced capital costs and/or reduced environmental impacts.	
Opportunity to support and establish real business capacity with the Tahltan Nation for services to the mine project (e.g., third party environmental monitoring).	
TSF embankments have been moved south since the 2013 FS. The 2021 PEA arrangement allows for future expansion of the Project beyond the current 21 year LOM for up to 2 Bt without changing the footprint.	

table continues…

26.4.1 TSF Embankment Construction Alternative

An alternative concept for construction of the TSF has also been considered for the Project that considers the use of pit-run waste rock material to construct the TSF embankments rather than cyclone sand. In this scenario, all tailings will be deposited in the TSF in a single tailings stream.

Material from local borrow will still be required to construct the North TSF Starter embankment. The waste rock fill embankments would be constructed with 2.25H:1V downstream slopes, with a toe buttress constructed to provide an overall downstream slope of 3H:1V.

The TSF embankments would be raised in stages, rather than annually, as is the current proposed concept for cyclone sand tailings. The Stage 1 north embankment crest elevation was selected to provide storage for the first two years of operation; ongoing raises would be carried out at Years 2, 7, 11, and 16. The south embankment would be constructed starting in Year 2 and raised throughout the mine life to meet tailings and water storage requirements.

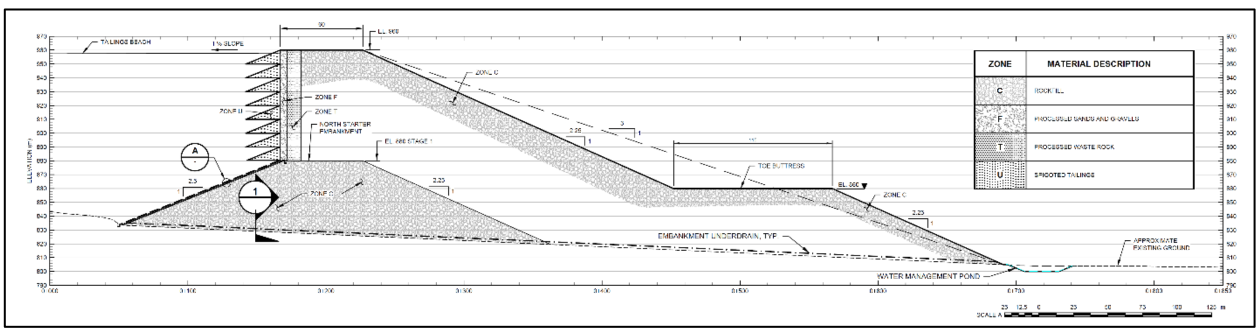
The TSF construction staging for the TSF alternative is summarized in Table 26-5 below.

Table 26-5: TSF Alternative Construction Stages

Stage	Crest Elevation (masl)	Embankment Construction
1	880	North
2	908	North, South
3	925	North, South
4	944	North, South
5	960	North, South

A typical cross-section for this proposed TSF embankment concept is provided on Figure 26-1 below.

Figure 26-1: TSF Embankment Cross-Section (Waste Rock Embankment Fill) (KP, 2020)



Source: KP, 2020

A preliminary comparative cost estimate has been prepared for the waste rock embankment configuration. The comparative cost estimate is shown in Table 26-8 and compares the cost of the TSF alternative with the base case that is presented in Section 18. The costs include a 35% contingency.

Table 26-6: TSF Alternative Comparative Cost Estimate

	Capex	Sustaining Capital	Opex	Total Cost
TSF Base Case (Cycloned Sand)	$204.1M	$312.2M	$52.5M	**$568.8M**
TSF Alternative (Waste Rock)	$191.5M	$422.6M	$16.1M	**$630.2M**
Cost Difference Δ	-$12.6M	$110.4M	-$36.4M	**$61.4M**

26.5 Infrastructure

Table 26-7: Infrastructure Risks, Opportunities, and Recommendations

Risks	Mitigation Measures	Overall Risk
Access Road – avalanche	▪ Install fence, berms or shelter in high risk areas. Active avalanche monitoring and control.	Low
Access Road – mud slide	▪ Apply appropriate geotechnical measures in high risk areas. Active monitoring.	Low
Access Road – long downhill gradients	▪ Soft berms on sides. Runaway lanes.	Low
Access Road – Soft ground	▪ Adequate survey for sections near water bodies. Avoid swamps.	Low
Access Road – flooding/culverts	▪ Consider a more frequent event than 1-in-100 year due to effect of climate change.	Low
Access Road – traffic control	▪ Make pull-outs. Improve gradient/curve radius to enhance traffic safety.	Low
Access Road – stream crossings	▪ The 1-in-100-year flood design in FS should be verified due to climate change.	Low
More Canyon bridge – schedule delay	▪ Robust construction scheduling and on-time delivery of equipment and consumables.	Low
More Canyon bridge – cost overrun	▪ Detail cost estimate and analysis. Increase contingency/reserve.	Low
More Canyon bridge – wind	▪ Robust design, detail analysis on structure and harmonics. Scale model testing in wind tunnel.	Low
More Canyon bridge – labour productivity	▪ Early off-site worker training by computer simulations or a scale model.	Low
More Canyon bridge – design basis	▪ The 1-in-100-year flood design in FS should be verified due to climate change.	Low
Site development – unable to obtain permit for site clearing	▪ Early engagement of all stakeholders and governmental agencies.	Low
Site development – adverse geotechnical conditions	▪ Sufficient site investigation and geotechnical evaluation.	Low
Site development – schedule delay due to weather	▪ Robust construction scheduling and on time delivery of equipment and consumables.	Low
Site development – logistics of equipment	▪ Robust planning. Use heli-lift where appropriate.	Low
Site development – surface water runoffs	▪ Build trenches and collection ditches prior to site activities. Portable gensets and pumps.	Low
Site development – inadequate drainage	▪ Consider a more frequent event than 1-in-100 year due to effect of climate change.	Low

table continues…

Risks	Mitigation Measures	Overall Risk
Site security – unauthorized traffic on site and Mess Creek access road	▪ Install check gates and surveillance at highway 37 junction and at site.	Low
BQLA – aircraft navigation/landing guide	▪ Install GPS navigation and landing instruments.	Low
Diesel tank / pipe leakage	▪ Good quality welding joints. Leak monitor devices with wireless transmit.	Low
Diesel tank / pipe leakage	▪ Sufficient containment. Berms around fuel tank farm. Geotextile liners.	Low
Seismic hazard	▪ Adequate structural engineering and foundation design.	Low
Port	▪ Environmental spill.	Low
Port	▪ Insufficient storage capacity for concentrate.	Low
Lack of current environmental baseline data (due to damaged equipment). Can potentially delay the project schedule by at least one year.	▪ Repair equipment and resume data collection progress.	Medium
Opportunities	**Recommendations**	
Forest harvesting prior to site clearing. Forestry service road to site funded/built by gov't or logging companies.	▪ Applications of risk mitigation measures as listed above. ▪ Contact government to third parties to seek implementation of the opportunities listed above. ▪ Early planning and initiates to seek re infrastructure funding or financing from gov't or third parties. ▪ Early access road, site pre-development and construction planning. ▪ Contact BC Hydro regarding opportunities in capital funding.	
Power transmission line funded by BC Hydro or financed by a utility contractor.		
Early power transmission line energization to reduce diesel power generation costs during construction.		
Alternative construction techniques should be evaluated, such as modularization of buildings and equipment.		
Infrastructure components, such as access road, power transmission line, substation, WTP, camp, fuel storage, crew transportation, can be outsourced to third party.		
On-site batteries or capacitors to moderate peak power demand charges.		
Optimization of construction schedule. Early mobilization and modularization to shorten construction schedule.		

table continues…

Risks	Mitigation Measures	Overall Risk
Optimizing the cash flow during project implementation to potentially improve NPV.		
Opportunities	**Recommendations**	
Pre-production staff training and process circuit tuning to shorten and optimize ramp-up.		
Financing, leasing, design-built, design-built-operate options to defer capital costs.		
Potential funding or low interest loans from gov't to support construction.		
Potential revenue stream from Schaft Creek Access Road toll charges collected from other road users.		

26.5.1 Geological and Geotechnical Drilling

It is imperative that the Stage-Gate process be used to take the planning and budgeting forward, for all disciplines involved. Most of the drill holes can serve multiple interests from each area of the budget. It is expected that this drilling will enable the project to go into a full feasibility study once the pre-feasibility is concluded.

The geological drilling for Liard area is estimated 7,300 m. With a unit cost of $620 per m all-included, the total cost is estimated at $4.5 million.

The geotechnical drilling for Skeeter Lake and Starter Lake areas and the new mill site is estimated 8,000 m. With a unit cost of $650 per m all-included, the total cost is estimated at $5.2 million.

26.5.2 Process, Infrastructure, and Environmental

The PEA describes a recommended work program for the Schaft Creek Project that contemplates a CAD$23.2M budget as part of a potential PFS. Activities include geological and geotechnical drilling, metallurgical testwork, and additional environmental and infrastructure studies to complete the PFS. The recommended budget includes contingencies, preparation of the PFS and direct costs related to completion of the recommended program.

Table 26-8 presents the estimated cost to implement PFS supporting works.

Table 26-8: Summary of the Estimated Cost to Implement Suggested Recommendations

Discipline	Cost (CAD$ million)
Geological Drilling	4.5
Geotechnical Drilling	5.2
Mining	0.2
Metallurgical Work	2.7
Pre-Feasibility	2.5
Environmental & Infrastructure	2.5
Subtotal 1	17.6
Contingency (20%)	3.5
Subtotal 2	21.1
Direct (10%)	2.1
Grand Total	23.2

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28.0 QP CERTIFICATES

I, Hassan Ghaffari, P.Eng., M.A.Sc., do hereby certify that:

- I am a Director of Metallurgy with Tetra Tech Canada Inc. with a business address at Suite 1000, 10th Floor, 885 Dunsmuir Street, Vancouver, BC, V6C 1N5.
- This certificate applies to the technical report entitled “Schaft Creek Preliminary Economic Assessment, NI 43-101 Technical Report”, with an effective date of September 10, 2021 (the “Technical Report”).
- I am a graduate of the University of Tehran (M.A.Sc., Mining Engineering, 1990) and the University of British Columbia (M.A.Sc., Mineral Process Engineering, 2004).
- I am a member in good standing of the Engineers and Geoscientists British Columbia (#30408).
- My relevant experience includes 30 years of experience in mining and mineral processing plant operation, engineering, project studies and management of various types of mineral processing, including hydrometallurgical processing for porphyry mineral deposits.
- I am a “Qualified Person” for the purposes of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for those sections of the Technical Report that I am responsible for preparing.
- I conducted a personal inspection of the Schaft Creek property on September 22, 2010 and inspected the overall project site, including site access roads.
- I am responsible for Sections 1.1, 1.6 (except access roads and TSF), 1.7, 1.8, 1.11, 2.0, 3.0, 18.0 (except access roads, TSF and power supply), 20.0 (except TSF, waste rock and water management), 21.1 (except access roads and TSF), 21.2.2, 24.0, 25.7 (except TSF and water management), 25.8 (except 25.8.1), 25.9, 26.4 (environmental), 26.5 and 27.0 (only references from sections for which I am responsible).
- I am independent of Copper Fox Metals Inc. as Independence is defined by Section 1.5 of NI 43-101.
- I have had involvement with the Schaft Creek property that is the subject of the Technical Report, in acting as a Qualified Person for the “Feasibility Study on the Schaft Creek Project, BC, Canada (the “Technical Report”) with an effective date of January 23, 2013 and “Mineral Resource Estimate Update for the Schaft Creek Property, British Columbia, Canada”, with an effective date of January 15, 2021.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 4th day of November, 2021

"signed and stamped"

Hassan Ghaffari, P.Eng., M.A.Sc.
Director of Metallurgy
Tetra Tech Canada Inc.

I, Jianhui (John) Huang, Ph.D., P.Eng., do hereby certify that:

- I am a Senior Metallurgist with Tetra Tech Canada Inc. with a business address at Suite 1000, 10th Floor, 885 Dunsmuir Street, Vancouver, British Columbia, V6C 1N5.
- This certificate applies to the technical report entitled "Schaft Creek Preliminary Economic Assessment, NI 43-101 Technical Report", with an effective date of September 10, 2021 (the "Technical Report").
- I am a graduate of North-East University, China (B.Eng., 1982), Beijing General Research Institute for Non-ferrous Metals, China (M.Eng., 1988), and Birmingham University, United Kingdom (Ph.D., 2000).
- I am a member in good standing of the Engineers and Geoscientists British Columbia (#30898).
- My relevant experience includes over 35 years involvement in mineral processing for base metal ores, gold and silver ores, and rare metal ores, and mineral processing plant operation and engineering including hydrometallurgical mineral processing for porphyry mineral deposits.
- I am a "Qualified Person" for purposes of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for those sections of the Technical Report that I am responsible for preparing.
- I conducted a personal inspection of the Schaft Creek property on August 9, 2010 and reviewed drill cores and overall project site, including the potential processing plant site.
- I am responsible for Sections 1.4, 1.5, 1.9, 1.10, 13.0, 17.0, 19.0, 21.2 (except 21.2.2 and 21.2.6), 22.0, 25.5, 25.6, 25.10, 26.3 and 27.0 (only references from sections for which I am responsible).
- I am independent of Copper Fox Metals Inc. as Independence is defined by Section 1.5 of NI 43-101.
- I have had involvement with the Schaft Creek property that is the subject of the Technical Report, in acting as a Qualified Person for the "Feasibility Study on the Schaft Creek Project, BC, Canada" (the "Technical Report") with an effective date of January 23, 2013 and "Mineral Resource Estimate Update for the Schaft Creek Property, British Columbia, Canada", with an effective date of January 15, 2021.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 4th day of November, 2021

"signed and stamped"

Jianhui (John) Huang, Ph.D., P.Eng.
Senior Metallurgist
Tetra Tech Canada Inc.

I, Michael F. O'Brien, P.Geo., do hereby certify that:

- I am an independent consultant and director of Red Pennant Communications Corp. a British Columbia Corporation, with a business address at 81-1380 Pinetree Way, Coquitlam, BC, V3E 3S6.
- This certificate applies to the technical report entitled "Schaft Creek Preliminary Economic Assessment, NI 43-101 Technical Report", with an effective date of September 10, 2021 (the "Technical Report").
- I am a graduate of the University of Natal, (B.Sc. Hons. Geology, 1978) and the University of the Witwatersrand (M.Sc. Engineering, 2002).
- I am a member in good standing of Engineers and Geoscientists British Columbia (#41338).
- I am a member in good standing of the South African Council for Natural Scientific Professions (South Africa, 400295/87). My relevant experience is 36 years of experience in operations, mineral project assessment and I have the experience relevant to Mineral Resource estimation of metal deposits. I have estimated Mineral Resources for greenstone-hosted gold, diatreme complex epithermal gold deposits, porphyry copper-gold, volcanogenic massive sulphide deposits and shear zone-hosted deposits. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- My recent personal inspection of the Property was on October 30, 2020 and reviewed drill cores and the general layout of the camp and topography.
- I am responsible for Sections 1.2, 4.0, 5.0, 6.0, 7.0, 8.0, 9.0, 10.0, 11.0, 12.0, 14.0, 23.0, 25.1, 25.2, 25.3, 25.11, 26.1 and 27.0 (only references from sections for which I am responsible).
- I am independent of Copper Fox Metals Inc. as Independence is defined by Section 1.5 of NI 43-101.
- I have had involvement with the Schaft Creek property that is the subject of the Technical Report, in acting as a Qualified Person for the "Mineral Resource Estimate Update for the Schaft Creek Property, British Columbia, Canada", with an effective date of January 15, 2021.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 4th day of November, 2021

"signed and stamped"

Michael F. O'Brien, P.Geo.
Owner
Red Pennant Geoscience Ltd.

I, Sabry Abdel Hafez, PhD, P.Eng., do hereby certify that:

- I am a senior mining engineer with Tetra Tech Canada Inc. with a business address at Suite 1000, 10th Floor, 885 Dunsmuir Street, Vancouver, British Columbia, V6C 1N5.
- This certificate applies to the technical report entitled “Schaft Creek Preliminary Economic Assessment, NI 43-101 Technical Report”, with an effective date of September 10, 2021 (the “Technical Report”).
- I am a graduate of Assiut University (B.Sc. Mining Engineering, 1991; M.Sc. in Mining Engineering, 1996; Ph.D. in Mineral Economics, 2000).
- I am a member in good standing of the Engineers and Geoscientists British Columbia (#34975).
- My relevant experience includes 25 years of experience in the evaluation of mining projects, advanced financial analysis, and mine planning and optimization. I have been involved in the technical studies of several base metals, gold, silver, and aggregate mining projects in Canada and abroad.
- I am a “Qualified Person” for purposes of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for those sections of the Technical Report that I am responsible for preparing.
- I am responsible for Sections 1.3, 15.0, 16.0, 25.4, 26.2 and 27.0 (only references from sections for which I am responsible).
- I am independent of Copper Fox Metals Inc. as Independence is defined by Section 1.5 of NI 43-101.
- I have had involvement with the Schaft Creek property that is the subject of the Technical Report, in acting as a Qualified Person for the “Feasibility Study on the Schaft Creek Project, BC, Canada” (the “Technical Report”) with an effective date of January 23, 2013.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 4th day of November, 2021

“signed and stamped”

Sabry Abdel Hafez, Ph.D., P.Eng.
Senior Mining Engineer
Tetra Tech Canada Inc.

I, B.G. Masson of Calgary, Alberta, do hereby certify that:

- I am a Senior Project Engineer with McElhanney Ltd. with a business address at 100, 402 – 11th Ave SE Calgary AB T2G 0Y4.
- This certificate applies to the technical report entitled "Schaft Creek Preliminary Economic Assessment, NI 43-101 Technical Report", with an effective date of September 10, 2021 (the "Technical Report").
- I am a graduate of the University of New Brunswick, (B.Sc., Civil Engineering, 2007.).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#36610).
- My relevant experience is 16 years of location, survey, design, and construction of roads in the Forestry, Mining, and Oil & Gas sectors.
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was on Dec, 10, 2010.
- I am responsible for Sections 1.6.1, 18.4, 21.0 (access roads), 25.0 (access roads), 26.0 (access roads) and 27.0 (only references from sections for which I am responsible) of the Technical Report.
- I am independent of Copper Fox Metals Inc. as Independence is defined by Section 1.5 of NI 43-101.
- I have no prior involvement with the Schaft Creek property that is the subject of the Technical Report.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 4th day of November, 2021 at Vancouver, British Columbia

"signed and stamped"

B.G. Masson, P.Eng.
Project Engineer
McElhanney Ltd.

CERTIFICATE OF QUALIFIED PERSON

I, Daniel Friedman, P. Eng. do hereby certify that:

1. This certificate applies to the technical report entitled, "Schaft Creek Preliminary Economic Assessment, NI 43-101 Technical Report", with an effective date of September 10, 2021 (the "Technical Report").
2. I am employed as a Specialist Civil Engineer of Knight Piésold Ltd. with an office at Suite 1400 – 750 West Pender Street, Vancouver, British Columbia, V6C 2T8, Canada.
3. I am a graduate of McGill University, Montreal, Canada, B.Eng. (Civil), 2003. I have practiced my profession continuously since 2004. My principal experience is in the areas of water and waste management for mining projects and hydrotechnical engineering.
4. I am a registered professional engineer in good standing in the following jurisdictions:
   - Yukon, Canada (No. 3404)
   - British Columbia, Canada (No. 32571)
   - New Brunswick, Canada (No. L5001)
   - Arizona, USA (No. 53722)
5. 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 purposes of NI 43-101.
6. I visited the Schaft Creek property from July 28 to August 11, 2008 and conducted an overview of the proposed general project and TSF site.
7. I am a contributing author of the Technical Report and am responsible for 1.6.3, 18.8, 18.10.1, 20.4.2, 20.4.3.1, 20.5, 21.1 (Capex related to TSF, power supply, closure and reclamation), 21.2.6, 25.7 (TSF), 25.8.1, 26.4 (TSF and water management) and 27.0 (only references from sections for which I am responsible) of the Technical Report.
8. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
9. I have prior involvement with the property that is the subject of the Technical Report. I have developed various engineering studies related to tailings and water management for Copper Fox Metals Inc. from 2008 through 2020.
10. I have read National Instrument 43-101 and the sections of the Technical Report referenced in this Certificate and confirm that these sections have been prepared in accordance with National Instrument 43-101.

11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I have accepted responsibility contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

Effective Date: September 10, 2021
Signing Date: November 4, 2021

"signed and stamped"

Daniel Friedman, P. Eng.
Specialist Civil Engineer
Knight Piésold Ltd.